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PREFACE

This volume contains papers on miscellaneous subjects, presented at the New York Meeting in February, 1917, but omitted from Vol. LVI for lack of space. All of these appeared in *Bulletins* 119 to 122 inclusive.

It also contains a report of the St. Louis Meeting, October, 1917, and the papers presented at that meeting except those relating to Iron and Steel, Ore Deposits, and Miscellaneous Subjects, which have been reserved for Vol. LVIII. These omitted papers were scattered through *Bulletins* 123 to 130, inclusive; hence this volume supersedes completely only *Bulletins* up to No. 122, February, 1917, inclusive.

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PROCEEDINGS OF THE ONE HUNDRED FIFTEENTH MEETING, ST. LOUIS-JOPLIN-MIAMI-TULSA

Oct. 8 to 13, inclusive, 1917

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Many meetings of the Institute have exceeded in some respect all the meetings which preceded; for example, the technical papers of the Montana Meeting and the technical papers and visits of the Arizona Meeting, but it remained for the St. Louis-Joplin-Miami-Tulsa Meeting to establish a record in several respects: the greatest enthusiasm, the largest attendance, the most striking example of patriotic expression, and the most active discussion of the technical papers. Added to these records, mention should be made of the cordial hospitality extended

everywhere, but in this respect comparisons of Institute meetings are impossible. It is due to our hosts to record, however, that no detail of courteous welcome was omitted that could contribute to the comfort and pleasure of the visiting members and guests.

Three special cars attached to one of the regular trains of the Pennsylvania Railroad brought to St. Louis members from New York, Washington, Baltimore, Philadelphia, Harrisburg, Pittsburgh, and intermediate points. There were also delegations from important centers such as Chicago, Butte, Denver, Salt Lake City, San Francisco, Arizona. The headquarters of the Institute were at the Planters Hotel in St. Louis, and over 350 members had registered there before the evening of the opening day of the meeting. As the different parties arrived in St. Louis, they were met by local committees and conveyed in automobiles chiefly to the Planters Hotel, or the Missouri Athletic Association.

On Monday morning, Oct. 8, 1917, two well-attended and most interesting sessions, one on Milling and the other on Coal, were held at the Planters Hotel, the papers being well received and actively discussed, as described in another section of the Proceedings. At noon a luncheon was tendered the members at the Mercantile Club by the Engineers' Club of St. Louis. In the absence of the President of the Engineers' Club, Vice-President Baxter L. Brown presided and welcomed the guests. Response on behalf of the Institute was made by Vice-President Sidney J. Jennings, at the request of President Moore.

On Monday afternoon, a Patriotic Meeting was held in the large ball room of the Planters Hotel. This was one of the features of the meeting and expressed so well the spirit of the gathering and of the time and occasion that there were frequent demands for other sessions of the same kind during the meeting.

Patriotic Meeting

In calling the meeting to order, President Philip N. Moore asked the Secretary to read a letter of greeting from Past President L. D. Ricketts who regretted his inability to be present. This letter was received with prolonged applause, as were also the telegram from our Honorary Member, Herbert C. Hoover, Food Administrator, and the letter from our Honorary Member, Sir Robert A. Hadfield. Herbert C. Hoover's telegram follows:

> I regret exceedingly my inability to be present at the annual meeting of the Institute, but pressure of business in Washington makes it imperative that I remain here. May I ask you to convey to the members of the Institute my regret and best wishes, and urge upon them that they preach and practice the gospel of saving service and sacrifice, which must be brought home to every citizen in this land if we are to emerge from this cataclysm a unified and free people. The war can be won only by the practice of this doctrine. Our supply of foodstuffs is all too scanty to meet the demands, and that they may be adequate in quantity necessitates saving on the part of our people. Government indebtedness can only be paid out of savings. We must, as a people, be prepared to respond at the call of the world's need, submerging our personal inclinations and welfare that we may serve our Nation in this the greatest crisis which has ever confronted it. We must make whatever sacrifice is necessary, whether it be of our own personal expenditures, or as volunteers in the National service, or the supreme sacrifice as soldiers in the trenches, to make the world safe for Democracy. We must all save, serve, and sacrifice.

The letter from Sir Robert A. Hadfield was dated Aug. 15, 1917, and was as follows:

Just a few lines to say that a magnificent reception was given today to your troops marching through London. God bless America!

Today has been an epoch-making one in the history of the Anglo-Saxon Race. Whilst we English may not show our emotions so freely as others, America's course of action in the war, and the splendid way in which she is making preparations, has touched deeply the hearts of my countrymen, one and all of us.

As one of your British Honorary Members, a position I value so much, may I beg you to convey the above message to your President and Board of Management.

President Philip N. Moore then addressed the meeting as follows:

The President.—We come together, ladies and gentlemen, friends, guests, at a time unparalleled in our history. We planned this meeting with questioning. We were criticised when it was thought wise to hold it, on the theory that it was an undignified and unworthy occasion at the time when our thoughts and our hearts are with our sons and our husbands, who are going to the supreme test, but we hope to show by our attendance the feeling that on us, as engineers, the burden of the task rests, that by our coming together we strengthen our hands and unify our minds, that we prepare for greater service than ever before; that in no undignified way do we foregather; that in no unpatriotic manner do we assemble and seem to carry our lives in what might be termed the normal manner.

The supreme test is coming to us. The load has not been felt yet; only the edge of the cloud is over us; but by our coming together may we hope, gentlemen, to strengthen ourselves for the trial, to prepare for the burden, no matter how great, which may be laid on our shoulders; and may we, by this action, render our capacities for service greater, more solid; and give more efficient and less wasteful service than ever before. We have thought wise to gather the whole assemblage here, men, women and guests, to hear from the one representative committee of our profession now most prominently before the public as servants of the government, the one committee which perhaps is doing as much as any other committee to unify work in Washington, the committee which is abolishing duplication and which is an instance of governmental efficiency, our War Minerals Committee. (Applause.) We will hear their plans. We desire to welcome representatives of our Allies, and first from the old Motherland comes a greeting from a representative of the British Legation, Dr. Henry M. Ami. (Applause.)

Dr. Ami.—Your noble words have touched my heart most deeply. I am here at your request to join you in this work of seeing what can be done to advance and bring the end, peace, to an issue. I am deeply grateful and sensible of the honor bestowed upon me because, as a humble worker in the trade department of the British Embassy in Washington, I have been asked to come and in my humble capacity try and represent that wonderfully active and busy institution today, an institution which says very little, but works, I can assure you, very actively. I bring greetings to this audience from the Embassy and from the British War Mission to which I have been accredited, and the kind invitation of your President to be here present has come as a very deep sense of honor and privilege.

We are all here assembled for the one purpose, with the one spirit. The large gathering which I see before me is an earnest and a pledge of what will soon happen; that the United States has entered this great conflict on the side of righteousness, is one of the greatest assets of the Allies. I am a Canadian by birth, every drop of blood in me being French,

yet I am a British subject and here I am standing before you as a humble representative of the British Embassy; at the same time we feel that there is union in all our actions and it must be coöperation.

France, Britain and the United States have enormous interests on this continent, and they are united interests. I have always felt, Mr. President, in dealing with rocks and geology, that these three nations are the granite nations of the world. (Applause.) Granite, the most durable and the strongest of all rocks, combines three essential constituents, mica, quartz, and feldspar. To me, mica represents the French element of that rock, polished, smooth, shiny. Then comes the British, the quartz, so durable, lasting, resisting, typifying in a sense the British who came next as settlers on this continent; and then the feldspar, a useful and essential and also a composite mineral, which typifies also in other respects the United States. There we have the granite, the mica, the quartz, and the feldspar, each one in its own way useful and in every way necessary singly and separately, but together forming the strongest rock, typifying the nations to which we look forward to help bring peace and to form that peace league of nations which will endure forever. (Applause.)

The President.—It is our honor and our pleasure to have with us an official representative of the French High Commission to the United States. By a peculiarly happy combination, he comes to us as a soldier and a fellow mining engineer, Lieutenant Colonel Edouard de Billy, of the French Commission, representative of our allies of 140 years ago, who helped us then through a tight place, a representative of our allies who are helping us through a tight place now—Colonel de Billy. (Applause.)

Colonel de Billy.—It is a very agreeable duty for a Frenchman to speak on behalf of his country here in the United States, and especially in the city of St. Louis, so full of French remembrances, and in these solemn circumstances when nearly all the liberal countries of the world are engaged in this world war. None of us wanted the war and we can honestly say that all of us did everything possible to avoid it, but we were forced into it. (Applause.)

Russia and France were forced into the war, having been attacked and partly invaded. Great Britain was forced into the war because she had pledged herself to guarantee the neutrality of Belgium (Applause), and she did not think that a treaty bearing the signature of Great Britain was merely a scrap of paper. (Applause.) And then the United States has been forced into the war by the German intrigue in this and in neighboring countries, as well as by the submarine warfare, that direct attack on the rights of American citizens to travel on the seas; free commerce is a right for which the United States, from its beginning, has always made or always been ready to make war. (Applause.)

And now that we all have been forced into this war, we stand as one man, knowing that our defeat would mean the triumph of those political principles to which we are so strongly opposed, knowing that we are fighting for what is most dear to our hearts, for liberty, for justice, and for democracy, and that we also fight for independence; you also, Americans, as President Wilson told the first departing drafted men a few weeks ago, when he said that your boys were going to fight on European soil for the independence of the United States. (Applause.)

Speaking to engineers who have devoted their time partly to technical matters and partly to those social problems which confront all leaders

of men, I would like to point out very rapidly something of the characteristics of this war. The characteristics of the war are direct results of the development of modern weapons. Already the South African War had shown the importance of trenches, even the smallest trenches, the covering which makes it possible for a good shot, being protected, to stop a great many men for a long time, if he has a good weapon. The Manchurian War had afterward shown that to destroy a well made trench light artillery was not sufficient, that heavy guns were necessary. It had also shown that in this modern warfare an enormous supply of ammunition was necessary. In all the battles of Manchuria, both armies had to stay idle for many days, because their supplies had been spent and they were obliged to wait till new supplies should arrive from their distant bases.

Before 1914, the armies were only parts of the nations engaged in a war. It is only in this last war that whole nations have been in arms. In preceding wars, the armies were not sufficient in number to hold very long lines of trenches; trenches were short or discontinuous, so that the tactics consisted in trying to turn these lines and oblige the enemy to retreat and so to develop from a war of positions to a war of movements. But universal service, which had been the German method and compelled us French also to resort to it, brought it about that in a few weeks after the mobilization, whole nations on both sides were in arms, making a sufficient number of men to hold a continuous line of trenches extending for 739 kilometers from the frontier of Switzerland not only to the Belgian frontier, but even to the North Sea.

On both sides the armies entrenched along this whole line, and thus began the actual warfare, this war of trenches, with its engineering characteristics. I will not dwell upon those characteristics; you know them better than myself. All of you have been following with eager interest all the developments of trench making, of sapping, of mine blowing, and all the industries, behind the trench line, engaged in manufacturing guns, ammunition, propellants, explosives and all that is necessary for actual warfare; hence in the fighting countries practically all the industrial forces are now engaged in work for the war.

Not only has industry taken hold of all that is used to prepare for war, but the methods of war itself have been modelled after the methods of industry. Modern industries make efficiency out of specialization, and so it is on the front; men are selected according to their special ability or their physical strength or for some other quality. In the training schools, which are not far behind the fighting line, they are trained for some special purpose, so that when sent to the front some are specialized for the quick-firing machines, others for the throwing of hand grenades, others for trench mortars, others for other work, each one having his job as in a well conducted factory.

As regards the social results of this war, I said a while ago that it would lead the countries involved in the war more surely toward democracy, for three reasons, as I want to point out to you. First, since whole nations are in arms, whole nations must know what is the origin and what are the purposes of the war. If the governments were not backed by the great majority of the people, the war could not be continued. The second reason is that life in the trenches is a very democratic life; it does not prevent strong discipline from prevailing in the fighting armies, but this discipline does not prevent very friendly com-

panionship during those long days and months spent together in very hard conditions, in cold and in hot weather. The men, non-commissioned officers and men, have to stand together the same dangers, the same difficulties, enjoy the same glories and, from time to time, have to suffer the same unsuccesses, which leads to a friendly fellowship which will certainly have its results, after the war, in the settling, in more friendly manner, of the difficulties which may arise. The third reason is that in this war, where the men in command must constantly give the example, the loss of officers is very great, and it is from the ranks that the new officers have to be commissioned. In the French army, since the beginning of the war, more than 80,000 men have received commissions as officers, being selected and trained in special schools for two or three months and then returned to the front as officers, the door being open to everybody and the selection being made according to the ability and to the power of intelligence and to the effort, which is at the basis of sound democracy. (Applause.)

In this war of engineers, the part of the mining engineer has not been the least. Those who have remained behind, engaged in their work, are busy mining coal, copper and iron, and also gold. All these are as necessary to the maintenance of war as is food for the maintenance of life. The mining engineers who have enlisted or have been drafted have been assigned to the regiments of engineers, and there, whether building or rebuilding railroads, roads or bridges, or in the more dangerous work of making or repairing trenches, in sapping, in blowing mines, or in that most dangerous work of all—coming out of the trenches at night to place barbed-wire entanglements that will stop any assault of the enemy so long as they have not been destroyed by shelling—in all those operations the mining engineers have been called to show their professional ability as well as their gallantry and their contempt of danger.

Many of them will have a special task to do when the war is over, and it becomes necessary to rehabilitate those mines which the enemy is destroying while he is retreating, and to restore life to those valuable properties which the Hun is trying to doom to a definite destruction. Some of our American friends have asked what help they could render us in the rehabilitation of mines. To that I cannot give today any definite answer. A few mines have already been retaken, and have been found in a terrible state. Worst of all, the shafts have been blasted, but where and to what extent we do not know, for the mines are flooded. The first thing we shall have to do is to pump for weeks; only then shall we be able to know whether the shafts can be repaired or whether other shafts have to be sunk, and what will be the total work necessary for the rehabilitation of those mines.

Although I cannot yet tell you anything precise about that matter, I know that whenever we do ask help from you it will be offered tenfold as much as we ask (applause); and for that help that is to come, as for that which we are receiving now that you are at war beside us, for all that help, with all my heart, in behalf of my country, I thank you.

(The audience rose and applauded.)

The President.—Gentlemen, three cheers for France, France the inscrutable, France the superb, France the faithful. (The cheers were given.)

There comes to us also today, gentlemen, a messenger from another

old friend and ally, that great nation, greater even than ours by far, always the friend of America, struggling now with the new tools of democracy which have come to its unaccustomed hands, but still animated with a spirit which we are sure will eventually restore order and develop righteous government—another mining engineer, director of important enterprises—Dr. Fedor Foss, of the Russian Embassy. (Applause.)

Dr. Foss.—In these days when American mining and metallurgical men have an oportunity to review their work to win the war, Russian mining engineers are greeting you and sending to you the warmest wishes of good luck and success.

Every Russian who is familiar with the mineral industry knows what an amount of work the mining engineers of the United States have been able to do, and everybody realizes the part which the mining industry of the United States is playing in the world war. Your coöperation with your Government has brought essential results, and we hope will bring more. Russian mining men have not been so fortunate as their colleagues in the United States. From the early beginning of the war, united and animated by a patriotic movement, we came and offered our services to our Government, came with the hope that the coöperation of the Government and the people would lighten the burden of the war, but the old imperial Russian Government knew everything better than anybody and refused the assistance of men who did not belong to the bureaucracy.

You know the result of this policy; that one of the members of that Government was sentenced to hard labor for life for treason. We are not such stupid, dull-witted men, we mining and industrial men of Russia, that we could feel it consistent with our duty to be deprived of the right to help our country in time of danger, and we started to fight for permission to take our part in defending our country. You hardly can understand such a position, but it happened with a nation of 180,000,000 inhabitants. As you know, the big war started in August, 1914. We offered our help immediately after the beginning of the war and succeeded in receiving the orders for ammunition from our Government in May and June, 1915, when lack of ammunition had been felt on the Galician front very severely, and only then, under the pressure of retreat, could we organize public war industrial committees in which all intelligent forces of Russia have been united in producing munitions and war materials.

The old Government looked with great suspicion on these democratic efforts and institutions and tried to hinder, where it was possible, the work of the committees. In the files of the old Government we found orders not to send the iron and steel for the committee's work. In spite of these difficulties, Russian industrial men have shown some energy and pushed the work through in such a shape that in March, 1916, Russian artillery started to treat Germany very severely with 6-in. shells, and during the winter of 1916 we have had an abundance of 6- and 3-in. shells of our own production. I hardly need to mention what kind of help we have received from the United States producers and engineers. More than three-quarters of our works are equipped with American machines.

The initiative of Russian mining engineers was not limited to the production of shells. We tried to extend our work into all branches of

mining and metallurgy. We started to mine tungsten and smelt spelter and lead. We renewed quicksilver smelting, installed the recovery of coke byproducts for explosives, and increased more than three times the production of sulphuric acid. We tried to develop to the greatest possible extent our platinum production and have sent to our allies many tons of this important war material. During the war, Russia mobilized about 15,000,000 men from 19 to 43 years of age. That deprived the industries of their best class of workingmen. We substituted female labor to a large extent, but still it was a big task to maintain the production of all raw materials, such as coal, petroleum, iron, copper, etc., on the same level as before. In spite of these circumstances, we worked hard, and until the last we succeeded in supplying our armies with all the prime necessities in such amounts that our armies have been able to make brilliant offensives.

Beginning with the days of the revolution, all Russian industry began to be demoralized and production has been declining. We are sure that this is temporary. Now, when all the difficulties of the former regime are gone forever and when we are assisted by the United States and the Allies, we have full confidence that the time is not far away when new democratic Russia, defending her newborn freedom, will reassume very intensive work and, in common with our gallant Allies, bring her full industrial and military power to win victory over feudalism, over the Germans, the killers of women and children and the violators of all human rights. Going through the United States, I have often heard from many Americans that they are praying for the freedom of Russia. Permit me to assure you that we in Russia are praying for the welfare of the United States, the great protector of freedom. (Applause).

The President.—To those of us who feel that we, with our possibly inefficient democracy, arrive slowly at results which it seemed to many of us should have been reached earlier, it should arouse our sympathy to hear this tale of our fellow technical men, ready and willing to serve their land at the front, and yet held back by bureaucracy, and worse. That the day has come when they will work toward efficiency, let us all hope and believe.

And now for our own problems. I think every engineer who has been called to Washington, or has called himself there in his exuberance of patriotism, begins by feeling that there is a great loss of efficiency, a certain mulling around which could be avoided; but we should bear in mind that here is a governmental machine, planned and keyed for a certain duty, a task of a certain size, which suddenly has laid upon it a burden many times the weight of any that it had ever heretofore assumed. If we will think of that, gentlemen, and visualize what would be the result, even in our most highly specialized and systematized great organizations, if many times its normal task were forced upon it, it will make us more tolerant of the errors which are only too patent, and encourage us as to what the outcome may be. I lately had the fortune to visit Washington and get some slight glimpses from the outside of the functioning of things, and it was borne in on me that the work is being better and better done. I trust it will be felt by all of you a matter of pride that the one committee on which we are officially represented is performing good service and is of great value.

There comes to us to speak for his department, one which is largely

interested in this matter, an old friend, F. W. DeWolf, State Geologist of Illinois, now Assistant Director of the U. S. Bureau of Mines, an old friend with a new title, efficient, faithful, and friendly. (Applause).

Mr. DeWolf.—It is quite likely that the brief review which I shall try to give of the organizations outside those of the Army and Navy will be poorer than what you might read in some recent review or some periodical; nevertheless, a patriotic meeting of this sort is a good place to bring home to ourselves some of the outstanding features of this program.

Many of us recall the circumstances of the creation of the Naval Consulting Board, consisting of representatives of the national engineering societies and of eminent inventors and geniuses for organization. You will recall that as a result of other activities of the engineering societies, long before we were in the war, steps had been taken to make an industrial inventory of the entire country. Whether the direct or indirect results of that were far reaching, I do not know, but I think it is a very significant step that at that time the civil population of the country was ready to place at the Government's command the industries of the entire nation.

Then, a little later, came the creation of the Council of National Defense, consisting of the Secretaries of War, Navy, Interior, Agriculture, Commerce and Labor, and the creation of an Advisory Board of that Council with representatives of the leading industries of the country. Similar to this movement, which was essentially industrial, you will recall the activity of the scientists, the appointment by the National Academy of Science of representatives to constitute the National Research Council, which, like the Council of Defense, has committees almost without number.

These two councils, somewhat independent, nevertheless work together. The National Research Council in effect is the scientific branch of the Council of Defense, and they are housed together in a single building in Washington. In that building we have concentrated particularly the effort that is being made by civilians to assist the Government at this time and to put their resources at the disposal of the other Government agencies. Through this center also there is contact with the States and an intelligent effort to correlate the State activities with those of the Nation. Now, besides these two main groups of workers, we have had other special war boards and agencies created by statute or by presidential appointment. You are familiar with the War Industries Board which comprises representatives of the Army and of the Navy, and the chairmen of many of these defense committees. Together they comprise an advising agent on all purchases of munitions and necessary supplies. We have also the Shipping Board. While it has some concern in priority of shipments, as you know, its chief function is the building and operating of ships. Then we have the Priority Board, concerned in the systematic routing of essential materials and the execution of embargoes, if any such are required; also the Export Council, authorized to license all exports and to limit those licenses to the things that can be spared or that are to go to friendly countries. The more spectacular agencies, of course, at the present time are the Fuel Administrator and the Food Administrator; there is hardly need to say anything about the functions which they are exercising. The development in all these lines has been rapid and progressive and the evolution has been very real.

Of course, mistakes have been made. In many cases there has been lack of proportion which is now apparent but was not apparent at first. Some agencies, some efforts which at first seemed of immense importance, are now sidetracked or are mere eddies in the larger stream of activity. Similarly there has been rather objectionable independence, if not downright jealousy at times, between certain agencies. It was very natural that a group of business men coming to Washington to do a war job should lack information as to the work already being done in many of the old operating bureaus and departments. It is also obvious to all of us that many of those old bureaus and departments found it impossible to respond quickly to a war need, and therefore the reaction of these two groups, the volunteers, if I may call them that, and the old standbys in the departments, has made for a quicker and a more beneficial service than either could have given alone; and the initial confusion is growing less. Certain bureaus and departments are putting themselves into war lines. I recall the incident of an employee of the Quartermaster's Department who was greatly provoked when the war broke out; he said he had been busy for 20 years perfecting a system for handling the work of that department, and he had just completed and perfected it, and now the darn war had broken out and his system was no good. And so all of the existing departments have had to adjust themselves to an entirely new and unexpected condition.

A review of the official committees, hundreds of them, would show that many of them have disappeared, many of them are alive but not very active, and that there has been combination and coördination of still others; out of the confusion and the early mess and muddle, a great clearing up is already evident, and out of this great melting pot of ideas there is to be seen now, I think, the crystallizing of a definite policy and of strong, efficient agencies that will carry on the work. When the full story is told, I trust and believe that it will be found that the advantage, in the long run, is not with the nation that has a temporarily perfect organization, but with the nation that can receive and utilize spontaneous contributions and ideas from the great mass of its citizenship, to meet a crisis of this sort. (Applause.)

The President.—The War Minerals Committee has done the Institute the honor of electing as its Chairman our representative, Mr. William Young Westervelt. He will tell us something of the way their work is being conducted. It is to be remembered that in the membership of the American Institute of Mining Engineers are comprised producers of all the metals; in their files is probably a body of information regarding the mineral resources of this country which cannot be duplicated. I hope to impress upon you the desire of the War Minerals Committee that you may put at their service all this vast accumulation of knowledge, which they may sift and sort and gather for the service of our land.

Mr. W. Y. Westervelt.—President Moore has asked me, as he has just explained, to give you a few remarks on the origin and scope of the War Minerals Committee. Shortly after the outbreak of the war, Mr. Moore's personal patriotic feelings began to arouse a sympathetic note throughout the entire membership of the Institute. Apparently from all over the country men were writing in and asking "What can we mining engineers do to help in this crisis?" And at every Section meeting of the Institute, at Board meetings and wherever mining people were getting

together, the subject would crop up "What are we going to do?" The reason is that the American Institute of Mining Engineers will give precedence to none as a patriotic organization. (Applause.) Its members are eager and earnest to serve. They want to serve along the lines that they best understand; they want to serve as engineers, as mining engineers.

The question then became a very pressing one. Here was this vast amount of talent all over the country, eager and anxious to serve; what was going to be done about it? President Moore got very busy. He made a number of trips to Washington and found that other forces were working along common lines, and he eventually joined forces with these other influences. He found that the Association of American State Geologists, through their principal officers, Mr. Hotchkiss and Mr. DeWolf, had been down there for months working with an appreciation of the fact that large tonnages of certain necessary minerals were normally imported into this country and that these imports were, or naturally would be, curtailed by the cutting off of shipping; they had formed an Import Committee as a committee of the National Research Council, and had already commenced some work along that line. Then again there were the two great bureaus in Washington with which we, as miners and geologists, are so intimately connected, the United States Geological Survey, which has its long record of accomplishment, and the Bureau of Mines with a shorter but proportionately equally great accomplishment. The Directors of these bureaus had been receiving constant applications by letter and personally. Men had come there from all over the country, saying, "Mr. Director, put us to work, give us something to do." They realized that this willingness was of great value to the country, once placed in the right direction, and they were ready to take action.

The result of all these forces was that a committee was appointed by the two Government bureaus, the directors of the two bureaus each nominating a man, joining with the Association of State Geologists, with our own Institute and with the Mining and Metallurgical Society of America. Mr. David White, Chief Geologist, was nominated for the U. S. Geological Survey; Mr. A. G. White, Mining Economist, was nominated for the Bureau of Mines. Mr. Hotchkiss, with Mr. DeWolf as alternate, as these gentlemen could not both be in Washington all the time, was nominated to represent the American Association of State Geologists, and the National Research Council, and I had the honor of being appointed to represent you gentlemen and the Mining and Metallurgical Society of America. We had our organization meeting on July 12. The two Government bureaus, the Geological Survey and the Bureau of Mines, gave us every facility they could in the way of quarters and the use of their experts, whom they caused to come before us and review, in a general way, what had already been accomplished. Of course, as you know, both the Survey and the Bureau had been collecting various statistics, and a special issue of these was being rushed out on account of the war.

After listening to these gentlemen for a short while, it became evident that the work of the committee was going to be divided among three different classes of mineral products: First, those minerals which have heretofore been imported but are now restricted, although their pro-

duction can be stimulated by ordinary commercial means to meet the need; secondly, those minerals which, while they exist in sufficient quantities to carry us through the present crisis, cannot be developed by commercial means—that is to say, they cannot command sufficient price to justify the cost of exploitation, of equipment for exploitation, and therefore can be developed only with some sort of fostering on the part of the Government; thirdly, there undoubtedly are certain minerals which this country is not prepared to produce, and we shall have to put it straight up to the Shipping Board to see that ships are provided for these essential imports.

When we began to see something of the immediate scope of work that we had before us, we saw what it was we were going to have to ask the members of the profession to undertake for us. Mr. Moore has just spoken to you of the vast amount of information which must necessarily rest in the files of the membership of this great national organization. We have placed in the Bulletins for September and October, to which I would like especially to call attention, appeals for information along these lines. We particularly need information as to pyrites and manganese.

I will digress just a moment to dwell upon those two subjects. In 1916, we imported something like a million and a quarter tons of Spanish pyrites. In 1917, I suppose we shall not import more than a few hundred thousand tons. Import is steadily dropping off and has almost disappeared. We must meet that need by more shipping, or by domestic production. It is improbable, as nearly as we have been able to gather thus far, that the entire pyrites requirements can be met by stimulating production of simple pyrites deposits. It is, however, practically certain that if the technology of sulphuric acid making can be modified, not merely within the lines of what has already been accomplished, but along lines which are yet to be made general, with such modification we could furnish the necessary material by utilizing the pyrites deposits of the country. Those are the questions upon which we want information. We need information as to what the sulphuric acid industry can take, the names of concerns who will undertake to work on pyrites, requirements as to pyrites, and information as to pyrites properties.

On manganese the situation is somewhat similar. Consumption has to be supplied almost entirely by imports from Brazil, and involves a big shipping problem; we want to replace as much of that import as we can, and we have made a rough estimate that we might replace nearly half of it by certain modifications in the process of consumption and development of the deposits.

Mr. White, a member of the Committee, who will follow me, will explain to you that we are very anxious to gain details from those of you who can give us information, any information you have as to localities, nature of deposits, etc. Any reports you may have on pyrites and manganese, chromite, nickel—and we are beginning to worry about zinc and lead deposits—will be most valuable. We want also to get answers to the questionnaires which were placed in the last Bulletin. When you go home, if you are sinners of my own order, please dig out your Bulletin from your pile in the corner, where it is nicely filed to be read next month, and read this month's Bulletin this month, to the extent of the questionnaire, and see if you can help us in that. I am

sure it is only a question of getting you gentlemen to realize the necessities of the situation, the really serious situation; that we must have this pyrites, must have this manganese, and, to a lesser extent, other minerals. We must have them to win the war. I am sure that as soon as the members of this Institute realize the need for this information, we shall receive an ever increasing response to this request.

Just in closing, I want to mention another thing that is steadily coming to the front and is going to be one of the important functions of our committee work. We have got to appoint committees of voluntary engineers to go to various properties and investigate them. We are already organizing one of these committees to go into Virginia and assist in the development, to speed up the production, in the manganese field. Shortly it will be necessary to have other committees, and we shall need to call upon our membership for that purpose. We shall be glad to hear from engineers who have time to give, what their specialties are, and where they are to be found.

The President.—Mr. David White, of the U. S. Geological Survey, a gentleman who would probably dispute my statement that the members of the American Institute of Mining Engineers possess a larger body of information than any other, but representative of an organization which, I think I can safely say, possesses the largest body of publishable information, will now talk to us. (Applause.)

Mr. David White.—I am glad to tell you something of what the Survey is doing to help the more efficient prosecution of the war.

We once had disturbances on the Mexican border. Other exciting events have made us forget them, but at the time the country was brought keenly to realize the need for adequate maps, suitable for military use along the Mexican frontier and on our coast. Plans were made for coöperation with the War Department and the extensive mapping, as rapidly as it might be accomplished, of frontier and border strips. This has been going forward energetically, but the entry of the country into war has probably crippled rather than helped it. Yet we have, at the present moment, 38 parties of topographers in the field, and we are now covering, on a scale of 1 mile to the inch, and with every detail, between 100,000 and 200,000 square miles per month. (Applause.)

Before the war broke out, a great many of the members of the topographic branch, by virtue of their engineering training and interest, had entered the Army Reserve Corps, and after the outbreak of the war we found that a very large part of the topographic branch, the field engineer portion of it, was enrolled in the military service. We have now altogether 100 men who are wearing the uniform of the United States. Topographic mapping is essentially under army control or advice, although the engineers are detailed to the Geological Survey and are operating under the direction of Director George Otis Smith, who is himself a member of the Military Committee. About 40 members of the topographic staff are in France, engaged in making the surveys, as the lines are moved up to the front, and in the mapping of the recovered ground, as an army that we all hope will be victorious is moving toward Germany. (Applause.)

This detail abroad, this parting company with many of our colleagues has reduced the number, and it has been necessary to conduct a training school for topographic engineers. More recently the army has come to

our help and is detailing to us, from the volunteers and draft camps, men who are found on examination to have had training suitable as qualification for topographic mapping. In connection with the detachment that has gone to France, I might mention a rather interesting experiment which was contemplated, namely, the application of the topographic, rapid, plane-table triangulation from observation stations for locating enemy batteries, in order that they may be the more quickly paralyzed and given the least possible time for the removal of guns. How successful that has been, I do not know.

The entrance of the United States into the war naturally created an enormous demand for maps of different kinds. The War and the Navy Departments immediately realized the very great necessity for chartographic work. The Engraving Division of the Survey accordingly reproduced and furnished to the Navy all of the admiralty charts which a very greatly increased navy will need in this work. We have at the same time worked diligently to secure rare maps and unpublished maps of portions of our own country and of adjacent foreign countries—I do not count Canada here; she is not really a foreign country—and those have been furnished to the Army engineers and to the War College. I wish here to give a grateful vote of thanks for the many engineers attached to the oil companies and to other mining companies who have been engaged in foreign exploration who have so loyally given their unpublished maps for the use of the Government. We have published, in addition, a great number of maps for the War Department, and latterly we have, I may mention, a rather novel bit of work which we are doing; namely, we are just issuing a series of maps for the use of the air service, which follow the conventions of the French and British air-plane maps. They are to be used in our training camps by our aviators in order that they may be accustomed to the reading and interpretation of such maps and to the use of similar maps when they take them in their hands in France. (Applause). A paper published about a year ago on the use of photographic and topographic mapping, a bulletin by Mr. Bagley, promises to be invaluable to Allies and to Americans alike in the reproduction of aviation photographs and their interpretation in map form.

In the geologic branch, the Survey began with the assiduous field exploration of some of the alloy metals in particular, and the structural examination and search for possible oil supplies. At the same time, before war was declared and when it became apparent that information as to geologic and geographic conditions, water supplies, drainage, etc., were as important, if we were ever to have an army and put it in training, as to have topographic maps, we began the preparation of a series of reports. Twenty-three reports covering the underground water supplies of as many States, were transmitted to the War College and the Army engineers before the first of June. In the preparation of those reports, we had the most hearty coöperation of State geological surveys as well as the water resource branch of our own Survey.

Naturally you are most interested in that which I have the least time for, I am sorry to say, the work of the Survey, especially on war minerals. We are attempting to engineer and to coöperate in engineering what is in fact an inventory of the undiscovered or partially developed principal war supply commodities. In this work we are getting in closer union with the State geological surveys, mining bureaus, Council of Defense,

and through this Committee on War Minerals, with this great body of mining engineers. The work has been progressing from Alaska to the Gulf, and from East to West. It aims at the discovery of new deposits, at the estimation of the discovered deposits, at bringing those deposits to the attention of the mining engineer and of the entire mining fraternity, and promoting their development in every way. As geologists, we are diligently looking for trouble for the mining engineers because wherever the geologist believes that mining engineering advice and assistance are needed by some people who may be blacksmiths and farmers and retail merchants but who are spending money and time in very poor mining, we hope to bring their needs to your attention indirectly if not directly, and I know that you will help.

About the time the war began, or possibly before, we had begun to appreciate the public need for more frequent returns as to mineral production, but the outbreak of the war itself made that much more imperative, and we have, as you know, for quite a long time, been issuing monthly reports of the production of petroleum. However, the needs of the war administration boards of different kinds, of the mining fraternity and of the industrial fraternity, for more frequent reports of the progress attained in mining, have brought us to the consideration of monthly reports of manganese, pyrites and sulphur, and a number of others. I might say that at the request of the Administration Board, which is reaching out more and more into the future and is concerning itself with the question of adequate future supplies from domestic sources, since every ship that can be released must be released as soon as possible, they are taking an interest in these affairs also, and they have asked us to give weekly returns now of copper, lead, and zinc. We have been giving weekly returns, as you know, of coal for a couple of months or more, and in those returns we are not only stating production but the percentages of failure to reach the full production, with an explanation, by percentages, of the causes therefor. This is the point at which we come in contact with you again, first as geologists, and then as bookkeepers on the result of your work, if you please. (Applause.)

The President.—A representative of that great Government department which stands as the monument of our departed friend, Dr. Holmes, comes to us this evening, Mr. A. G. White, member of the War Minerals Committee, who will give us some idea of the functions, duties and activities of the Bureau of Mines in that department.

Mr. A. G. White.—As Secretary of the War Minerals Committee, I have a detailed knowledge of the inner working of the Committee on Mines. I hoped that I could be the last speaker and give you the advantage of some of that knowledge, but one of the committee members follows me, so it would hardly be safe to go into details at the present time.

The committee has been unanimously agreed in the problems on which we are working. The four agencies represented by this committee, the technical societies, the two Government bureaus, and the state geologists, are all primarily interested, and more and more so from the patriotic standpoint, in the development of the domestic mineral resources to meet our present needs and requirements. The Bureau of Mines has been making extensive studies and researches along many of the problems, particularly those dealing with the production and

technical methods of treatment of many of the minerals that are of particular importance at the present time. We have been making studies of the manganese situation, nickel, mercury, extensive questions of fuel economy, and investigation of nitrate plants. I am not going to speak in detail on the work of the Bureau of Mines at the present time, because our Assistant Director, at a later meeting, will discuss that matter in more detail. My own work with the Bureau has particularly dealt with a general survey of the war mineral situation, and the relationship of our work to that situation.

The question comes up in correspondence to the committee from many people who do not understand why sometimes we haven't got to their specialty; yet the question in our mind has been one of the relative weight and of the immediate character of the problem that might come before us. From the standpoint of essential use and importance, the two great basic materials necessary for the production of modern high-power explosives are nitrate, chiefly in the form of nitric acid, and concentrated sulphuric acid produced from pyrite, sulphur or smelter fumes. In the manufacture of this acid, platinum is essential for the contact process of producing sulphuric acid and for a possible similar use in the oxidation of ammonia to nitric acid. Mercury is essential as the material used for explosives and war munitions, for detonating practically all high-power explosives.

As a second group of minerals for military purposes, come the minerals for munitions and military equipment. Of these the chief group is formed of the steel alloys, which are small amounts of the rarer metals added to our iron and steel products to give them absolutely essential qualities particularly required in munitions, ordnance, battleships, machine tools, aeroplanes and automobiles. Of these the most important are manganese, tungsten, chromium, nickel, cobalt, molybdenum, vanadium and uranium. The major metals, iron, copper, lead and zinc, represent materials of which we have produced a very large percentage of the world's supply. The problems in those minerals are quite large, involving transportation and labor supply, but not requiring new research work or the stimulation of small and unknown deposits.

For the general purposes of immediate war work, those matters have been taken care of by larger and more general committees, and our particular Committee on War Minerals has not dealt extensively with those materials. In this second general group come aluminum, which is largely used for automobile and aeroplane parts; antimony, which is used as an alloy for hardening lead bullets; and magnesium, which is used in shells for smoke and light purposes, to detect the point at which they burst. As another group are the minerals essential to industry, including particularly agricultural fertilizers. Of those, potash, which is used chiefly for fertilizers, is one of the largest elements coming in this group; likewise phosphate. Then comes a group of miscellaneous minerals: tin, which is related to food containers; flake graphite, which is used for crucibles for steel, brass, bronze, etc., and is essential in metallurgy; mica, which is used as an insulating material for electrical apparatus and particularly as a transparent material in the construction of gas masks and in the automobile service; asbestos, which is used for fireproofing, ship construction, etc.; magnesite, which is used in the construction of refractory brick in many metallurgical furnaces.

The final group represents the large fuel group, coal, coke, petroleum and natural gas, and those again fall into the large group of materials which represent specific problems of transportation, labor supply, etc., and are quite largely outside the activity, at the present time at least, of the specific group of neglected minerals in which new production needs to be stimulated. In considering this list of problems, the question of weight is one of more and more importance. The problem today is the problem of the material which is hard to move. By eliminating certain of these large groups which we have not so far specifically considered, we have left about five and a quarter million tons of material which we should have liked to import if we could; in 1916 a great deal of it we did import. What we will do in 1917 or 1918 is an unanswered question.

Of this large group, nitrates from Chili represent, roughly, 1,350,000 tons, which we must still import; pyrite from Spain (some from Cuba) represents 1,400,000 short tons, which we can produce or for which we can substitute sulphur; potash from Germany formerly represented 1,100,000 tons, which we cannot import and did not import in 1916, and of which we must produce what we can; manganese ore and alloy which we imported from Brazil, Russia and India, represents 800,000 tons, which we are endeavoring to supply partly by domestic supplies; magnesite, which formerly came from Austria, represents 170,000 tons, which we cannot import, but we have good prospects of producing all of that material. Asbestos still comes from Canada; it represents 120,000 tons and we still expect to get it without interruption. Chromic iron ore comes from foreign sources and must still be imported, although we are doing our best to stimulate domestic production. It represents 130,000 tons. Tin represents 70,000 tons, which we must import, although we produce a little. It comes from the Straits Settlements and Bolivia. Graphite comes from Ceylon and represents 40,000 tons, but we can produce most of our graphite. Nickel comes from Canada; it represents about 40,000 tons. Sulphur comes from Japan; it represents 20,000 tons, and we can produce it. Antimony comes from China and from the smelters. It represents 15,000 tons; we must import some and can produce some. Tungsten comes from Burmah, Portugal, Bolivia and Australia; it represents 4000 tons and there is considerable domestic production. Mica comes from India and Canada; it represents 1200 tons, and there is considerable domestic production. Mercury comes from Spain, Italy and Austria; it represents 200 tons, and there is considerable domestic production. Cobalt comes from Canada and represents 100 tons. Molybdenum represents 8 tons. Platinum comes from Russia and Colombia; it represents 2.3 tons and there is very little produced in this country.

The point I want to make is this: we have here a group of minerals representing all the way from a million and a half tons to two tons of imports. The problems vary with the quantity of the material and the difficulty of domestic development. The problems we, as a War Minerals Committee, are attempting to take up, have got to be measured from the standpoint of tonnage, difficulty of transportation and domestic production. In some cases a small quantity may be readily imported, and those materials we are not giving immediate attention to because small amounts can be easily imported while the larger tonnages mean the sacrifice of military purposes. In this brief outline, which was the

first step we had to consider in our meeting of the War Minerals Committee, the specific minerals were taken up as the first and most essential mode of attack because of their relative importance.

I am coming down to a little detail now of the work of the War Minerals Committee: the general plan of our work is this, to concentrate information by establishing lines of contact to all the most readily available sources of information, to establish that information in a central file available for the information of all the Government agencies working on this problem; then from that central file to separate the wheat from the chaff as far as the preliminary indication of those sources which are the best sources to attempt to stimulate under the difficult conditions, under the conditions of higher expense and the fear of future foreign competition. Having done that, the next specific question is to get the feeling of the local pulse in different communities as to what the troubles and the needs of their domestic mineral situation may be; to take those troubles and needs, put them with the national needs, and see what policies this committee may recommend in the carrying out of certain of our widespread and far-reaching legislation and policies which are dealing with the mineral situation. At the present time, the measures that are being undertaken will have very far-reaching effects. We have as possibilities, in fact, transportation control, use control and price control, which may be essential. These broad questions of policy are continually being raised, and in order to be most wisely solved they require that every bit of information that will deal with this situation and make it clearer should be centralized for the use of those in whose hands the determination of those policies rests. (Applause.)

The President.—It needs no word by your Chairman to show you that the War Minerals Committee is looking at things in a broad way.

Among the men to whom, at least in the mind of your President, should be credited the vision which resulted in the organization of this War Minerals Committee, is the gentleman who will next speak to you. When he first came to me with the information that the plans had been laid for this scientific and efficient organization, I said to him, "How in the world did you manage it?" His answer was, "I told them I was going to sit on the stoop until they let me in." They have let him in, gentlemen, to the great enlightenment and betterment of all of us; Mr. Hotchkiss, State Geologist of Wisconsin, Secretary of the Association of State Geologists of the United States. (Applause.)

Mr. W. O. Hotchkiss.—As it is late, I shall say very little about the War Minerals Committee and its work. If any word that I can say can bring home to you the fact that this war is not an objective thing, that it is my war and that it is your war, then my purpose will have been satisfied. The War Minerals Committee then won't have to appeal to you any more for the sources of information which you have in your files. If each of you realizes in his heart the real fact of the situation and can say to himself "This is my war," with all the firmness and intensity of his being, then he will go home and he will give us all of the information that he has, that we may use it for the benefit of all of us in winning this war. (Applause.)

We are undertaking, as a nation, the biggest job that has ever fallen to our lot. We are fighting an enemy who knows every last pound of his resources and is using those resources with an inflexible purpose of

ultimate victory. On the other hand, we ourselves are scattered and divided; we are approaching this crisis practically unprepared and unarmed. Now we have got to get together. The fundamental basis of getting together is for each of us to realize that within him. I would give anything to have each man here realize, to have each citizen of this country realize that "This is my war," that it is not "That war over there," and that "it is up to me to do every last thing that I can to bring victory." We have admired in the past the German genius for organization; the patient plodding, result-getting activities of the German scientist, the perfect control which the Germans have exercised over economic, social and labor conditions, and now it appears to us that that whole structure has been built up with the one object in mind of waging a successful war which would put us all at the feet of an autocracy. (Applause.)

There are four fundamental things down to which we can analyze the whole situation of all preparedness. The first of these is that thing which the member of this Institute whom we most delight to honor is doing nobly and well, the work of Mr. Hoover, on food. Food is the first essential of war. The second essential of war is ammunition. The munition side is where our War Minerals Committee is trying to do its little bit, trying to contribute what we can, trying to accumulate all that is needed in regard to these minerals which are otherwise not attended to, the small things that are so likely to be forgotten.

Other members of the Committee have defined for you a war mineral. I wish to go a little further in that definition and illustrate; perhaps it may bring home to you a little better what a war mineral is, when I say that the needs of the United States in this war this year will take probably in excess of a quarter of our lead production of this year. Out of what is going normally into industries, it is going to take 25 per cent. or more and use it for an entirely different purpose. That is a tremendously upsetting thing on industry, quite naturally. Another thing, the Navy Department advertised for bids on one item of paint for the bottom of battleships that would take in excess of 10 per cent. of our entire production of quicksilver in one year. Now those things are what we mean when we say "war minerals," and in that way do they differ from peace minerals. We have got to have peace minerals; we must continue our use of minerals in every way possible, but we must make paramount and foremost the use of these things for war purposes.

I think the attitude of the average patriotic member of this Institute, the point of view, the frame of mind that he has had—I hope he won't have it any more—was the frame of mind in which I myself went to Washington as Secretary of the State Geologists' Association. I wanted to do my bit in my own state survey; I knew that every other member of our State Geologists' Association wanted to do his bit, so I got on the train to go to Washington to get someone to tell us what to do. The first man I went to was Daniel Willard, President of the Baltimore & Ohio Railroad, who was down there giving his services to the Government as Chairman of the Advisory Council of National Defense. I said, "To whom shall I go to tell me what to do so I can tell the rest of these people who are anxious to do something?" Mr. Willard looked at me and said, "Your frame of mind is exactly that in which I came here; I came down as a transportation expert in January, or earlier, I forget the exact date, and offered my services to the Government, expecting that some

high authority, someone who had long studied these problems, would tell me what I should do. I expected then to go to work as a transportation man and figure out the best way of accomplishing the transportation that would be needed. You will find undoubtedly the same situation. We have all been waiting for someone in authority to tell us what to do. The only thing for you to do is to do anything that you see to do, and you and the other State Geologists who get together any information that may be of value can consider that you are doing a patriotic service, even though that information may never be necessary." (Applause.) Now I think from conversations I have had with your President, that that has been the point of view of many mining engineers as well as many geologists. Each one of us has been looking for someone to tell him what to do. We have forgotten that this is a democracy of intellect as well as a democracy politically, that each man of us has had upon himself and upon himself alone, the responsibility for doing his share and everything that he could.

The third fundamental thing I would mention is the matter of ships. Others have mentioned it before, but I want to put this concrete example to you; the Quartermaster General's department has figured that it will take 400 ships in constant service to maintain a million troops in France. Now a million troops in France, gentlemen, is about 10 per cent. of the battle front. This great, wealthy nation, we who think we are doing so much, is going to put 10 per cent. of the men on the battle front, probably. Later on we hope to put on more, but we are not doing such an awful lot just at present. Four hundred ships in service are absolutely necessary to maintain that million men. I read a newspaper item the other day which said that the total American registry of ships capable of trans-Atlantic voyages was less than 650. Furthermore, the curve of that shipping is a decreasing curve; the submarine toll, it is estimated, will exceed the accessions by construction of new vessels put upon the water for the next year. There is our situation; that points out the fact that every man in his business who uses a thing that is taking any ships must conscientiously search his mind to see if there is not some other way in which that material can be gotten at home without the use of ships, because every bit of space on a ship means just so much more of it available for maintaining troops. More troops on the battle front means a shorter war and less serious loss. So much for shipping, gentlemen. It is a mighty important situation and the crucial situation so far as our mineral aid, our War Minerals Committee, is concerned.

The last item of war materials is men. In that also, gentlemen, I would that each one of us would think of it not objectively but subjectively. Before another year has rolled by, this Institute will have posted casualty lists of its members. More than that, you or I, before the year is gone, may have answered our last roll call in France. Now in view of that, that this is "my war", there are only two questions which each patriotic citizen can ask himself in general, mining engineers and everybody else. The first question is this, "Am I doing every last thing that I can that is my share?" The second thing is his attitude toward the Government. The question which every patriotic citizen should ask of the Government is not, "Are you wasting money?" But that question is and should be, "Are you doing everything, are you getting

every last thing that is necessary for this fight so that, in the words of President Wilson, the world may be made "safe for democracy?"

The President.—It is not necessary, after these talks this afternoon, for me to ask you, "Is this meeting worth while?" To have the privilege, before an audience like this, of bearing testimony to service, carries privileges both to speaker and to audience. Before we scatter, your President begs to call from you certain lines of information. It has come to be almost a habit for one committee or another in Washington to ask from your President nominations of men of our profession who are ready for service when called on. It may be the service of a single investigation requiring only a week or two; it may be that men able to go to Washington and join that great army of patriotic citizens on the Government payroll at $1 per annum are wanted. At the present time I have a request to name a number of men, experts in various lines of our profession, who are ready for work in the latter class. As you go, I beg you to bear this request in mind. Those of you who are in either of these classes ready for temporary or ready for permanent service will do us and the Government a favor if you will give your names. It may be that you will never hear from the act. Names have gone to committees in Washington, nominations of engineers for service, that have never been heard from. Nominations for serious tasks have gone to Washington which have been heard from and have brought great results in service.

The Patriotic Meeting then adjourned.

On Monday evening a dinner was tendered the members and guests at the Planters Hotel by the St. Louis members of the Institute. President Philip N. Moore presided and addresses were made by Dr. H. M. Ami, Lieut. Colonel Edouard de Billy, Dr. Fedor F. Foss, F. W. deWolf, Capt. Robert W. Hunt, T. A. Rickard, Edwin Ludlow, and Vice President Jennings.

On Monday the ladies attended the Patriotic Meeting and the dinner, but were entertained at luncheon by the St. Louis ladies at Scruggs, Vandervoort & Barney.

Technical Excursions

Tuesday, October 9, 1917.—Two alternate optional trips were arranged for the men on Tuesday morning, as follows:

Coke Trip.—An excursion to the 1000-ton byproduct coking plant of the Laclede Gas Co. This is a Koppers plant for producing gas for city distribution, and coke for furnace and domestic consumption.

Fire-brick Trip.—An excursion to the large fire-brick plants of the Laclede-Christy Co. and the Evens & Howard Co., in West St. Louis, who also make sewer pipe, tiling, etc., from local plastic clays and central Missouri flint clays.

On Tuesday morning there were also two simultaneous sessions—one on Lead and one on Petroleum—at the Planters Hotel, while the ladies met and organized the St. Louis Section of the Women's Auxiliary.

At noon the party boarded the Mississippi River steamer "*Sidney*" and immediately upon embarking luncheon was served on the upper and lower decks. An orchestra and vocal and instrumental quartet provided music for the guests. After luncheon, many availed themselves of the opportunity to dance in the enclosed cabin on the upper deck. At 2 p.m.

there was a session of the War Minerals Committee which attracted so much interest that it was continued well into the afternoon. About 4 o'clock, the boat was landed at the Herculaneum smelter of the St. Joseph Lead Co. Careful and efficient arrangements had been made for taking care of the visitors at this plant and all were given ample opportunity to visit and observe the chief operations, after which the party again boarded the boat and the return trip to St. Louis was begun. Supper was served on board the boat, the same excellent arrangements for handling the very large party being observed, and the remainder of the evening was occupied with a technical session on Miscellaneous Subjects. Those who did not care to attend the technical session were given an opportunity to dance in another part of the boat, and the occasion was unanimously voted a conspicuous success, from both the social and technical standpoints.

Wednesday, October 10, 1917.—Some members of the party who were especially interested in the subject of coal availed themselves of an opportunity for an optional excursion to Nokomis, Ill., a distance of 72 miles from St. Louis. Breakfast was served on the train, and a most interesting inspection was made of the mines of Rutledge & Taylor, which have a capacity of 5000 tons, and of the Keller Mining Co. This party returned in time to take the special train to the Southwest with the remainder of the Institute party.

On Wednesday morning there were also optional visits as follows:

Steel Trip.—An excursion by automobile to Granite City, Ill., to visit the steel castings plants of the Commonwealth Steel Co., the American Steel Foundry Co., and the National Enameling & Stamping Co. (granite-iron ware).

Zinc Smelter Trip.—Granby zinc smelters of the American Zinc, Lead & Smelting Co. in East St. Louis, Ill., where spelter and sulphuric acid are made from Joplin ores.

Electric Trip.—An excursion to the plants of the Wagner Electric Co., and the Curtis Manufacturing Co., manufacturers of mill machinery in West St. Louis.

Wednesday afternoon at 2 o'clock there were simultaneous sessions on Ore Deposits and on Iron and Steel. There was also an optional excursion to the Busch-Sulzer Diesel Engine Co. at Second and Utah Streets, in South St. Louis, and to the adjoining new Bevo plant.

From 4:00 to 6:00 p.m., a reception was given at the residence of President and Mrs. Moore. This proved to be a most welcome opportunity for the guests to express their appreciation to their host and hostess and congratulations upon what had already been established as one of the most successful Institute meetings ever held, and also gave to many visitors their first introduction to the hospitalities of a charming southern home. All were forced to leave earlier than desire prompted in order to prepare to join the party of 198 persons which left on the special train for the Joplin district.

The entertainment of the ladies on Wednesday included an automobile trip to the St. Louis Art Museum, followed by luncheon at the Country Club, and the reception at the residence of President and Mrs. Moore.

Thursday, October 11, 1917.—Very early Thursday morning, the special train for Joplin arrived at Arcadia, Kan., where an opportunity was given to see one of the mammoth steam shovels in operation in the

Southeast Kansas coal field. Breakfast was then served in the diners attached to the special train, and the excellent arrangements of the local committee were again in evidence in that there were good accommodations even for the party of much larger size than had been expected.

Upon the arrival of the special train at Webb City, Mo., it was met by a very large number of automobiles and a most interesting trip was taken through the sheet ground district, visiting a number of zinc-lead mines and mills, and finally assembling at the American Davey mine, where luncheon was served, after which an opportunity was given for visits underground. At 4:00 p.m. the party assembled at the Connor Hotel in Joplin, Mo., where a session on Zinc was held, and, simultaneously, a session of the War Minerals Committee. In the meantime the special train had been taken to Joplin and was available to all the travelers, now numbering over 200.

A dinner tendered by the members of the Institute in the Joplin district was held at the Connor Hotel. A very interesting feature preceding this dinner was a Wild West Exhibition given by several of the local members in costume. C. T. Orr presided at the dinner and the speakers were: Victor Rakowsky, Chairman of the local committee, O. D. Royse, President Philip N. Moore, Henry M. Ami, Edmond Paix, Fedor F. Foss, W. Y. Westervelt, and W. H. Seed. After the dinner and speaking, some songs were rendered by the "Wild West" contingent and then several of the guests engaged in dancing until the end of a very delightful evening.

Friday, October 12, 1917.—During the early morning hours, the special train had conveyed the party into the State of Oklahoma. A large number of automobiles were in readiness to convey it to the top of Blue Mound, from which an expansive and interesting view was had of a large part of the Miami district. Breakfast was served on the top of Blue Mound and the party was then taken in automobiles for visits to mines and mills of the Miami district. A luncheon was tendered by the citizens of Miami in the basement of the First Christian Church. The ladies of Miami acted as hostesses on this occasion, and too much cannot be said of the hospitable and efficient way in which all were cared for, notwithstanding that the party very much exceeded in numbers anything that had been anticipated. The train then conveyed the members and guests to Tulsa, Okla., a stop being made just outside of Tulsa at the Cosden oil refinery. The staff of this refinery had prepared in a most efficient way for the reception and guidance of the party. A leader divided the different members into groups of ten, and each group was taken in charge by one of the technical men of the refinery and shown through the different parts of the plant. In this way every detail that could be made public was explained to all. This is one of the largest refineries of the Mid-continental field, and besides the effectiveness of the arrangements, the visitors were particularly impressed with the great orderliness and cleanliness of the works, and the extensive application of scientific methods to the processes in operation.

Upon arrival at Tulsa, a dinner was tendered by the Tulsa members. This was held at the Tulsa Hotel and the great feature of the occasion was the presence among us of the much beloved Past President and Honorary Member, Dr. James F. Kemp. Dr. Kemp made the welcoming address at the dinner and was received with prolonged and en-

thusiastic applause. Vice-President Jennings presided during the dinner, and the speakers, besides Past President Kemp, were: Mayor Simmons, of Tulsa, Senator Owen, Dr. H. M. Ami, Edmond Paix, Ivan M. Goubkin, H. A. Buehler and Prof. W. E. McCourt. It was expected that the dinner would be followed by a session on Petroleum, but the assembled members voted to substitute for this session a meeting to discuss the formation of a Tulsa Petroleum Section of the Institute. Such a meeting was, therefore, held, David White presiding.

Saturday, October 13, 1917.—A special train conveyed the members and guests to the town of Oilton, Okla., where automobiles carried them through what was literally a forest of oil derricks. Excellent arrangements had been made both for taking care of the members without confusion or delay and for the trips, in order that each one might have an opportunity to see what would most interest him, whether it were the geology of the Cushing oil field, the refining of petroleum, or the extraction of gasoline from casing-head gas. At Drumright a stop was made for buffet luncheon, which was tendered by the members and the citizens of that town, and was served by the ladies, and then the trip was continued by automobile through the district to the town of Shamrock, where the party was met by the special train again and conveyed, first, to Tulsa, and thence to St. Louis, or other points of destination, thus ending a meeting that will long be remembered for the hospitality of the entertainment offered, the courtesy of the hosts and hostesses, the variety of professional and technical interests, and the painstaking and admirable manner in which a most complicated series of visits was accomplished.

The following members and guests registered at different points during the meeting, but many who were in attendance failed to sign the register:

R. E. Adam, Washington, D. C.
B. C. Adams and Guest, Joplin, Mo.
Lawrence Addicks, New York, N. Y.
W. H. Alexander, St. Louis, Mo.
R. H. Allen, Joplin, Mo.
Mrs. R. H. Allen, Joplin, Mo.
H. M. Ami, Washington, D. C.
R. G. Amidon, Cornucopia, Ore.
L. D. Anderson, Midvale, Utah.
Eiji Arakawa, Kyoto, Japan.
D. M. Armstead, St. Louis, Mo.
Mrs. D. M. Armstead, St. Louis, Mo.
F. H. Armstrong, Vulcan, Mich.
R. E. Baird, Webb City, Mo.
Mrs. R. E. Baird, Webb City, Mo.
R. E. Baker, Keokuk, Ia.
W. R. Askwith, Joplin, Mo.
C. K. Baldwin, Chicago, Ill.
Mrs. C. K. Baldwin, Chicago, Ill.
W. A. Bankson, Joplin, Mo.
Mrs. W. A. Bankson, Joplin, Mo.
E. S. Bardwell, Great Falls, Mont.
G. Barnett.
J. A. Barr, Mt. Pleasant, Tenn.
R. W. Barell, St. Louis, Mo.
E. Barth, Tulsa, Okla.
K. C. Barth, Chicago, Ill.
A. F. Bassett, Huntington, W. Va.
Mowry Bates, Tulsa, Okla.
R. R. Bayless, Commerce, Okla.
Mrs. R. R. Bayless, Commerce, Okla.
D. J. Bedford, St. Louis, Mo.
A. L. Beekly, Tulsa, Okla.
C. W. Benedict, Tulsa, Okla.
Mrs. C. W. Benedict, Tulsa, Okla.
O. M. Bilharz and Four Guests, Baxter Springs.
E. E. Billow, Chicago, Ill.
R. A. Bingham, W. Orange, N. J.
C. A. Blair, Carthage, Ill.
Mrs. C. A. Blair, Carthage, Ill.
E. L. Blossom, New York, N. Y.
J. R. Boddie, St. Louis, Mo.
W. C. Bohn, Galconda, Ill.
A. A. Bonsack, St. Louis, Mo.
Mrs. A. A. Bonsack, St. Louis, Mo.
N. J. von Borries, Picher, Okla.
C. P. Bowie, San Francisco, Cal.
N. B. Braly, Butte, Mont.
T. T. Brewster, St. Louis, Mo.
A. D. Brokaw, Chicago, Ill.
O. F. Brinton, Baxter Springs.
Mrs. O. F. Brinton, Baxter Springs.
R. H. Brown, Jr., Joplin, Mo.

Mrs. R. H. Brown, Joplin, Mo.
C. S. Bruner, Granby, Mo.
George Bryant, Joplin, Mo.
J. E. Buckley, Tulsa, Okla.
L. P. Buchanan, Joplin, Mo.
L. R. Budrow, Sonora, Mex.
F. N. Bendelari, Joplin, Mo.
Ross Bowles.
P. B. Butler, Joplin, Mo.
H. A. Buehler, Rolla, Mo.
C. T. Bumgarner, Tulsa, Okla.
I. L. Burch and Guest, Joplin, Mo.
C. C. Burger, New York, N. Y.
Mrs. C. C. Burger, New York, N. Y.
C. L. Burger, Webb City, Mo.
C. W. Burgess, Webb City, Mo.
K. Burnes, St. Louis, Mo.
A. J. Burnham and Guest, Joplin, Mo.
O. C. Burrell, Mascot, Tenn.
G. E. Burton, Norman, Okla.
E. W. Buskett and Guest, Joplin, Mo.
Mrs. P. B. Butler, Joplin, Mo.
R. S. Butler and Guest, Joplin, Mo.
D. H. Cadmus, Huntington, Ark.
G. H. Cady, Urbana, Ill.
A. F. Carmean, Carthage, Ill.
J. G. Carmean and Guest, Joplin, Mo.
A. B. Carpenter, Los Angeles, Cal.
Mrs. A. B. Carpenter, Los Angeles, Cal.
M. E. Carr, Tulsa, Okla.
F. Carroll, Denver, Colo.
H. F. Carter, Mexico City, Mex.
R. M. Catlin, Franklin, N. J.
Mrs. R. M. Catlin, Franklin, N. J.
J. A. Caselton, St. Louis, Mo.
G. S. Chapman, Joplin, Mo.
H. Charlton, Chicago, Ill.
R. Chavez, Rolla, Mo.
T. H. Clagett, Bluefield, W. Va.
H. S. Clark, Rolla, Mo.
J. M. Clark.
S. S. Clark, Leadwood, Mo.
Mrs. S. S. Clark, Leadwood, Mo.
C. Y. Clayton, Rolla, Mo.
L. Clayton and Guest, Joplin, Mo.
E. M. Clelland.
J. S. Cocking, Calumet, Mich.
H. S. Coe, Mound City, Kan.
W. W. Coe, Roanoke, Va.
F. F. Colcord, New York, N. Y.
P. R. Coldren and Guest, Joplin, Mo.
J. T. Cole, Miami, Ariz.
R. A. Conkling, Tulsa, Okla.
Mrs. R. A. Conkling, Tulsa, Okla.
E. T. Conner, Rydal, Pa.
Mrs. E. T. Conner, Rydal, Pa.
Miss Margaret Conner, Rydal, Pa.
J. W. Coos, Jr.
C. S. Corbett, Edwardsville.
G. B. Corless, Joplin, Mo.
Mrs. G. B. Corless, Joplin, Mo.
D. C. Corner and Guest.
Mrs. H. C. Cosgrove, Joplin, Mo.
Thos. Cowperthwaite.
G. H. Cox, Rolla, Mo.
H. A. Coy, Mascot, Tenn.
W. R. Crane, State College, Pa.
J. B. Cunningham, Houghton, Mich.
D. H. Curry, Denver, Colo.
G. M. Davidson, Chicago, Ill.
Mrs. G. M. Davidson, Chicago, Ill.
L. J. Davis, St. Louis, Mo.
S. M. Davis, Baxter Springs.
E. de Billy.
L. A. Delano, Bonne Terre, Mo.
L. de Gall, Joplin, Mo.
A. N. Detweiler, Springfield, Ill.
F. W. DeWolfe, Urbana, Ill.
Mrs. F. W. DeWolfe, Urbana, Ill.
A. W. Dickinson, St. Louis, Mo.
Mrs. A. W. Dickinson, St. Louis, Mo.
E. S. Dickinson, Carterville, Mo.
E. E. Dieffenbach, Newark, N. J.
C. F. Dike, Joplin, Mo.
W. E. Doan, Miami, Ariz.
W. E. Dodge, Shawnee, Okla.
H. W. Doennecke, Rolla, Mo.
W. E. Doll, St. Louis, Mo.
L. G. Donnelly, Tulsa, Okla.
A. H. Donnewald, Tulsa, Okla.
P. W. Donovan, Minneapolis, Minn.
R. R. Dook, St. Louis, Mo.
J. J. Dorney, St. Louis, Mo.
J. V. N. Dorr, New York, N. Y.
Miss A. Dorr, New York, N. Y.
C. B. Douglas, Tulsa, Okla.
W. A. Downing, Tulsa, Okla.
M. D. Draper, Baxter Springs.
C. G. Dresser, Mine La Motte, Mo.
F. D. Druding, Flat River, Mo.
Mrs. F. D. Druding, Flat River, Mo.
J. B. Duga, Bellaire, Ohio.
E. T. Dumble, Houston, Texas.
D. D. Dunkin, Webb City, Mo.
M. Dwyer, Joplin, Mo.
W. H. Eardley.
Frank Eberle, Joplin, Mo.
N. P. Eberlein, Miami, Ariz.
C. R. Eckes, Tulsa, Okla.
Mrs. C. R. Eckes, Tulsa, Okla.
J. A. Ede, LaSalle, Ill.
F. S. Elfred, Baxter Springs.
J. R. Elmendorf, Denver, Colo.
J. B. Emerson, Tulsa, Okla.
J. N. Esselstyn, Kuttawa, Ky.
H. C. Estep, Cleveland, Ohio.
W. S. Estep and Guest, Joplin, Mo.
Esterley, Joplin, Mo.
W. H. Evans, St. Louis, Mo.
C. G. Ewing, St. Louis, Mo.
A. H. Fay, Washington, D. C.
C. P. Fiske, Palmerton, Pa.
T. E. Flaherty, St. Louis, Mo.
Mrs. T. E. Flaherty, St. Louis, Mo.
J. E. Flanders, Rolla, Mo.
R. Fleming, Tulsa, Okla.
J. R. Fordyce, Little Rock, Ark.
F. F. Foss, Petrograd, Russia.
R. S. Foster, Bonne Terre, Mo.
Mrs. R. S. Foster, Bonne Terre, Mo.
E. L. Foucar, San Francisco, Cal.
R. W. Freyschlag, Joplin, Mo.

H. L. Friberger, Salt Lake City, Utah.
A. M. Gaines and Guest, Joplin, Mo.
J. H. Gardner, Tulsa, Okla.
H. Garlichs, New York, N. Y.
F. V. Gartung, Miami, Ariz.
E. S. Gatch, St. Louis, Mo.
N. Gatch, St. Louis, Mo.
E. Gayford, Salt Lake City, Utah.
F. B. Gaarhart, Iola, Kan.
J. I. Geddes, Jr., Joplin, Mo.
F. H. Geib, Rolla, Mo.
F. E. Glasgow, St. Louis, Mo.
Mrs. F. E. Glasgow, St. Louis, Mo.
R. J. Glendinning, Salt Lake City, Utah.
Mrs. R. J. Glendinning, Salt Lake City, Utah.
L. H. Goldman, St. Louis, Mo.
T. F. Golick, Rolla, Mo.
C. W. Goodale, Butte, Mont.
O. Gotsch, Jr., St. Louis, Mo.
I. M. Goubkin, Washington, D. C. (Petrograd, Russia).
W. M. Garred, Miami, Ariz.
B. Gratz, St. Louis, Mo.
Arba Green, Miami, Ariz.
Mrs. Arba Green, Miami, Ariz.
R. C. Greenfield, Milwaukee, Wis.
A. P. Greensfelder.
F. C. Grelein.
A. L. Gresham, New York, N. Y.
John Grifford, Miami, Ariz.
C. A. Grisham, St. Louis, Mo.
A. D. Groff, New York, N. Y.
E. L. Gruver, New York, N. Y.
H. S. Gulick, St. Louis, Mo.
J. W. Gwinn, Joplin, Mo.
Dorsey Hager, Tulsa, Okla.
Mrs. Dorsey Hager, Tulsa, Okla.
C. M. Haight, Franklin, N. J.
Mrs. C. M. Haight, Franklin, N. J.
R. D. Hall, New York, N. Y.
R. G. Hall, St. Louis, Mo.
Mrs. R. G. Hall, St. Louis, Mo.
F. Hamilton, San Francisco, Cal.
H. P. Hamilton, Two Rivers, Wis.
J. H. Hance, Iowa City, Ia.
K. F. Hansen, Rolla, Mo.
H. W. Hardinge, New York, N. Y.
C. C. Harris, Webb City, Mo.
A. L. Harvey, Baxter Springs.
B. Hasbrouck, Tulsa, Okla.
J. B. Hastings, Los Angeles, Cal.
J. W. Howell, Joplin, Mo.
Mrs. J. W. Howell, Joplin, Mo.
R. S. Hazeltine, Kansas City, Mo.
K. C. Heald, Pawhuska, Okla.
A. G. Heggem, Tulsa, Okla.
G. O. Helmstaedter, Chicago, Ill.
J. N. Herald, Tulsa, Okla.
C. M. Herron, Baxter Springs.
Mrs. C. M. Herron, Baxter Springs.
D. F. Hewett, Washington, D. C.
J. C. Hewitt, San Francisco, Cal.
L. Hill, St. Louis, Mo.
Mrs. R. R. Hill, New Mexico.
E. C. Hinder, Bartlesville, Okla.
C. Hitchcock, Bonne Terre, Mo.
H. G. Hixon, Chicago, Ill.
J. E. Hodge, Minneapolis, Minn.
E. A. Holbrook, Urbana, Ill.
L. S. Holstein, Palmerton, Pa.
C. W. Honess, Norman, Okla.
J. E. Hope, Tulsa, Okla.
F. A. Hornaday.
W. O. Hotchkiss, Madison, Wis.
J. N. Houser, St. Louis, Mo.
Mrs. J. N. Houser, St. Louis, Mo.
E. R. Householder, Rolla, Mo.
Mrs. B. B. Howard, Joplin, Mo
Carl Howard.
S. A. Hughes, St. Louis, Mo.
Victor Hughes, Tulsa, Okla.
W. D. Hume, Tulsa, Okla.
G. P. Hulst, E. Chicago, Ind.
Dan Hunt, Tulsa, Okla.
R. W. Hunt, Chicago, Ill.
S. L. Hurlbut and Guest, Houston, Tex.
J. B. Hutchison, Mingus, Tex.
L. L. Hutchison, Tulsa, Okla.
Mrs. L. L. Hutchison, Tulsa, Okla.
J. C. Iguirin, Webb City, Mo.
A. O. Ihlseng, Joplin, Mo.
S. Inone, Tokio, Japan.
Alfred Invenz, Joplin, Mo.
I. A. Ionides, Denver, Colo.
F. D. James, St. Louis, Mo.
C. H. Jaccard, Webb City, Mo.
P. F. Jarvis, St. Louis, Mo.
W. R. Jarvis, Pittsburgh, Pa.
S. J. Jennings, New York, N. Y.
Mrs. S. J. Jennings, New York, N. Y.
J. A. Jess, Springfield, Ill.
F. G. Jewett, Minneapolis, Minn.
A. G. Johnson, Baxter Springs.
Mrs. A. G. Johnson, Baxter Springs.
E. C. Johnson, Chicago, Ill.
E. M. Johnson and Two Guests, Joplin, Mo.
H. M. Johnston, Tulsa, Okla.
J. Johnston, St. Louis, Mo.
Mrs. J. Johnston, St. Louis, Mo.
Vernon Jones, Joplin, Mo.
Mrs Vernon Jones, Joplin, Mo.
F. F. Jorgensen, Gillespie, Ill.
J. F. Kemp, Tulsa, Okla.
Mrs. J. F. Kemp, Tulsa, Okla.
S. A. Kenoyer.
R. B. T. Kiliani, New York, N. Y.
H. Kingsbury and Guest, Baxter Springs.
Mrs. C. T. Kirk, Tulsa, Okla.
S. F. Kirkpatrick, Kingston, Ontario, Canada.
H. W. Kitson, New York, N. Y.
I. V. Korts, Carthage, Ill.
H. B. Kummel, Trenton, N. J.
J. N. Knapp, Ardmore, Pa.
Jos. E. Klaner.
H. Landes, Seattle, Wash.
W. H. Landreth, Joplin, Mo.
Mrs. W. H. Landreth, Joplin, Mo.

O. M. Landreth, Joplin, Mo.
L. W. Longhorne, Joplin, Mo.
C. L. Larson, Kellogg, Idaho.
W. G. LaRue, Duluth, Minn.
N. W. Latimer and Guest, Joplin, Mo.
E. T. Lednum, Joplin, Mo.
Mrs. E. T. Lednum, Joplin, Mo.
J. H. Lees, Des Moines, Ia.
A. Leggat, Butte, Mont.
E. F. L'Engle, Joplin, Mo.
Mrs. E. F. L'Engle, Joplin, Mo.
T. F. Lennan, Joplin, Mo.
Mrs. T. F. Lennan, Joplin, Mo.
O. H. Leonard, Tulsa, Okla.
A. J. Levorson, Kansas City, Mo.
J. G. Lewis, St. Louis, Mo.
H. K. Li, Houghton, Mich.
P. B. Liebermann, Newark, N. J.
S. P. Lindau, Herculaneum, Mo.
Mrs. S. P. Lindau, Herculaneum, Mo.
C. Lindmueller, Chicago, Ill.
O. Longacre, Jr., and Guest.
Mrs. O. Longacre, Jr.
R. D. Longyear, Minneapolis, Minn.
S. Lubarski, Russian Embassy.
A. F. Lucas, Washington, D. C.
E. Ludlow, Lansford, Pa.
Mrs. E. Ludlow, Lansford, Pa.
R. S. McBeth, New York, N. Y.
W. E. McCourt, St. Louis, Mo.
Mrs. W. E. McCourt, St. Louis, Mo.
P. M. McCorkle, Gillespie, Ill.
Mrs. E. W. McCrary, Tulsa, Okla.
A. J. McElheuir, Miami, Ariz.
C. W. McFerron, Tulsa, Okla.
J. J. McLellan and Guest, Webb City, Mo.
Angus McLeod, Tulsa, Okla.
V. H. McNutt, Lexington, Ky.
Mrs. V. H. McNutt, Lexington, Ky.
A. L. McRae, Rolla, Mo.
J. W. Malcolmson, Kansas City, Mo.
Mrs. J. W. Malcolmson, Kansas City, Mo.
D. D. Malloy, Joplin, Mo.
J. L. Malm, Denver, Colo.
O. N. Maness, Rolla, Mo.
H. T. Mann, Rolla, Mo.
G. E. Marble, St. Louis, Mo.
Miss M. Mark, St. Louis, Mo.
G. W. Mark, Kingman, Ariz.
Miss E. Mark, St. Louis, Mo.
A. W. Marsh, Webb City, Mo.
C. E. Marshall, New York, N. Y.
R. L. Marston, Rolla, Mo.
F. L. Matthews.
Mrs. F. L. Matthews.
L. W. Mayer, New York, N. Y.
A. J. Meier, St. Louis, Mo.
W. Meredith, Tulsa, Okla.
J. B. Mesmer, Tulsa, Okla.
J. B. Millar.
C. E. Miller, Drumright, Okla.
Walter Miller, Tulsa, Okla.
Willet G. Miller, Toronto, Canada.
J .O. Mitchell, Tulsa, Okla.
D. W. Moffitt, Tulsa, Okla.
J. D. Mooney, Newark, N. J.
P. N. Moore, St. Louis, Mo.
Mrs. P. N. Moore, St. Louis, Mo.
R. C. Moore, Lawrence, Kan.
N. J. Morgereidge, Kansas City, Mo.
J. M. Morris, Jr., Rolla, Mo.
H. L. Morton, Globe, Ariz.
M. Moss, Huntsville, Ala.
H. S. Mudd, Los Angeles, Cal.
D. D. Muir, Jr., Miami, Ariz.
Mrs. D. D. Muir, Jr., Miami, Ariz.
Wm. Murdoch, Oilton, Okla.
Mrs. Wm. Murdoch, Oilton, Okla.
J. S. Murphy, Denver, Colo.
S. E. Murphy.
J. M. McLaughlin, Joplin, Mo.
F. X. Nachtmann, St. Louis, Mo.
H. K. Najarian, Fredericktown, Mo.
J. L. Nelson, Baxter Springs.
M. H. Neumann, Carthage, Ill.
Mrs. M. H. Neumann, Carthage, Ill.
L. M. Neumann, Tulsa, Okla.
J. B. Newby, Tulsa, Okla.
Edmund Newton, Minneapolis, Minn.
F. B. Newton and Guest, Carthage, Ill.
Askin Nicholas, Queensland, Australia.
C. Nickle, Tulsa, Okla.
H. E. Nold, Columbus, Ohio.
T. R. Noon, Peru, Ill.
R. V. Norris, Wilkes-Barre, Pa.
H. L. Norton, Globe, Ariz.
T. H. O'Brien, Dawson, New Mex.
Mrs. T. H. O'Brien, Dawson, New Mex.
P. J. O'Gara, Salt Lake City, Utah.
W. H. Ohnsorg, St. Louis, Mo.
Earl Oliver, Tulsa, Okla.
Felipe B. Oréll, Rolla, Mo.
C. T. Orr, Webb City, Mo.
C. L. Orth, St. Louis, Mo.
Mrs. C. L. Orth, St. Louis, Mo.
S. K. Ortt and Guest, Webb City, Mo.
W. A. Overstreet, Joplin, Mo.
E. W. Owen.
Robert L. Owen, Oklahoma.
E. C. Paix, Paris.
L. S. Panyity, Columbus, Ohio.
G. S. Patterson, Edwards, N. Y.
K. A. Pauly, Schenectady, N. Y.
G. W. Paymal, New York, N. Y.
Mrs. G. W. Paymal, New York, N. Y.
J. H. Payne, New York, N. Y.
E. T. Perkins and Guest, Joplin, Mo.
Mrs. E. T. Perkins, Joplin, Mo.
Mrs. F. W. Perkins, Flagstaff, Ariz.
Miss J. W. Perkins, Kansas City, Mo.
Miss M. J. Perkins, Joplin, Mo.
E. S. Perry, Miami, Ariz.
F. P. Peterson, Joplin, Mo.
W. B. Phillips, Houston, Tex.
W. H. Picher, Joplin, Mo.
F. E. Pierce, New York, N. Y.
W. B. Plank, Pittsburgh, Pa.
G. H. Playter, Joplin, Mo.
A. M. Plumb, Platteville, Wis.
C. H. Plumb, Platteville, Wis.

Mrs. C. H. Plumb, Platteville, Wis.
F. B. Plummer, Tulsa, Okla.
W. F. Plummer, Picher, Okla.
H. K. Porter, Newark, N. J.
J. Clarke Porter, St. Louis, Mo.
W. C. Powell, Rolla, Mo.
Sidney Powers, Washington, D. C.
Mrs. Sidney Powers, Washington, D. C.
H. A. Poy, Mascot, Tenn.
H. M. Preston, Tulsa, Okla.
G. Preuss, Calumet, Mich.
S. S. Price, Tulsa, Okla.
C. L. Proctor and Guest, Joplin, Mo.
A. H. Purdue, Nashville, Tenn.
J. S. Qualey, Joplin, Mo.
Victor Rakowsky, Joplin, Mo.
Mrs. Victor Rakowsky, Joplin, Mo.
A. H. Ramage and Three Guests.
E. R. Ramsey, Denver, Colo.
M. S. Randall, Chicago, Ill.
F. L. Ransome, Washington, D. C.
W. A. Rayson.
J. E. Rea, Joplin, Mo.
M. K. Read, Kansas City, Mo.
T. T. Read, New York, N. Y.
W. H. Reber, Rolla, Mo.
Mrs. W. H. Reber, Rolla, Mo.
A. H. Reed, Marion, Ky.
Mrs. A. H. Reed, Marion, Ky.
E. C. Reeder, Chicago, Ill.
Mrs. E. C. Reeder, Chicago, Ill.
H. C. Reimer, New York, N. Y.
A. H. Reller, St. Louis, Mo.
G. W. Repetti, Denver, Colo.
J. W. Richards, Bethlehem, Pa.
G. B. Richardson, Washington, D. C.
T. A. Rickard, San Francisco, Cal.
H. Ries, Ithaca, N. Y.
Ira Rinehart, Tulsa, Okla.
A. E. Ring, Fredericktown, Mo.
W. H. Risher, Huntington, Ark.
J. D. Robertson, St. Louis, Mo.
H. M. Robinson, Washington, D. C.
J. F. Robinson, Miami, Ariz.
Mrs. L. S. Robson, Claremore, Okla.
A. E. Robtaille, Tulsa, Okla.
N. P. Rood, Joplin, Mo.
Mrs. N. P. Rood, Joplin, Mo.
O. D. Royse and Guest, Joplin, Mo.
Otto Ruhl, Joplin, Mo.
Mrs. Otto Ruhl, Joplin, Mo.
J. J. Rutledge, McAlester, Okla.
S. St. Clair, Chicago, Ill.
C. V. Safford, St. Louis, Mo.
G. W. Sapp, Miami, Ariz.
C. F. Sapper, Herculaneum, Mo.
John Schaefer and Guest, Joplin, Mo.
L. R. Scheurer, Rolla, Mo.
L. A. Scholl, Jr.
C. Scholz, Chicago, Ill.
Mrs. C. Scholz, Chicago, Ill.
C. E. Schwarz, Miami, Ariz.
W. W. Seed, Miami, Ariz.
Mrs. W. H. Seed, Miami, Ariz.
E. H. Sellards, Tallahassee, Fla.
C. L. Severy, Tulsa, Okla.
W. B. Shackelford and Guest, Webb City, Mo.
M. Shanfield, Rolla, Mo.
C. W. Shannon, Norman, Okla.
Harvey Shaw.
Mrs. Harvey Shaw.
C. E. Siebenthal, Washington, D. C.
J. H. Simmons, Tulsa, Okla.
Helen Skewes, Tulsa, Okla.
F. M. Smith, Miami, Ariz.
P. S. Smith, Washington, D. C.
Mrs. P. S. Smith, Washington, D. C.
W. A. Smith, Herculaneum.
Mrs. W. A. Smith, Herculaneum.
W. N. Smith, Platteville, Wis.
Mrs. W. N. Smith, Platteville, Wis.
J. J. Smythe, Hannibal, Mo.
A. N. Snietkoff, Russia.
F. W. Sperr, Houghton, Mich.
C. B. Spicer and Guest, Pittsburg, Kan.
H. N. Spicer, New York, N. Y.
Mrs. H. N. Spicer, New York, N. Y.
C. F. Spencer, Pittsburg, Kan.
Mrs. G. C. Spillers, Tulsa, Okla.
E. G. Spilsbury, New York, N. Y.
H. G. Spilsbury, Pittsburgh, Pa.
P. G. Spilsbury, New York, N. Y.
R. W. Sproal, Chicago, Ill.
J. C. Squires and Guest, Carthage, Ill.
H. R. Stahl, Rolla, Mo.
A. C. Starkey, Rolla, Mo.
A. W. Stepanoff, Petrograd, Russia.
John Stevenson, Jr., Sharon, Pa.
Miss J. M. Stevenson, Sharon, Pa.
E. C. Stewart.
Leighton Stewart, New York, N. Y.
R. F. Stewart, Webb City, Mo.
W. M. Stewart, Joplin, Mo.
C. G. Stifel, St. Louis, Mo.
R. K. Stockwell, Salt Lake City, Utah.
Mrs. R. K. Stockwell, Salt Lake City, Utah.
H. H. Stoek, Urbana, Ill.
G. C. Stone, New York, N. Y.
Bradley Stoughton, New York, N. Y.
C. B. Strachan, Mascot, Tenn.
Mrs. C. B. Strachan, Mascot ,Tenn.
R. H. Summer, Wilmington, Del.
Henry Tackmann, Baxter Springs.
J. C. Taylor, New York, N. Y.
J. H. Taylor, Isabella, Tenn.
L. D. Templeton and Two Guests, Joplin, Mo.
A. D. Terrell, Chicago, Ill.
A. C. Terrill, Lawrence, Kan.
Mrs. A. C. Terrill, Lawrence, Kan.
Arthur Thacher, St. Louis, Mo.
Mrs. Arthur Thacher, St. Louis, Mo.
J. E. Thomas, Tulsa, Okla.
Mrs. J. C. Thompson, St. Louis, Mo.
E. E. Thum, New York, N. Y.
L. W. Trumbull, Cheyenne, Wyo.
R. J. Tuthill, Miami, Ariz.
J. A. Udden, Austin, Tex.
G. E. Ude, Morenci, Ariz.
J. R. Underwood, Granby, Mo.

Mrs. J. R. Underwood, Granby, Mo.
H. H. Utley, Florence, Cal.
Mrs. H. H. Utley, Florence, Cal.
M. M. Valerius, Tulsa, Okla.
W. van der Gracht, Tulsa, Okla.
S. G. Victor, Afton, Okla.
R. Vose.
Mrs. R. Vose.
G. R. Wadleigh, Webster Groves, Mo.
F. C. Wallower and Guest.
R. T. Wallace, Bartlesville, Okla.
H. T. Walsh, Chicago, Ill.
Mrs. H. T. Walsh, Chicago, Ill.
G. W. Walton and Guest, Webb City, Mo.
W. W. Wanfler, Webb City, Mo.
G. H. Waring and Guest, Oronogo, Mo.
W. G. Waring and Guest, Webb City, Mo.
J. H. Warner, Butte, Mont.
J. S. Warren, St. Louis, Mo.
S. P. Warren, Joplin, Mo.
J. A. Watson, Blacksburg, Va.
T. L. Watson, Charlottesville, Va.
A. P. Watt, Mine La Motte, Mo.
C. M. Weaver and Guest, Joplin, Mo.
F. M. Webber, St. Louis, Mo.
W. H. Weed, New York, N. Y.
Mrs. W. H. Weed, New York, N. Y.
Mrs. F. E. Weeks, Joplin, Mo.
R. S. Weimer, Rolla, Mo.
L. E. Wemple, St. Louis, Mo.
R. L. Wensley, New York, N. Y.
H. A. Wentworth, Boston, Mass.
W. Y. Westervelt, New York, N. Y.
C. A. Wettingel, St. Louis, Mo.
H. A. Wheeler, St. Louis, Mo.
A. G. White, Washington, D. C.
David White, Washington, D. C.
E. E. White, Ann Arbor, Mich.
H. H. White, Joplin, Mo.
Mrs. H. H. White, Joplin, Mo.
C. C. Whittier, Chicago, Ill.
Mrs. C. C. Whittier, Chicago, Ill.
Edward Wilkins.
S. L. Wilkins and Guest, Oronogo, Mo.
R. D. Willets, Joplin, Mo.
Bruce Williams.
Mrs. Bruce Williams.
D. C. Wise and Guest, Joplin, Mo.
A. W. G. Wilson, Ottawa, Canada.
G. B. Wilson, Racine, Wis.
H. M. Wilson, Pittsburgh, Pa.
M. E. Wilson, Rolla, Mo.
Mrs. M. E. Wilson, Rolla, Mo.
E. A. Wiltsee, San Francisco, Cal.
C. S. Witherell, New York, N. Y.
H. B. Wolcott, Joplin, Mo.
B. L. Wolfe, Lawrence, Kan.
Y. C. Wong, Hunan, China.
W. A. Wood, Joplin, Mo.
R. H. Worcester, St. Louis, Mo.
Mrs. R. H. Worcester, St. Louis, Mo.
W. E. Wrather, Wichita Falls, Tex.
Allen Wright, McAlester, Okla.
I. S. Wright, Hurley, New Mex.
C. Yabe, Japan.
C. M. Young, Urbana, Ill.
H. I. Young, Carterville, Mo.
Mrs. H. I. Young, Carterville, Mo.
C. W. Youngman, New York, N. Y.
E. B. Young, Butte, Mont.
W. G. Brenneke.
B. L. Brown.
P. M. Bruner.
W. E. Bryan.
D. W. Clark.
R. S. Colnon.
C. G. Cox.
W. R. Crecilius.
W. E. Doll.
J. T. Garrett.
R. P. Garrett.
A. L. Grodzensky.
W. A. Heimbuecher.
R. Hellinger.
J. A. Hooke.
J. Hornbrook.
W. W. Horner.
K. Hunter.
E. R. Kinsey.
R. Jackson.
W. A. Layman.
D. R. Luyties.
W. M. Matheny.
J. B. Myers.
R. L. Radcliffe.
W. E. Rolfe.
——Ruckert.
P. Schlingman.
E. J. Spencer.
W. Stoecker.
C. M. Talbert.
——Uhrig.
J. D. Von Maur.
E. E. Wall.
J. W. Woermann.
J. L. Woodress.

Technical Program of St. Louis Meeting

Monday Morning, Oct. 8, 1917.—One session was under the auspices of the Committee on Milling, and O. M. Bilharz presided.

The following papers were presented by their authors or authors' representatives:

The Hancock Jig in the Concentration of Lead Ores. By Harold Rabling. (Written discussion by A. P. Watt.)

Concentration Practice in Southeast Missouri. By A. P. Watt. (Discussed by O. M. Bilharz, E. Gayford, E. R. Ramsey, L. A. Delano, H. S. Coe, R. B. T. Kiliani, A. P. Watt, F. D. Druding, S. A. Ionides.)
The Milling Practice of the St. Joseph Lead Co. By L. A. Delano. (Discussed by L. A. Delano, O. M. Bilharz, E. Gayford, A. P. Watt.)
Ore Dressing Practice in the Joplin District. By C. A. Wright. (Discussed by O. M. Bilharz, H. A. Wheeler. Written discussion by F. H. Gartung.)
A Standard Screen Scale for Testing Sieves.

The following paper was presented by title:

A Uniform Sizing Diagram from Different Screen Standards. By John Randall.

A simultaneous session was held under the auspices of the Committee on Coal, and Carl Scholz presided.

The following papers were presented by their authors or authors' representatives:

The Effect of Anti-Friction Bearings on the Haulage of a Coal Mine. By P. B. Liebermann. (Discussed by P. B. Liebermann, F. F Jorgensen, T. G. Clagett.)
Coal Wastage. By Francis S. Peabody. (Discussed by Carl Scholz, R. V. Norris, J. A. Udden, Edwin Ludlow, H. H. Stoek, R. W. Hunt, E. T. Conner, T. H. Clagett, F. W. Sperr, F. F. Jorgensen, R. D. Hall, E. A. Holbrook.)
Steam-shovel Mining of Bituminous Coal. By H. H. Stoek. (Written discussion by J. B. Warriner.)
Merit Rating of Coal Mines under Workmen's Compensation Insurance. By E. C. Lee. (Discussed by H. M. Wilson.)
The Coal Industry of Illinois. By C. M. Young. (Discussed by Carl Scholz, John Stevenson, Jr., F. F. Jorgensen, H. M. Wilson.)

Tuesday Morning, Oct. 9, 1917.—One session was under the auspices of the Committee on Lead, and Arthur Thacher presided.

The following papers were presented by their authors or authors' representatives:

The Media Mill, Webb City, Mo. By H. B. Pulsifer. (Discussed by J. J. McLellan, Arthur Thacher, O. M. Bilharz, S. J. Jennings.)

The following papers were presented by title:

Salt in the Metallurgy of Lead. By Oliver C. Ralston, Clyde E. Williams, Marvin J. Udy, G. J. Holt. (Discussed by E. L. Blossom, Arthur Thacher; written discussion by C. L. Larson, Stuart Croasdale.)
The Tredinnick-Pattinson Process. By William E. Newman.
Lead Mining and Smelting at Galetta, Ont. By William E. Newman.
The Metallurgy of Lead Ores in the Lower Mississippi Valley. By Herman Garlichs.

A simultaneous session was held under the auspices of the Committee on Petroleum and Gas, and Captain A. F. Lucas presided.

The following papers were presented by their authors or authors' representatives:

The Practical Value of Oil and Gas Bureaus. By W. G. Matteson. (Discussed by I. N. Knapp, L. L. Hutchison, A. F. Lucas, H. M. Ami, W. van der Gracht, Dorsey Hager, W. E. Wrather; written discussion by I. N. Knapp, William Kennedy, E. G. Woodruff.)
A Review of the Exploration at Belle Isle, Louisiana. By A. F. Lucas. (Written discussion by I. N. Knapp, J. A. Udden, E. T. Dumble, W. van der Gracht, W. E. Wrather, F. B. Plummer.)

Tuesday Evening, Oct. 9, 1917.—This session comprised papers on miscellaneous subjects, and J. W. Malcolmson presided.

The following papers were presented by their authors or authors' representatives:

Mine Models. By H. H. Stoek. (Discussed by W. R. Crane, F. W. Sperr, E. B. Young.)
Increasing Dividends through Personnel Work. By T. T. Read. (Discussed by C. W. Goodale, H. M. Wilson, W. Y. Westervelt, R. M. Catlin, J. A. Ede, T. T. Read, C. M. Haight.)
Resistance of Artificial Mine Roof Supports. By W. Griffith. (Discussed by E. T. Conner, W. R. Crane, H. M. Wilson.)
The History and Legal Phases of the Smoke Problem. By Ligon Johnson. (Discussed by P. J. O'Gara.)

The following papers were discussed by title:

Comparative Tests of Hammer Drill Bits. By C. R. Forbes and J. C. Barton.
Mining Methods of the American Zinc Co. of Tennessee. By H. A. Coy and H. B. Henegar.
Influence of Base Metals in Gold Bullion Assaying. By Frederic P. Dewey.
Graphic Solutions of Some Compressed-Air Calculations. By C. W. Crispell.
Tests on the Hardinge Conical Mill. By Arthur F. Taggart. (Discussed by R. B. T. Kiliani.)
The Enrichment and Segregation of Mill Tailings for Future Treatment. By F. E. Marcy.
Methods for Determining the Capacities of Slime-Thickening Tanks. By R. T. Mishler. (Written discussion by H. S. Coe.)
A Study of the Microstructure of Some Clays in Relation to Their Period of Firing. By H. Ries and Y. Oinouye.
Zinc Dust as a Precipitant in the Cyanide Process. By W. J. Sharwood.
Experiments in the Recovery of Tungsten and Gold in the Murray District, Idaho. By R. R. Goodrich and N. E. Holden.

Wednesday Morning, Oct. 10, 1917.—This session was under the auspices of the Committee on Petroleum and Gas, and I. N. Knapp presided.

The following papers were presented by title:

A Feasible Plan for Gaging Individual Wells. By Roswell H. Johnson and W. E. Bernard. (Discussed by I. N. Knapp, C. P. Bowie, D. Hager, E. T. Dumble.)
Geosynclines and Petroliferous Deposits. By Marcel R. Daly. (Written discussion by W. van der Gracht, F. G. Clapp.)
Funnel and Anticlinal-Ring Structure Associated with Igneous Intrusions in the Mexican Oil Fields. By V. R. Garfias and H. J. Hawley. (Discussed by E. T. Dumble, W. E. Wrather, I. N. Knapp, J. A. Udden, F. B. Plummer, W. R. Crane, L. L. Hutchison, W. van der Gracht.)

Wednesday Afternoon, Oct. 10, 1917.—One session was under the auspices of the Iron and Steel Committee, and E. Gybbon Spilsbury presided.

The following papers were presented by their authors or authors' representatives:

Manganiferous Iron Ores of the Cuyuna District, Minn. By E. C. Harder. (Discussed by E. Gybbon Spilsbury, Edmund Newton.)
The Tayeh Iron Ore Deposits. By Chung Yu Wang.

The following papers were presented by title:

The Ferrous Iron Content and Magnetic Susceptibility of Some Artificial and Natural Oxides of Iron. By R. B. Sosman and J. C. Hostetter.
Zonal Growth in Hematite, and Its Bearing on the Origin of Certain Iron Ores. By R. B. Sosman and J. C. Hostetter.
The Supposed Reversal of Inheritance of Ferrite Grain Size from that of Austenite. By Henry M. Howe. (Written discussion by W. E. Ruder.)
Some Unusual Features in the Microstructure of Wrought Iron. By Henry S. Rawdon. (Written discussion by Henry Fay.)

A simultaneous session was held under the auspices of the Committee on Ore Deposits, and H. A. Buehler presided.

The following papers were presented by their authors or authors' representatives:

The Pyritic Deposits near Roros, Norway. By H. Ries and R. E. Somers.
The Sulphur Deposits in Culberson Co., Texas. By William B. Phillips. (Discussed by S. J. Jennings, W. B. Phillips, E. T. Lednum, L. W. Trumbull.)
Ore Deposits of the Boulder Batholith of Montana. By Paul Billingsley and J. A. Grimes. (Written discussion by J. B. Hastings, W. E. Gaby.)
A New Silicate of Lead and Zinc. By P. A. van der Meulen.
Geology and Mineral Deposits of the Ozark Region. By H. A. Buehler.

The following papers were presented by title:

Exploration of Metalliferous Deposits. By W. H. Emmons.
The Effects of Cross Faults on the Richness of Ore. By E. K. Soper.
The Replacement of Sulphides by Quartz. By H. N. Wolcott.

Thursday Afternoon, Oct. 11, 1917.—This session was under the auspices of the Committee on Zinc, and George C. Stone presided.

The following papers were presented by their authors or authors' representatives:

The Zinc Ores of the Joplin District. By W. Geo. Waring. (Discussed by J. W. Richards, G. C. Stone. Written discussion by V. H. Gottschalk.)
Development and Underground Mining Practice in the Joplin District. By H. I. Young. (Discussed by F. W. Sperr, H. I. Young, J. A. Ede.)
Oxide of Zinc. By Geo. C. Stone. (Discussed by F. D. James, E. G. Spilsbury, G. C. Stone. Written discussion by L. E. Wemple.
Some Economic Factors in the Production of Electrolytic Zinc. By R. G. Hall. (Discussed by Lawrence Addicks, C. E. Schwartz, J. W. Richards, S. J. Jennings, G. C. Stone.)
Zinc Mining at Franklin, N. J. By C. M. Haight and B. F. Tillson. (Written discussion by Robert Peele.)
Characteristics of Zinc Deposits in North America. By Frank L. Nason. (Discussed by H. A. Buehler. Written discussion by J. T. Boyd.)
Palmerton Zinc Refractories. By C. P. Fiske. (Discussed by H. Ries, C. P. Fiske, F. E. Pierce.)
Zinc Burning as a Metallurgical Process. By W. R. Ingalls. (Discussed by G. C. Stone.)

The following paper was presented by title:

The New Jersey Zinc Co.'s Franklin Laboratory. By D. Jenkins.

Friday Evening, Oct. 12, 1917.—This session was arranged by the Committee on Petroleum and Gas, with David White as chairman.

The following papers were presented by title:

A Few Notes on the Future Work of the Petroleum Geologist in the Mid-Continent Oil Fields. By Dorsey Hager.
Geologic Structure in the Cushing Oil and Gas Field, Oklahoma. By Carl H. Beal.
Granite in Kansas Wells. By Park Wright.
Review of Present Knowledge Regarding the Petroleum Resources of South America. By F. G. Clapp. (Discussed by I. C. White.)
The Estimation of Petroleum Reserves. By Robert W. Pack.
The Southern Extremity of the "Clinton" Gas Pools in Ohio. By L. S. Panyity.
Relation of Sulphur to Variation in the Gravity of California Petroleum. By G. Sherburne Rogers.

PAPERS

A Study of the Silica Refractories*

BY J. SPOTTS McDOWELL, B. S.†

(New York Meeting, February, 1917)

TABLE OF CONTENTS

* An investigation conducted by the author at the laboratories of the Massachusetts Institute of Technology from October, 1915, to February, 1916, and published with the consent of Professor H. O. Hofman, Acting Head of the Department of Mining and Metallurgy. The supplementary report represents work done later while associated with the Harbison-Walker Refractories Co. at Pittsburgh, Pa.

† Research Department, Harbison-Walker Refractories Co., Pittsburgh, Pa.

Acknowledgments

To a very considerable degree I am indebted to the kindness of others for the material herein presented. The subject was suggested to me by Kenneth Seaver, Chief Engineer of the Harbison-Walker Refractories Co., Pittsburgh, Pa., and represents, in fact, the continuation of an investigation begun by him.*

The materials studied were furnished by the Harbison-Walker Refractories Co., and the test brick were manufactured at the Hays Station plant of that company under the supervision of R. H. H. Pierce, Chief Chemist. The conditions under which the test brick were made were devised by members of the Harbison-Walker Refractories Co.'s engineering department.

Whatever value the microscopic determinations may possess is due entirely to the careful instruction and kindly counsel of Professor C. H. Warren of the Massachusetts Institute of Technology. I am particularly grateful to Victor Dolmage, who was extremely generous with his time in assisting with some of these determinations. The photomicrographs are the work of W. L. Whitehead. The crushing and cross-breaking tests were made with the assistance of I. H. Cowdrey of the Mechanical Engineering Department of the Institute of Technology. I wish to express my thanks to all those named, as well as to others who have rendered assistance in various ways.

Introduction

The magnitude of the thermal expansion of silica brick, and its inability to withstand rapid temperature changes, present problems of considerable importance in the manufacture of siliceous refractories.

In the study represented by the experimental data herein described, these problems are considered from the point of view afforded by the results of recent investigators: Fenner and his associates of the Geophysical Laboratory, Washington, D. C., who have established the stability relations of the silica minerals, and Endell and co-workers in Berlin, who have studied the changes brought about in the constitution of siliceous refractory materials upon the application of heat.

In the process of manufacture, the quartzite from which the brick is made changes in part into cristobalite and tridymite. The attempt is herein made to determine by microscopic methods the degree of transformation in various specimens of test bricks manufactured at slightly varying temperatures of burning, with varying coarseness of grain, and to determine the effect of repeated burning. The effect of variations

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* Kenneth Seaver: Manufacture and Tests of Silica Brick for the Byproduct Coke Oven, *Trans.* (1916), **53**, 125–139.

in these conditions upon crushing strength and modulus of rupture has also been considered.

Coincidently, a study has been made, from the available literature, of the properties of the silica minerals and the silica refractories. From the knowledge so gained, combined with the experimental data regarding the constitution of the test bricks, deductions have been made as to the conditions of manufacture under which silica brick of decreased thermal expansion and increased power to withstand rapid temperature changes might conceivably be produced. It is hoped that these deductions may serve to indicate possible starting points for future investigations.

The first part of this report comprises data compiled from various publications. Although much of this has no bearing upon the specific problem in hand, it is believed to be of interest in any thorough study of the properties of siliceous refractory materials.

THE SILICA MINERALS

Stability Relations

The crystal modifications of silica important in this connection are quartz, tridymite and cristobalite, each of which possesses α and β phases. Any one of these three modifications may be converted into either of the others by appropriate heat treatment. The formation of tridymite seems always to require the presence of a flux or catalyzer, while cristobalite is formed in the absence of a catalyzer. The inversion temperatures[1] are as follows:

870°C $\pm$ 10 quartz $\rightleftharpoons$ tridymite
1,470° $\pm$ 10 tridymite $\rightleftharpoons$ cristobalite
575° α quartz $\rightarrow \beta$ quartz; 570° β quartz $\rightarrow \alpha$ quartz
117° α tridymite $\rightarrow \beta_1$ tridymite; 163° β_1 tridymite $\rightarrow \beta_2$ tridymite; reversions on cooling not sharp.
274.6° − 219.7° α cristobalite $\rightarrow \beta$ cristobalite
240.5° − 198.1° β cristobalite $\rightarrow \alpha$ cristobalite

At ordinary temperatures each mineral exists only in the α phase, changing into the β phase as the temperature is raised. The α to β transformations are all rapid, while considerable time is required for the transformation of one mineral into another.

In the presence of a flux, when any form of silica is heated a sufficient length of time below 870°, quartz is always formed;[2] between 870° and 1,470° tridymite, and from 1,470° upward to the melting point of silica, cristobalite. *Heated without a flux*, the inversion from quartz to tridymite does not occur, but the transformation is direct to cristobalite,

[1] C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4 (1913), **36**, 383.

[2] *Ibid.*, 358.

and the temperature of incipient formation of cristobalite is depressed to about 1,250°; under the same conditions, tridymite is converted into cristobalite, with the inversion temperature raised to about 1,570°. Fenner's experiments on quartz powder heated without a flux gave the following results:

108 hr. at 1,250°: a very small per cent. of cristobalite formed.
90 hr. at 1,360°: resultant product two-thirds cristobalite, one-third quartz.
1 hr. at 1,570°: transformation into cristobalite nearly complete.

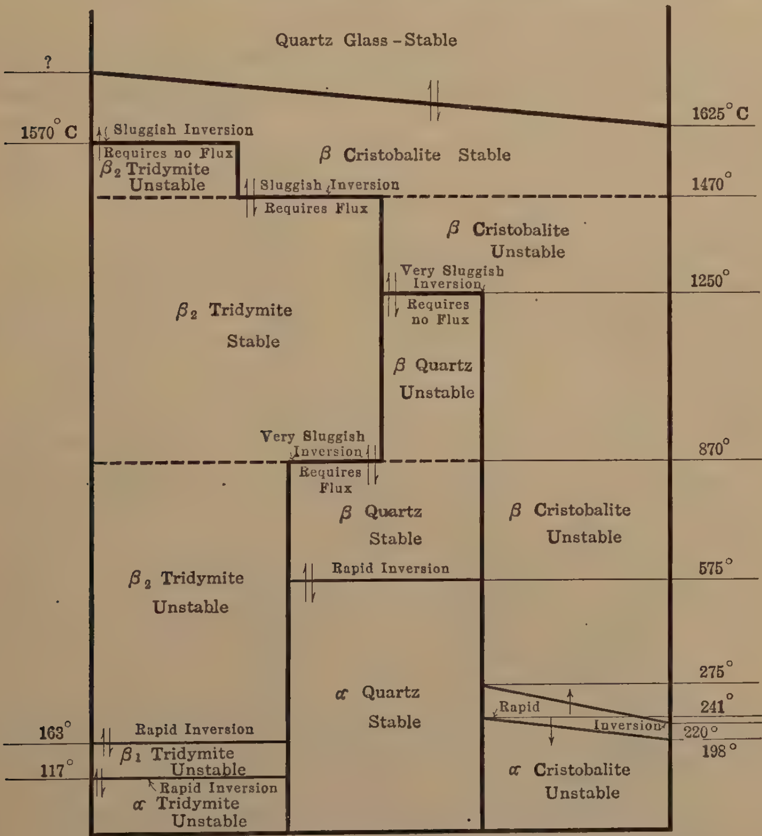

FIG. 1.—STABILITY RELATIONS OF THE SILICA MINERALS.

Quartz glass heated above 1,200° becomes gradually converted into cristobalite.[3] The percentages of cristobalite formed under various conditions of heating are shown in the accompanying table, taken from the work of Rieke and Endell.

[3] Rieke and Endell: Devitrification of Quartz Glass, *Silikat Zeitschrift* (Jan., 1913), 1, No. 1, 8.

TABLE 1.—*Cristobalite Formed by Heating Quartz Glass under Various Conditions (Rieke and Endell)*

Time of Heating, Hours	Temperature				
	1,200°	1,300°	1,400°	1,500°	1,600°
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
½	—	—	5	34	60
1	0	2	17	50	90
2	0	5	25	100	
4	0	10	—		

Quartzite heated three to five times to 1,450° in a porcelain kiln is converted chiefly into cristobalite, with here and there scattered residual

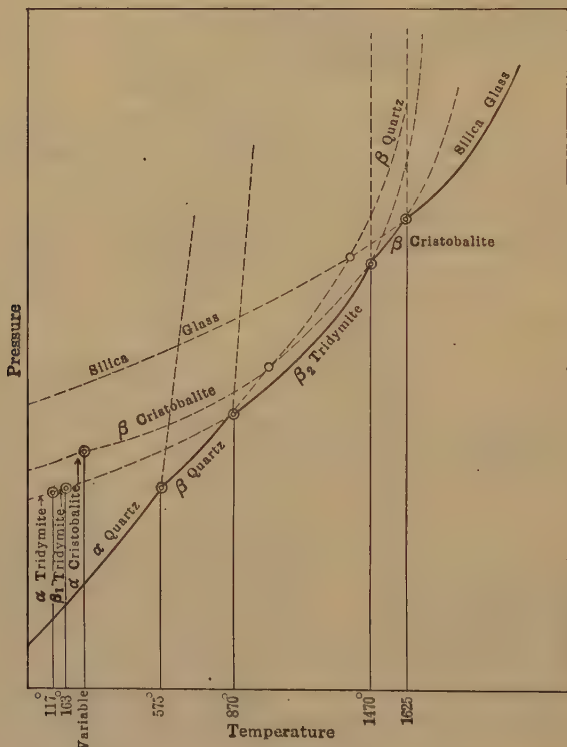

FIG. 2.—STABILITY RELATIONS OF THE SILICA MINERALS (AFTER FENNER).

quartz grains; after 10 burnings at 1,450° wedge-shaped twin crystals of tridymite begin to appear.[4] Silica brick, which consists principally of

[4] K. Endell: *Stahl und Eisen* (Nov., 1913), **33**, 1855.

cristobalite when placed in use, gradually changes at high temperatures in the furnace to tridymite, as explained later on in this paper.

The stability fields and existence ranges of the various forms of silica are shown in Fig. 1; an equilibrium diagram (after Fenner) in Fig. 2.

TABLE 2.—*Optical Properties**

Composition	Crystal System	Crystal Habit	Elongation	Optical Orientation	nNa		Optical Character	Remarks
					α	γ		
α Quartz.....	Hexagonal Trapezohedral Tetartohedral	Pyramidal	Y	$c = \gamma$	1.544	1.553	+	Stable below 575°.
β Quartz......	Hexagonal Trapezohedral Hemihedral	Pyramidal	Y	$c = \gamma$			+	Stable between 575° and 870°. Inverts on cooling to α quartz.
α Tridymite...	Orthorhombic	Thin pseudo-hexagonal plates			1.469	1.473	+	Exists below 117°. Usually finely intergrown aggregates. Optic axial angle large.
β Tridymite...	Hexagonal	Hexagonal plates		$c = \gamma$			+	Exists only above 117°. Inverts on cooling to α tridymite.
α Cristobalite.	Probably tetragonal	Equaint grains		$c = \alpha$	1.484	1.487	−	Exists below 275°.
β Cristobalite..	Isometric	Equaint grains						Exists only above 198°. Inverts on cooling to α cristobalite.

* C. N. Fenner: *American Journal of Science*, ser. 4, vol. 36, pp. 351–356 (1913).
N. L. Bowen: *American Journal of Science*, ser. 4, vol. 38, p. 245 (1914).
Rankin and Wright: *American Journal of Science*, ser. 4, vol. 39, pp. 4 and 74 (1915).

Tridymite crystals usually occur as crystalline aggregates of random orientation, as broad, thin hexagonal plates, which appear as needles or laths when turned on edge, and as wedge-shaped twin crystals. The hexagonal plates of usual thinness appear perfectly isotropic when lying on the base; the needle or lath-like sections have weak double refraction, parallel extinction and negative elongation.

Cristobalite does not develop a well-defined crystal form but usually occurs as minute branching skeleton crystals, with the branches often showing octahedral terminal caps. The higher temperature form of cristobalite is isometric; on cooling it becomes weakly birefringent. The birefringence in thin sections is barely discernible with the sensitive tint plate.

Table 3.—*Density*

	Specific Gravity	
Quartz........................	2.65	
Tridymite......................	2.270	Artificial tridymite (Fenner).
	2.28	Natural tridymite (Mallard).
	2.32	(Endell).
Cristobalite....................	2.333	Artificial cristobalite (Fenner).
	2.34	Natural cristobalite (Mallard).
	2.33	(Endell).
Cristobalite....................	.....	At 1,500° has same density as quartz glass; at 300°, slightly greater density than quartz glass (Rieke and Endell).
Quartz glass....................	2.21	(Dana, Endell).
	2.194	(Schwarz).

Melting Points

"It appears that the fusing point of quartz is lower than 1,470°, but that at this temperature it passes into cristobalite almost as rapidly as it melts. The fusing point of tridymite should lie between those of quartz and cristobalite."[5] "It is clear that the indicated melting point of cristobalite must be higher than 1,625°, the value found by Fenner, higher than 1,685° even, the value found by Endell and Rieke.[6] . . . . . As Dr. Fenner has suggested, cristobalite may have a variable molecular constitution and a similarly variable melting point according to the conditions under which it was formed."[7] Kanolt[8] finds that pure silica flows distinctly at 1,750°, which is, therefore, the apparent melting point and higher than the true melting point. Quartz glass softens at 1,500° and melts between 1,700° and 1,800°.[9]

Thermal Expansion

The results of LeChatelier's[10] experiments are shown in Fig. 3; those of Day, Sosman and Hostetter[11] in Fig. 4.

[5] C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4 (1913), **36**, 383.

[6] Endell and Rieke: *Zeitschrift für Anorganische Chemie.* (1913), **79**, 239–259.

[7] N. L. Bowen: Ternary System: Diopside-Forsterite-Silica, *American Journal of Science*, ser. 4 (1914), **38**, 218.

[8] C. W. Kanolt: Melting Point of Fire-Bricks, *U. S. Bureau of Standards, Technologic Paper* No. 10 (1912), 14.

[9] A. E. Marshall: Fused Silica Ware, Its Manufacture, Properties and Uses, *Metallurgical and Chemical Engineering* (Apr., 1912), **10**, 248–249.

[10] LeChatelier: La Silice, *Revue Universelle des Mines*, ser. 5, (1913), **1**, 90.

[11] Day, Sosman and Hostetter: Determination of Mineral and Rock Densities at High Temperatures, *American Journal of Science*, ser. 4 (1914), **37**, 1–39.

LeChatelier gives the mean linear expansion of quartz glass as $0.0_6 7$; Kaye[12] gives the following figures: 0° to 30°C., $0.0_6 42$; 30° to 100°, $0.0_6 53$; 100° to 500°, $0.0_6 58$; 500° to 900°; $0.0_6 50$; 900° to 1000°; $0.0_6 80$. Quartz glass, partly devitrified by long-continued heating at a high tem-

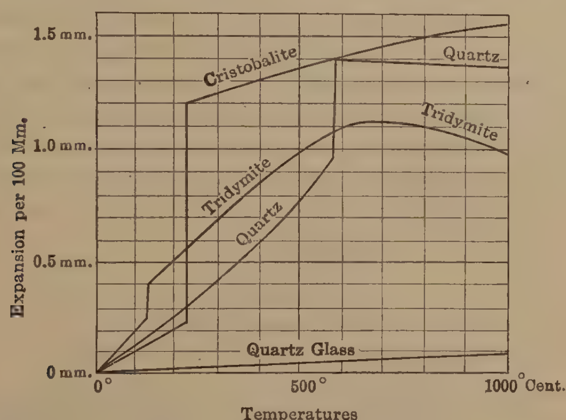

FIG. 3.—RESULTS OF LE CHATELIER'S EXPERIMENTS IN THERMAL EXPANSION.

perature, so that it contains much β cristobalite, if cooled rapidly to 300° remains completely clear and transparent, and but few cracks appear. When further cooled, however, to 230°, the sudden formation of numerous

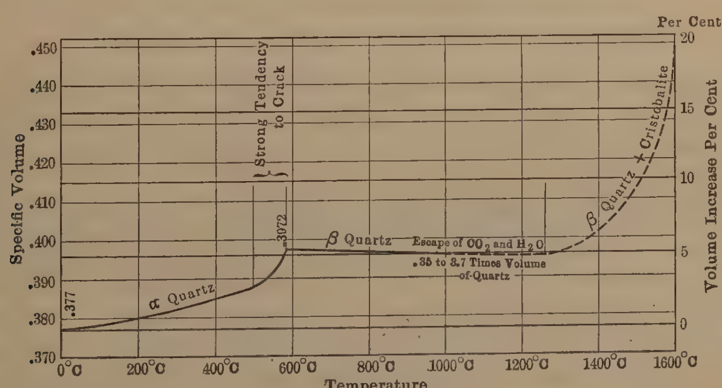

FIG. 4.—THERMAL EXPANSION OF QUARTZ.

fine cracks causes the glass to become white and opaque, coincident with the inversion of β to α cristobalite. The specific volume and coefficient of expansion of β cristobalite are, therefore, nearly the same as those of quartz glass; the presence of a few cracks at 300° indicate that the

[12] G. W. C. Kaye: Expansion and Thermal Hysteresis of Fused Silica, *Philosophical Magazine*, ser. 6 (Oct., 1910), **20**, 718–728.

cristobalite probably has a slightly greater coefficient of expansion than the glass.[13]

Solubilities

Rammelsberg,[14] and Lange and Milberg[15] studied the solubility of quartz, opal and amorphous silica in alkaline solutions, but not that of tridymite or cristobalite. Cramer found the solubility in KOH of quartzite burned once in a porcelain kiln to be 41 per cent.; burned 10 times, 70 per cent. After the first burning, this material probably consisted of a mixture of cristobalite and quartz; after the tenth burning, of cristobalite and tridymite, with a small amount of quartz.[16] Schwarz[17] obtained the results shown in Table 4 on boiling powders with grains approximately 0.04 mm. in diameter.

Table 4

Reagent	Solubility of				Time, Hours
	Quartz, Per Cent.	Tridymite, Per Cent.	Cristobalite, Per Cent.	Amorphous Silica, Per Cent.	
5 per cent. Na_2CO_3 solution	2.11	2.77	....	....	½
5 per cent. HF solution...	30.17	76.30	74.3	96.6	½
1 per cent. HF solution...	5.20	20.30	25.8	52.9	1

Identification of Cristobalite and Tridymite

The method of identification most readily applied is by determination of the indices of refraction by the immersion method, described later on in this paper. Tridymite may also be recognized by its characteristic needles, laths and wedge-shaped twin crystals.

The methods of Rieke and Endell,[18] who have gone into this subject with great care, are given in the following paragraphs. Specific-gravity determinations were of little value in the case of quartz converted by heating into cristobalite or tridymite, for several reasons. The specific gravities of the two last-named are not greatly different; the material always contains some unaltered quartz; and the end products are so

[13] Rieke and Endell: Devitrification of Quartz Glass, *Silikat Zeitschrift* (Jan., 1913), **1**, No. 1, 6.

[14] *Annalen der Physik und Chemie, Poggendorf* (1861), **112**, 177.

[15] *Zeitschrift für Angewandte Chemie* (1897), 393 and 425.

[16] K. Endell: *Stahl und Eisen* (Oct., 1913), **33**, 1770 *et seq.*

[17] R. Schwarz: Chemical Relations of the Various Modifications of SiO_2, *Zeitschrift für Anorganische Chemie* (1912), **76**, 424.

[18] Rieke and Endell: Volume Change of Some Ceramic Raw Materials on Burning, *Silikat Zeitschrift* (Apr., 1913), **1**, No. 4, 67, 85.

filled with microscopic cracks that the results of specific-gravity determinations are always low.

These fine cracks interfered with microscopic investigations. On account of total reflection at the borders, according to Rieke and Endell, the greater part of the end product appears isotropic, and the birefringence can be detected only upon very careful observation.

By the use of the thermal microscope, it was seen that cristobalite becomes isotropic at 225° to 230°. The effect was particularly evident in cristobalite derived from quartz glass or from quartz heated above 1,600°; it could not be used as a means of identification in the case of cristobalite formed from quartz below 1,450°, which, on account of

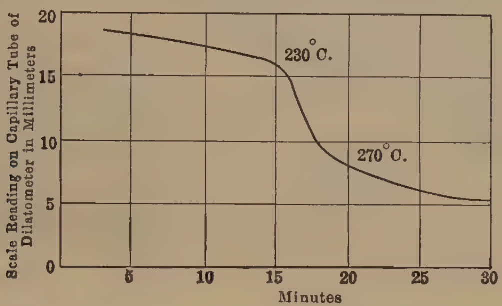

FIG. 5.—CURVE SHOWING EFFECT OF HEAT ON A SPECIMEN OF SILICA BRICK COMPOSED ESSENTIALLY OF CRISTOBALITE.

the cracks above mentioned, appears nearly isotropic. The change in volume of cristobalite at 230° is considerable. For that reason, quicksilver dilatometer observations offered a sure way of distinguishing cristobalite from tridymite. A specimen of silica brick, composed essentially of cristobalite, on heating[19] gave the curve shown in Fig. 5.

Rieke and Endell have designated the sudden clearing up shown by cristobalite heated carefully over the flame at about 230°, as the "cristobalite reaction." This effect can easily be observed with the naked eye in the case of cristobalite derived from quartz glass.

THE SILICA REFRACTORIES

RAW MATERIAL

American Deposits

The raw material used for the manufacture of high-grade American silica brick is a true quartzite; the deposits most extensively used are described by Seaver.[20] Named in the order of their importance, they

[19] *Stahl und Eisen* (Nov., 1913), **33**, 1856, Fig. 4.

[20] K. Seaver: Manufacture and Tests of Silica Brick for the By-Product Coke Oven, *Trans.* (1916), **53**, 125–139.

are the so-called Medina or Tuscarora sandstone of Huntingdon and Blair Counties, Pennsylvania, the Baraboo quartzite of the Devil's Lake

FIG. 6.—GANISTER FLOE IN THE GORGE OF THE JUNIATA RIVER, NEAR MOUNT UNION, HUNTINGDON CO., PENNSYLVANIA.

FIG. 7.—GANISTER FLOE ON THE NORTH SIDE OF THE JUNIATA RIVER, NEAR MOUNT UNION, HUNTINGDON CO., PENNSYLVANIA.

region, Wisconsin, and the deposits of eastern Alabama and of Colorado.

Where quarried for refractory purposes, the Medina is white, hard

and resistant. Its steeply pitching beds form the crests of many of the mountains of central Pennsylvania, and the steep slopes below the summit are covered by great bodies of talus, composed of blocks of the rock, varying in size from pieces weighing a few pounds to those weighing many tons. The talus slopes sometimes cover areas of a thousand acres or more where the beds have been cut through by streams. For the manufacture of silica brick, most of the rock used is taken from these bodies of loose rock, locally called "ganister floes," although rock from the solid measures is sometimes used.

Fig. 6 and 7 are pictures of ganister floes near Mt. Union, Huntingdon Co., Pa., where the Juniata River cuts through Jack's Mountain.

TABLE 5.—*Typical Analyses**

Constituent	Medina Quartzite from Pennsylvania	Baraboo Quartzite	Alabama Quartzite
SiO_2	97.80	97.15	97.70
Al_2O_3	0.90	1.00	0.96
Fe_2O_3	0.85	1.05	0.80
CaO	0.10	0.10	0.05
MgO	0.15	0.25	0.30
Alkalies	0.40	0.10	0.31

* K. Seaver: *Op cit.*, p. 127–128.

Properties Determining Usability

Not all pure quartzites are adapted to the manufacture of refractories. For refractory purposes three properties are to be considered: mechanical strength, melting point and behavior on burning. Experience shows that to produce a physically strong brick the stone should be hard, dense, and give splintery, angular fragments of non-uniform size and shape on crushing; one that crushes down to rounded grains can not be used. The strength of the brick and its melting point are conditioned by the analysis of the rock. The silica content may vary from 96 to 98 per cent.; if less silica is present the fusion point is lowered. Attempts to use pure quartzite of 99 per cent. SiO_2 have not met with success. The presence of Al_2O_3 and Fe_2O_3 is necessary to form a bond; their combined percentage is usually about 1.75 per cent., and should not exceed 2.5 per cent. The amount of alkalies should be less than 0.5 per cent.

The melting interval of the stone depends upon its content of bases. For good quartzite it is usually considered to be cone 35 to 36; that is, 1,755 to 1,775°C.[21] Endell[21] obtained the following results, measuring temperatures by means of an iridium-iridium-ruthenium thermoelement

[21] K. Endell: *Stahl und Eisen* (Nov., 1913), 33, 1857.

in the calibration of which the melting points of gold, palladium and platinum were taken as 1,063°, 1,549° and 1,755°, the values determined by Day and Sosman:[22]

At 1,650°C., fine powder of good quartzite sintered together; pea-sized pieces showed no signs of fusion.

At 1,690°C., fine powder of good quartzite melted together into drops; the pea-sized pieces were superficially melted with visible rounding of the edges.

At 1,710°C., little pieces of pure Brazilian quartz 1 to 2 mm. in cross-section were melted round after 10 min. heating; a piece of ¼-in. cross-section had strongly rounded edges. Pea-sized pieces of good quartzite were strongly vitrified and showed fully rounded edges.

Expansion on Heating[23]

Chemical analysis alone does not afford an adequate index to the behavior of a quartzite on heating. Material suitable for refractory purposes does not weaken or crack perceptibly under the influence of high temperatures, and shows considerable volume increase after a first burning, retaining essentially the same volume later after repeated burning. Unsuitable quartzites, notably the pure, coarsely crystalline variety, become badly cracked or completely disintegrated on heating; they usually expand but little on a first burning and considerably thereafter.

TABLE 6

Material	Volume Increase after Burns				
	1	2	3	4	5
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Good quartzite (average of three samples)......................	9.4	10.7	11.0	11.3	11.5
Quartzite unsuitable for refractory purposes (average of three samples)	5.3	6.5	6.5	9.7	10.7
Melting point of all samples cone 35 to 36.					

In Cramer's[24] experiments, hard, fine-grained quartzites, which did not weaken in the heat treatment, gave the greatest expansion on the first burn. The volume increase of one specimen of good quartzite heated

[22] Day and Sosman: *Zeitschrift für Anorganische Chemie* (Aug., 1911), 72, 1–10.

[23] F. T. Havard: *Refractories and Furnaces*, 36. McGraw-Hill Book Co., New York, 1912.

[24] *Stahl und Eisen* (July, 1901), 21, 772; *Tonindustrie Zeitung* (1901), 25, 864.

to cone 16 (1,460° ±) was 17.1 per cent. after the first burn, 22 per cent. after the sixth. This enlargement was due to two factors: (1) an increase in porosity and (2) an expansion of the mineral itself, resulting from the gradual transformation of the quartz into cristobalite and tridymite. Only the latter effect is shown in the figures of Table 6, calculated from specific gravities of samples of quartzite heated repeatedly to 1,450° in a porcelain kiln.[25]

Texture

The insufficiency of analyses and melting-point determinations to indicate the value of a given quartzite for refractory purposes led Wernicke and Wildschrey[26] to seek an explanation in extensive microscopic studies of the texture of the rock. Their conclusions are as follows:

Typical quartzites, consisting principally of differently oriented intergrown quartz grains fairly uniform in size, metamorphosed quartzites, or those showing undulatory extinction under the microscope, can not be used. Good quartzites consist of quartz grains, mostly rounded, in a groundmass or cement of amorphous silica or crypto-crystalline quartz. No muscovite was found in such quartzites. Since expansion upon heating presents no difficulties in the case of quartzites containing a cement, only their melting point need be considered. They expand without cracking and acquire nearly their whole expansion on the first burn. In consequence of the fineness of division of the impurities in the cement and the slight sintering caused thereby, they are much stronger after burning than the typical quartzites in which the impurities are not so finely divided. The latter crack on burning and attain their complete expansion more slowly.

Grum-Grzimailo,[27] on the other hand, expresses the opinion that any quartzite with more than 94 to 95 per cent. SiO_2 may be used. He states that the purer the material and the larger the single quartz grains the longer must be the time of burning; the finer the quartz and its impurities the more rapidly will the required changes take place on heating.

Wernicke and Wildschrey's views are not supported by microscopic examinations of the most important American quartzites as shown later in this paper.

Silica Brick

The material herein discussed is termed "silica brick" in American practice, and is made from quartzite of 96 to 98 per cent. SiO_2 content to

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[25] Calculated by the writer from data of K. Endell: *Stahl und Eisen* (Oct., 1913), **33**, 1774.

[26] Wernicke and Wildschrey: Quartzite and Its Application in the Refractories Industry, *Tonindustrie Zeitung* (1910), **34**, 688, 723 and 768.

Wernicke: Quartzite and Silica Brick, *Stahl und Eisen* (Nov. 1913), **33**, 1860.

[27] *Stahl und Eisen* (1911), **31**, 225.

which about 2 per cent. of lime is added to serve as a bond. It is not to be confused with the so-called "quartzite" brick, which contains much less silica, and in which fire clay is the bonding material.

Manufacture

The processes of manufacture are discussed by Seaver,[28] Hofman,[28] and Havard.[28] After grinding in a wet pan, where the lime is added, the charge is molded into brick, dried, and burned in a down-draft kiln.

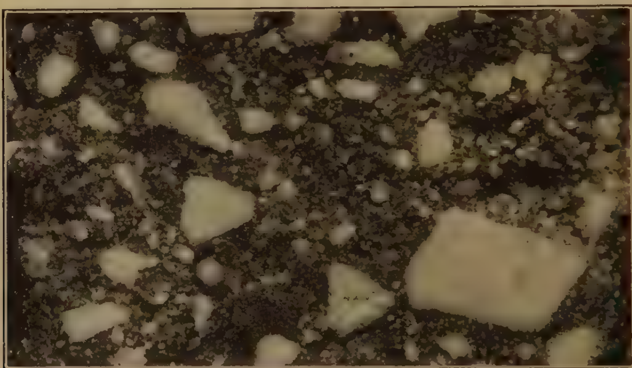

FIG. 8.—PHOTOMICROGRAPH OF POLISHED SURFACE OF 9-IN. SILICA BRICK. × 2.

The fineness of grinding is determined by the character of brick to be made; the grind used for special shapes has a larger proportion of fines

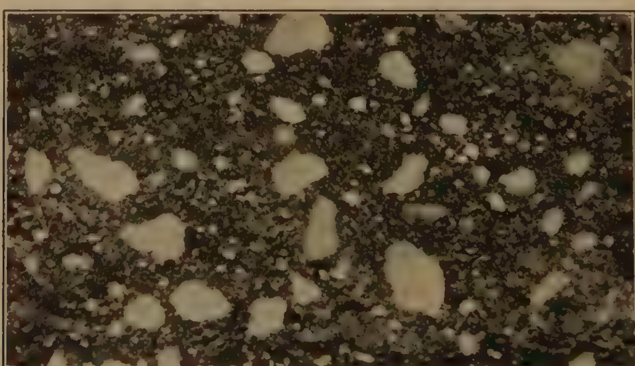

FIG. 9.—PHOTOMICROGRAPH OF POLISHED SURFACE OF SILICA SHAPE. × 2.

than that used for standard 9-in. brick. In either case, the ground material consists of a mixture of grains varying in size from that of a pea to the finest powder.[29] All silica brick are made in molds smaller than

[28] See Bibliography.
[29] See screen analyses, Table 13.

the proposed size of the finished brick; 3/8 to 7/16 in. per foot is allowed for expansion in burning.

The brick must be burned to such a temperature that as much as possible of the permanent expansion shall be acquired in the kiln, and not later in the furnace. The burning temperature is usually given as cone 15 to 16, or 16 to 17, and the time of burning as 20 days, including the time required for cooling. Lange[30] states that cone 15 should be melted down, cone 16 beginning to melt.

Properties of Silica Brick

Analysis.—A typical analysis[31] of brick made from Pennsylvania rock is as follows:

	Per Cent.
SiO_2	96.25
Al_2O_3	0.88
Fe_2O_3	0.79
CaO	1.80
MgO	0.14
Alkalies	0.39
	100.25

Brick with less than 94 per cent. SiO_2 or more than 3 per cent. CaO do not have the necessary refractory qualities; if less than 1.5 per cent. CaO is present, the brick are weak.

Density.—Principally on account of the progressive transformation of quartz into other modifications of silica upon the application of heat, the density of a silica brick is dependent upon the time and temperature of burning. Within certain limits, long-continued heating or burning at higher temperatures causes the density to diminish. The specific gravity of a silica brick should presumably lie between 2.65 (the value for quartz) and 2.28 (the value for tridymite). Microscopic cracks permeating the material render accurate determinations difficult, and results obtained in the ordinary way by the use of large pieces are quite sure to be inaccurate, even when the sample has been previously boiled.

For more accurate results, the finely powdered substance, boiled to remove adherent air bubbles, is usually tested in a pycnometer. Although somewhat laborious, this method with homogeneous materials gives results accurate in the second decimal place. As burned quartz is not homogeneous (see Fig. 16 and 17), and as the very fine cracks and pores are not always filled by the pycnometer liquid, the accuracy of the determinations in the case of silica brick is considered to be $\pm$ 0.02.[32]

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[30] *Stahl und Eisen* (Oct., 1912), **32**, 1729.

[31] K. Seaver: *Op. cit.*, 134.

[32] K. Endell: *Stahl und Eisen* (Oct., 1913), **33**, 1773.

In this way Goerens and Gilles[33] determined the specific gravity of certain samples as 2.44. The figures given for the volume density of silica brick range from 1.50 to 1.88, averaging about 1.67 for a well burned 9-in. brick. The values quoted by different writers for the "true specific gravity" of this material are quite inconsistent, varying from 2.05[34] to 2.75.[35]

Inconsistencies inexplicable by variations in the materials studied may perhaps be explained on the following grounds:

Let W = weight of specimen in grams.
V = entire volume of body in cubic centimeters
V_s = volume of rock substance.
V_p = volume of pores.
V_o = volume of open pores, which absorb water.
V_c = volume of closed pores and cracks, which absorb no water.
$V_p = V_o + V_c$.
$V = V_s + V_p = V_s + V_o + V_c$.

It is evident that three possible values may be obtained for the specific gravity.

1. G_b. Bulk specific gravity $= \dfrac{W}{V} = \dfrac{W}{V_s + V_o + V_c}$.

2. G. True specific gravity $= \dfrac{W}{V_s}$. As already pointed out, for silica brick this value should probably lie somewhere between 2.28 and 2.65, depending on the conditions of burning.

3. G'. Apparent specific gravity, determined from the weights of dry specimen and water displaced after saturation. Apparent specific gravity $= \dfrac{W}{V_s + V_c}$. Its value lies between that of G and G_b and is dependent on the relation of closed to open pore space. When $V_c = 0$, $G' = G$; when $V_o = 0$, $G' = G_b$.

In many cases, the values quoted for the "true specific gravity" of silica brick are obviously too low. It is not improbable that in such cases the value designated above as G' has been considered to be the true density and so reported.

Porosity.—The porosity of silica brick varies within wide limits, dependent upon the methods of manufacture. The importance of this property lies in its relation to thermal conductivity and permeability

[33] Goerens and Gilles, *Ferrum* (Nov., 1914), 12, 17.
[34] F. T. Havard: *Op. cit.*, 37.
[35] Wologdine and Queneau: Conductivity, Porosity, and Gas Permeability of Refractory Materials, *Electrochemical and Metallurgical Industry* (Oct., 1909), 7, 433.

to gases.[36] Under ordinary conditions pore space acts as a heat insulator.[37] Following the notation of the preceding paragraph,

Total pore space in per cent. of volume of body $= P_t = \frac{100\,V_p}{V} = 100\,\frac{(1 - G_a)}{G}$.

Open pore space in per cent. $= P_o = \frac{100 V_o}{V} = 100\,\frac{(1 - G_b)}{G'}$.

P_o is determined by absorption tests; P_t, by ascertaining the true specific gravity. Many writers upon silica brick do not make this distinction, but assume that $P_o = P_t$. The figures commonly given for porosity range from 18 to 43 per cent. of volume of the brick.

Strength.[38]—The figures in Table 7 are taken from Havard.

TABLE 7.—*Crushing Strength of Silica Brick**

Silica Brick Made in	Crushing Strength, Pounds per Square Inch		
	Side	Edge	End
Illinois	2,345	1,546	2,119
Pennsylvania	2,896	1,044	1,551
Missouri	2,370	1,792	1,803

Seaver[39] studied the effect of reburning on crushing strength and modulus of rupture with silica bricks made in Pennsylvania burned repeatedly to 1,540°±. His results (Table 8) show that with brick burned three times, the strength of the brick is increased by each reburning.

TABLE 8

Number of Burns	Average Modulus of Rupture, Pounds per Square Inch. Brick Stood on Edge, Supports 6 In. apart	Average Crushing Strength, Pounds per Square Inch. Brick Tested Flatwise
1	624	4,313
2	809	4,396
3	1,001	4,500

At high temperatures, the strength of a brick is much less than at

* F. T. Havard: *Op. cit.*, 37.

[36] Wologdine and Queneau: *Op. cit.*
Goerens and Gilles: *Op. cit.*

[37] Dougill, Hodsman and Cobb: Thermal Conductivity of Refractory Materials, *Journal of the Society of Chemical Industry* (May, 1915), 34, 465–471.

[38] See also under Cross-breaking and Crushing Tests.

[39] K. Seaver: *Op. cit.*, 138.

ordinary temperatures.[40] Subjected to a load of 50 lb. per square inch and gradually heated, chrome bricks fail suddenly at 1,450°, magnesia bricks at 1,550°. Silica bricks under the same conditions are unaffected at 1,450°.

Melting Point.—The melting interval of silica brick is generally considered to lie between cones 34 and 36 (1,740 to 1,775° C.)[41]; Havard[42] states that silica brick softens between 1,750 and 1,800°. Kanolt[43] measured the temperature at which pieces of hazelnut size visibly changed their form. For three different pieces the temperatures were 1,700°, 1,705°, 1,700°. Endell[44] tested brick consisting essentially of cristobalite, as well as brick consisting essentially of tridymite. Pea-sized pieces of both became completely melted at 1,700° ± 10.

The apparent discrepancy between the melting temperature of silica brick as determined in the laboratory, and the temperatures to which they are sometimes subjected in practice, is explained as follows:[45] Only the immediate surface of the brick reaches the furnace temperature. This is soon melted when 1,700° is exceeded, forming a viscous glaze, which protects the portion of the brick immediately adjacent. This portion does not reach its melting point on account of the temperature drop toward the exterior of the furnace.

TABLE 9.—*Specific Heat of Silica Brick**

t°	Heyn		Steger	Norton†
	True Specific Heat at t°	Mean Specific Heat between 20° and t°	Mean Specific Heat between 0° and t°	Mean Specific Heat between 20° and t°
0° C.	0.200	0.200		
200° C.	0.237	0.219	0.205	
400° C.	0.270	0.237		
600° C.	0.282	0.250		0.225
800° C.	0.285	0.261		0.236
1,000° C.	0.298	0.262		
1,100° C.				0.255
1,200° C.	0.291	0.266		

* E. Heyn: Thermal Conductivity of Refractory Building Materials, *Mitteilungen aus dem Königlichen Materialprüfungsamt* (1914), **32**, 180.

W. Steger: Specific Heats of Refractory Products, *Silikat Zeitschrift* (Nov., 1914), **2**, No. 11, 203.

† C. L. Norton: Private communication to K. Seaver.

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[40] G. H. Brown: Note on Load Tests Made on Magnesite, Chrome and Silica Brick, *Transactions of the American Ceramic Society* (1912), **14**, 391–393.

[41] K. Endell: *Stahl und Eisen* (Nov., 1913), **33**, 1856.

[42] F. T. Havard: *Op. cit.*, 29.

[43] *U. S. Bureau of Standards, Technologic Paper* 10 (1912).

[44] K. Endell: *Op. cit.*, 1857, 1858.

[45] K. Endell: *Op. cit.*, 1857, 1858.

Thermal Conductivity.—The divergence in the results obtained by different observers in the determination of thermal conductivity has occasioned considerable discussion. As Dudley points out, these variations are due to variations in the materials tested, as well as to differences between, or inaccuracies of, the methods employed. For comparative purposes, the coefficients are given in Table 10 for some fire-clay brick also. The coefficient k represents the flow of heat in calories per second, per square-centimeter area, through 1-cm. thickness, for a temperature difference of 1° C. To convert these figures into B.t.u. per 24 hr. per square-foot area per inch thickness per 1° F. temperature difference, they should be multiplied by 69,700.

TABLE 10

Material	Conductivity: Temperature t_1	Conductivity: Temperature t_2	Conductivity: Mean k between t_1 and t_2	Reference
Silica brick, "Star" brand 95.9 per cent. SiO_2	0 0	100 1,000	0.0021 0.0031	Calculated from results of Boyd Dudley, Jr.
Clay brick, "Woodland" brand 52.9 per cent. SiO_2, 42.7 per cent. Al_2O_3	0 0	100 1,000	0.0016 0.0025	*Transactions of the American Electrochemical Society*, vol. 27, p. 336 (1915).
"Quartzite" brick, 73.9 per cent. SiO_2, 22.9 per cent. Al_2O_3.	0 0	100 1,000	0.0020 0.0027	
Silica brick, 96 per cent. SiO_2, specific gravity, 2.44	0 0	100 1,000	0.0028 0.0031	Calculated from results of Goerens and Gilles, *Ferrum*, vol. 12, pp. 1, 17 (Oct., Nov., 1914).
Claybrick, 53.9 SiO_2, 40.2 Al_2O_3	0 0	100 1,000	0.0022 0.0027	

Notwithstanding the lack of numerical correspondence in results, Dudley, as well as Goerens and Gilles, finds that the coefficients are greater at high temperatures than at low, and that the conductivity of silica brick is greater than that of clay brick. The latter conclusion is corroborated by Brown,[46] who compared the flow of heat through the following materials:

Silica brick................95.5 per cent. SiO_2, 1.9 per cent. Al_2O_3.
Quartzite brick............83.4 per cent. SiO_2, 11.8 per cent. Al_2O_3.
Clay brick.................63.2 per cent. SiO_2, 33.0 per cent. Al_2O_3.

Cylindrical specimens were heated at one end to 1,300° C. while the other end was exposed to the atmosphere at 31° C. Under these condi-

[46] G. H. Brown: Relative Thermal Conductivities of Silica and Clay Refractories, *Transactions of the American Ceramic Society* (1914), **16**, 382–385.

tions, after equilibrium had been attained, the conductivity of the silica brick was 9 per cent. greater than that of the clay brick, and 3 per cent. greater than that of the quartzite brick.

Wologdine[47] shows that the conductivity of both silica and clay refractories increases with higher temperatures of burning. His figures indicate that silica is a poorer conductor than clay, for material burned at, or lower than 1,300°. Marshall[48] quite properly holds that while Wologdine's data are probably correct for the materials investigated, his figures are not practically applicable in a comparison of American refractories, since commercial silica brick are burned at a higher temperature than 1,300°.

The data given by Boyd Dudley, Jr.,[49] are of particular interest, in that the materials studied are three well-known brands of American refractories. The mean conductivities between any two temperatures t_1 and t_2 can be calculated from his results expressed by the following straight-line formula based upon measurements taken up to about 1,000° C.:

"Woodland" brick....................	$k = 0.0015 + 0.0_{6}10\,(t_1 + t_2)$
"Quartzite" brick....................	$k = 0.0019 + 0.0_{6}08\,(t_1 + t_2)$
"Star" silica brick....................	$k = 0.0020 + 0.0_{6}11\,(t_1 + t_2)$

These formulæ should be used with caution above 1,000°, and the formula for silica brick should probably not be applied above 1,050°, as Heyn[50] finds a sharp rise in the conductivity of this material between 1,050° and 1,100°.

Expansion.—The problem of the expansion of silica brick is thus stated by Seaver:[51] In the process of manufacture, a permanent expansion occurs under the influence of heat, swelling the brick from the mold or "green" brick size to the required or "burned" size. It is desired that the permanent expansion should, if possible, be completed in the burning, so as to prevent further swelling in subsequent use. "In every burned brick expansion will occur on heating—a true thermal expansion, identical in its general character with that incident to the heating of practically all bodies. Such thermal or temporary expansion will disappear upon cooling, and being the expression of the coefficient of expansion of the silica it is unavoidable. This thermal or temporary expansion occurring upon heating a burned brick is not to be confused with the permanent expansion produced in the body of the brick during its manufacture. It is argued that if a silica brick is not properly burned,

[47] *Electrochemical and Metallurgical Industry* (Sept.-Oct., 1909), **7**, 383 and 433.

[48] S. M. Marshall: Relative Thermal Conductivity of Silica and Clay Brick, *Metallurgical and Chemical Engineering* (Feb., 1914), **12**, 74.

[49] See Table 10.

[50] *Mitteilungen aus dem Königlichen Materialprüfungsamt* (1914), **32**, 181.

[51] K. Seaver: *Op. cit.*, 133.

then, upon reheating in actual coke-oven practice, there will be not only a normal thermal expansion, but an additional expansion of indeterminate amount which will be permanent. It is essential in the construction of the coke ovens that the expansion, whatever its nature, be as small as possible, and that, whatever it is to be, it be known and adequately cared for."

Since the final increments of permanent expansion are acquired but slowly under the influence of high temperatures it is found that reburned brick show a slight increase in volume after each of several heat treatments. It is generally held that the manufacturer can do nothing to diminish the temporary expansion.[52]

The linear expansion in the burning from "green brick" size to size of finished product is 3/8 to 7/16 in. per foot, that is, 3.1 to 3.6 per cent. It may be here noted that the change of volume in the complete transformation of quartz to cristobalite corresponds to a linear expansion of 4.35 per cent.; quartz to tridymite, 5.3 per cent.

With a number of brick burned repeatedly to cone 16 to 17, Cramer[53] found an average permanent linear expansion of 3.2 per cent. on the first burn, an increment of 0.6 per cent. on second burn and 0.2 per cent. on the third. In a test mentioned by Havard,[54] a Pennsylvania silica brick reheated to 1,400° C. had a total linear expansion of 1.7 per cent., of which 0.4 per cent. was permanent, 1.3 per cent. temporary. Stockman and Foote,[55] at the Massachusetts Institute of Technology, studied the temporary thermal expansion of two brands of silica brick with the results shown in Table 11.

TABLE 11

Brand	Average Total Linear Expansion at			
	300°, Per Cent.	600°, Per Cent.	900°, Per Cent.	1,200°, Per Cent.
I. Reese & Sons	0.9	1.5	1.9	
Star	...	0.7	0.9	1.1

Practical experience shows that the thermal expansion of well-burned brick amounts to 3/16 to 1/4 in. per foot (1.6 to 2.1 per cent.) of which the greater part occurs below 600° C.

"In laying silica brick it is important to leave room for expansion;

52 Experimental results hereinafter given indicate that this view is subject to certain qualifications.

53 *Tonindustrie Zeitung* (1901), **35**, 876; *Stahl und Eisen* (July, 1901), **21**, 772.

54 F. T. Havard: *Op. cit.*, 37.

55 Massachusetts Institute of Technology Mining Department Thesis No. 226, 1902.

thus in a reverberatory furnace for melting sulphide-copper ores, having 12-in. silica brick, ¼ in. is allowed to 1 ft. in the roof, 5/16 in. to 2 ft. in the bottom; the expansion space is occupied by cardboard when the brick is being laid."[56]

Effect of Rapid Temperature Change.—Closely connected with the expansion on heating is the inability of silica brick to withstand rapid changes of temperature. For this reason, both while being manufactured and later when put into use, heating and cooling must take place evenly and slowly. "By quick burning" (in the kilns) "brick may be easily expanded by double or treble the normal amount; but sound brick do not result. This excessive expansion is due to minute fissures opened up in the body of the brick . . . In the early stages of firing the heat must be raised very slowly."[57] If cooled too rapidly, silica brick become friable, crack and spall. An underburned brick is less likely to spall than a well-burned one, but is weak, and has an excessive expansion, largely of a permanent character, when heated in the furnace.

Constitution

The constitution of silica brick and its relation to the phenomena of expansion have engaged the attention of many investigators. K. Endell[58] gives a historical sketch of the most important work done in this connection. In 1890, E. Mallard[59] discovered tridymite crystals in brick which had been in use a year and a half; this observation was confirmed in 1911 by Grum-Grzimailo.[60] The latter found that silica brick, when first made, consisted of grains of unaltered quartz surrounded by an apparently amorphous groundmass, which was held to be glass, together in some cases with varying amounts of tridymite; upon several months' use in the open-hearth furnace, the brick gradually and completely went over into a mass of interlocking twin crystals of tridymite. To avoid continual expansion in the furnace, it was proposed to burn the brick in such a way as to effect the complete transformation to tridymite in the kilns.

Holmquist[61] discovered cristobalite in brick which had melted and hung down from the roof of the furnace as stalactites.

Endell[62] shows that the texture and constitution of silica brick are quite different before use, after 10 charges in the open-hearth furnace, after 50 charges, and again after having been melted. After the first burn, about two-thirds of the material was found to be converted into an

[56] H. O. Hofman: *General Metallurgy*, 368. McGraw-Hill Book Co., New York, 1913.

[57] K. Seaver: *Op. cit.*, 139.

[58] *Stahl und Eisen* (Mar., 1912), **32**, 392.

[59] *Bulletin, Société française de minéralogie* (1890), **13**, 172.

[60] *Stahl und Eisen* (Feb., 1911), **31**, 224–226.

[61] *Tonindustrie Zeitung* (1911), **35**, 1325.

[62] *Stahl und Eisen* (Mar., 1912), **32**, 392–397.

apparently isotropic modification of silica, enclosing unaltered quartz grains of unequal size. This apparently isotropic form was at first believed to be glass, but later shown to be cristobalite.[63] Since the transformation of the original quartz was not complete in the initial burn, the conclusion was reached that only about two-thirds of the expansion takes place in the kilns, and the rest must occur in the furnace. In a brick which had been through 10 charges in the open-hearth furnace, the groundmass was found to be converted to a large degree into tridymite, while the larger quartz grains were transformed into the apparently isotropic condition (cristobalite). After about 50 charges, a good silica brick consisted almost wholly of interlaced tridymite crystals. A brick was examined in which a rind 2 or 3 cm. thick had been melted, while the rest of the piece showed no signs of fusion. Three zones could be distinguished. The one lying nearest the heat had been fully melted, and later, in cooling, crystallized out in part as cristobalite, which was surrounded by a yellow iron-containing glass. The birefringence and other crystal properties of cristobalite formed in this way were much more easily recognized than with cristobalite formed by the transformation of quartz. The colorless cristobalite grains were penetrated by pearly cracks and twinned in a very complicated manner. In the thermal microscope they became isotropic at 220°. Analysis of this fused zone showed a silica content of but 90 per cent.; vaporization of SiO_2 had presumably been favored by intermediate formation of silicon carbide. The central zone showed the typical constitution of a tridymite brick which still contains some cristobalite. As the outer zone was approached, grains of quartz appeared; the latter zone contained considerable unaltered quartz in a groundmass of cristobalite, and in constitution appeared like an unused brick. It had evidently been kept cool by the outside air.

Endell[64] studied also the changes in the constitution of quartzites after repeated burning in a porcelain kiln to 1,450° C. It was found that the finely divided impurities of the quartzite serve as inversion centers and hasten the transformations.

Rock heated three times was converted principally into cristobalite, with some unaltered quartz grains remaining. The latter went over into cristobalite on longer heating and caused a continual expansion. After 10 heatings, there were still a few unaltered quartz particles, and a small proportion of tridymite had appeared in the form of the characteristic wedge-shaped twin crystals. Quartzite burned four times in the porcelain kiln, and later heated 2 hr. to 1,600° was completely transformed into cristobalite.

Seaver and Klein[65] made quantitative determinations of the constitution of silica brick and quartzite burned repeatedly to 1,540° C.,

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[63] K. Endell: *Stahl und Eisen* (Nov., 1913), **33**, 1856.

[64] *Op. cit.*, 1856.

[65] K. Seaver: *Op. cit.*, 137.

that temperature being maintained each time for about 40 hr. Their results are shown in Table 12.

TABLE 12

	Silica Brick			Quartzite	
	Burn			Burn	
	1, Per Cent.	2, Per Cent.	3, Per Cent.	1, Per Cent.	2, Per Cent.
Per cent. cristobalite	77	83	84	49	69
Per cent. quartz and silicates	23	17	16	51	31

No tridymite was to be expected in these tests, since that mineral is not known to be formed under any conditions above 1,470° C.

The relation of the constitution of silica brick to the expansion on heating is indicated by the facts that the volume increase in the transformation of quartz to cristobalite is 13.6 per cent.; quartz to tridymite, 16.7 per cent.

MICROSCOPIC STUDY AND EXPERIMENTAL DATA

Microscopic Examination of Quartzites

The accompanying photomicrographs represent the texture of specimens of quartzite from the deposits most important in the American refractories industry. All of these quartzites consist essentially of individual quartz crystals of random orientation closely packed together and interlocking with sharp angles. No trace of a groundmass or binding material is to be seen, except in a few small and isolated areas, which do not appear in the figures. Small inclusions are visible in great number. The accessory minerals are extremely small in amount, and have not been studied.

Fig. 10 shows the Medina quartzite from Mount Union, Huntingdon Co., Pa. This consists essentially of a mass of interfering quartz crystals, each with a sand grain as a nucleus. The rounded or subangular outlines of the original sand grains are marked by a row of inclusions, and are perfectly distinct. The spaces between these are filled with secondary quartz which has attached itself to the original grains in crystalline continuity with them, so that the original quartz grain and the new quartz border have the same extinction positions and interference colors. The borders contain but few inclusions, and consequently are much clearer than the nuclei.

The Baraboo Quartzite from Ablemans, Wis., is shown in Fig. 11. The grain is much coarser than that of the other quartzites examined, and the line along which the interlocking individuals join is peculiarly jagged. Many of the grains show a marked undulatory extinction with crossed nicols; that is, the whole of the crystal is not extinguished at

the same time, but a wave-like shadow passes over it on rotating the stage.

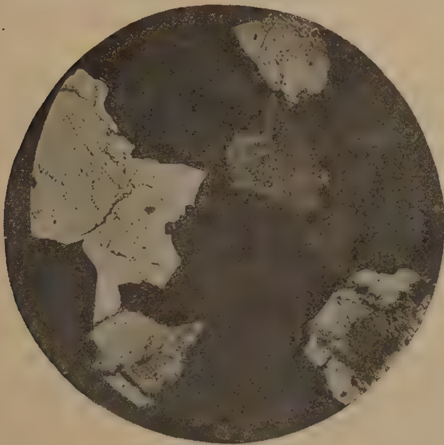


FIG. 10.

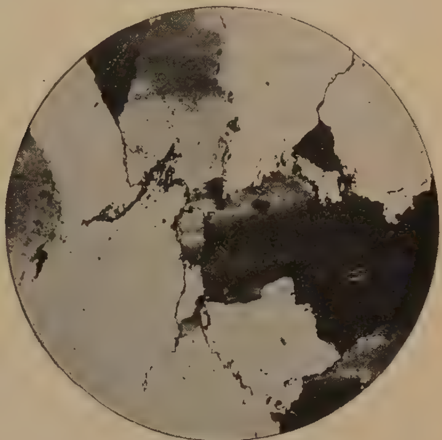


FIG. 11.

FIG. 10.—MEDINA QUARTZITE FROM MT. UNION, HUNTINGDON CO., PENNSYLVANIA (WHITEHEAD). THE FIELD SHOWS A MASS OF INTERFERING QUARTZ CRYSTALS EACH WITH A SAND GRAIN AS A NUCLEUS. THE OUTLINES OF THE ORIGINAL GRAINS ARE WELL SHOWN IN THE TWO CRYSTALS TO THE LEFT. CROSSED NICOLS. × 48.

FIG. 11.—BARABOO QUARTZITE FROM ABLEMANS, WISCONSIN (WHITEHEAD). UNDULATORY EXTINCTION IS SHOWN BY A GRAIN NEAR THE CENTER AND BY ONE NEAR THE TOP OF THE SECTION. CROSSED NICOLS. × 48.

FIG. 12.

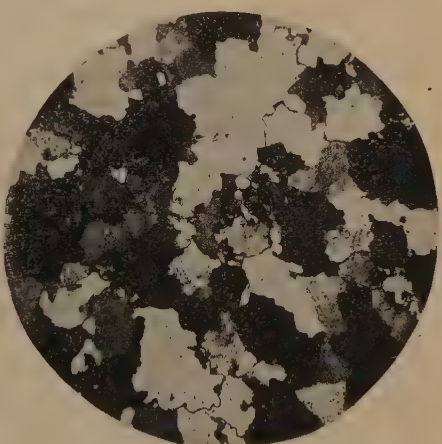


FIG. 13.

FIG. 12.—QUARTZITE FROM NORTHEASTERN ALABAMA, USED IN BRICK MAKING AT WYLAM, ALABAMA (WHITEHEAD). NOTE THE STRIATED APPEARANCE OF SOME OF THE GRAINS. CROSSED NICOLS. × 48.

FIG. 13.—QUARTZITE USED IN BRICK MAKING AT ANACONDA, MONT. (WHITEHEAD). CROSSED NICOLS. × 48.

The crystals of the Alabama quartzite contain numerous inclusions, arranged in nearly parallel lines, which with low magnifications appear under the microscope as fine striations. The Anaconda quartzite is extremely fine in grain.

The textural relations above described, and illustrated in the photomicrographs, do not sustain the conclusion[66] that only quartzites containing a fine-grained groundmass or cement can be used in the manufacture of high-grade silica refractories.

In Figs. 10, 11, 12, 13,[67] quartz grains in extinction positions are shown by black areas; quartz grains in other positions appear as lighter areas. The small black dots are mostly inclusions.

Study of Silica Brick

Description of Test Brick

The conditions under which the test brick were made were devised by the engineering department of the Harbison-Walker Refractories Co., with the purpose of studying the effects of variations in grind and temperature of burning, and of repeated burning, upon the physical properties and constitution of silica brick. Accordingly, three lots of 9-in. brick (approximately 9 by 4½ by 2½ in. in size) were made up. These lots will be hereinafter referred to as "regular", "medium", and "fine grind" respectively. The regular-grind specimens were made by the usual method for 9-in. brick; those of medium grind were given the grind used for silica shapes, and pounded into loose-side molds, as silica shapes are treated. The material used for the fine grind was screened through an 8-mesh hand screen, returned to the pan and reground long enough to add and mix lime thoroughly; the brick were pounded into loose-side molds. Screen analyses are shown in Table 13.

Table 13.—*Screen Analyses of Test Brick*

Grind	Screen Numbers in Meshes per Linear Inch					
	Held on					Through
	8, Per Cent.	16, Per Cent.	20, Per Cent.	30, Per Cent.	40, Per Cent.	40, Per Cent.
Regular	15.3	13.4	4.6	5.4	4.8	56.5
Medium or shape	10.2	13.1	2.2	5.0	4.7	64.8
Fine	....	14.3	4.1	6.5	4.1	71.0

A number of brick from each lot were burned under the ordinary conditions of manufacture at three positions in the kiln, designated for convenience as positions *A*, *B* and *C*. The brick were burned a number of times, being under fire approximately 10 days each time, and specimens were set aside for study after each burning.

[66] See under *Texture*.

[67] Figs. 10 to 12 inclusive are from specimens furnished by the Harbison-Walker Refractories Co. Fig. 13 is from specimen furnished by the Anaconda Copper Mining Co.

The temperatures attained on the various burns as measured by Orton-Seger cones are shown in Table 14.

TABLE 14.—*Temperature Measured by Orton-Seger Cones*

Number of Burn	Position		
	A	*B*	*C*
1	Cone 15	Cone 13	Cone 14
2	Cone 15	Cone 13	Cone 14
3	Cone 15	Cone 15	Cone 15
4	Cone 13	Cone 14	Cone 14
5	Cone 15	Cone 15	Cone 15
6	Cone 16	Cone 14	Cone 14

The fusion points of these cones are as follows:

Cone	Orton Scale, Degrees Centigrade	Seger Revised Scale,* Degrees Centigrade
13	1,390	1,380
14	1,410	1,410
15	1,430	1,435
16	1,450	1,460

* Advertisement in 1914 Calendar of *Tonindustrie Zeitung*

The fusion points in silica brick kilns probably average 70° lower than the figures given by the manufacturers,[68] but can not be told with exactness, since cones give very erratic results when used in such kilns.[69]

Physical Tests

Density and Porosity.—The bulk and apparent specific gravities and per cent. of open pore space of certain brick of the first burn are shown in Table 15. The term "apparent" specific gravity, is used in the sense defined earlier in this paper under the heading, "Properties of Silica Brick." The composition of the brick (hereinafter discussed) suggests that the value of the true specific gravity, of which no determinations were made, probably lies between 2.40 and 2.45. The per cent. of open pore space was calculated from the two specific gravities (see discussion under heading just mentioned).

From the figures given it appears that the greatest porosity is shown by the coarse (regular) grind, and the least by the medium (shape) grind.

[68] Hoffman: *Tonindustrie Zeitung* (1911), **35**, 1099–1100.

S. Geijsbeek: Melting Points of Pyrometric Cones under Various Conditions, *Transactions of the American Ceramic Society* (1912), **14**, 849–871.

[69] K. Seaver: *Op. cit.*, 133.

The data are insufficient to justify any conclusions as to the effect upon porosity of variations in burning temperature.

TABLE 15.—*Tabulated Data: Density and Porosity*

First Burn							
Mark	Grind	Burned to Cone	Bulk Specific Gravity	Apparent Specific Gravity	Open Pore Space, Per Cent. of Volume	Open Pore Space, Mean Per Cent. of Volume	Open Pore Space, Mean Per Cent. of Volume
83–A–1	Regular	15	1.678	2.259	25.7	26.1	
279–A–1	Regular	15	1.655	2.254	26.5		
69–C–1	Regular	14	1.637	2.229	28.5	28.5	
228–B–1	Regular	13	1.671	2.251	25.7	25.7	26.8
585–A–1	Medium	15	1.770	2.247	21.3	21.8	
601–A–1	Medium	15	1.748	2.249	22.3		
679–C–1	Medium	14	1.723	2.232	22.7	22.7	
443–B–1	Medium	13	1.839	2.277	19.2	19.2	21.2
1,148–A–1	Fine	15	1.757	2.272	22.7	22.5	
1,149–A–1	Fine	15	1.765	2.272	22.3		
854–C–1	Fine	14	1.725	2.242	23.2	23.2	
997–B–1	Fine	13	1.774	2.264	21.7	21.7	22.5

Cross-Breaking and Crushing Tests

In the transverse tests the brick were laid edgewise with a span of 6 in. and the load applied at midspan. Modulus of rupture was calculated from the formula $R = \frac{3\,Wl}{2\,bd^2}$, in which l is the distance between supports in inches, b the breadth and d the depth of the specimen in inches, and W the load in pounds at which failure occurred. Compression tests were made on half bricks resulting from the transverse tests. The bricks were bedded flatwise on blotting paper in the testing machine to secure a uniform bearing surface. The crushing strength was obtained by dividing the crushing load in pounds by the mean area in square inches of the two surfaces of the brick.

Unfortunately, but three brick of each lot were available for these tests, a number insufficient to permit the determination of the mean strengths with the proper degree of accuracy. The proposed specifications[70] of the American Society for Testing Materials direct that at least five specimens shall be used in transverse and compression tests upon building brick, and the same requirement should undoubtedly apply to the testing of refractory brick.

The results obtained are shown in Tables 16, 17, and 18.

[70] *Proceedings of the American Society for Testing Materials* (1913), 13, 284.

TABLE 16.—*Sample Log and Data Sheet*
Cross-Breaking and Crushing Tests

Cross-Breaking Span, 6 In.						Crushing: Half Brick Crushed Flat					
Mark	Breadth of Brick, In.	Depth of Brick, In.	W	R	Mean R	Crushing Area, Square Inches 1	2	Mean	W	C	Mean C
83–A–1	2.48	4.45	4,970	910		19.80	17.58	18.69	51,900	2,780	
279–A–1	2.46	4.45	4,820	890	920	19.54	17.97	18.75	45,600	2,430	2,680
289–A–1	2.52	4.43	5,260	960		16.46	16.16	16.31	46,100	2,830	
585–A–1	2.66	4.73	7,010	1,060		21.75	21.33	21.54	101,000	4,690	
601–A–1	2.66	4.71	7,450	1,140	1,050	21.18	20.24	20.71	120,500	5,820	4,780
715–A–1	2.68	4.71	6,320	960		20.26	20.00	20.13	77,000	3,830	

W = load causing failure in pounds.
R = modulus of rupture, pounds per square inch.
C = compressive strength, pounds per square inch.

TABLE 17.—*Collected Data*
Cross-Breaking and Crushing Tests

First burn — Brick Tested Edgewise 6-in. Span

Mark	Grind	Burned to Cone	Cross-Breaking. Load in Pounds	Modulus of Rupture: Modulus' Pounds per Square Inch	Mean	Average Deviation, Per Cent. of Mean	Crushing Strength, Brick Laid Flat: Crushing Strength, Pounds per Square Inch	Mean	Average Deviation, Per Cent. of Mean
83–A–1	Regular	15	4,970	910			2,780		
279–A–1	Regular	15	4,820	890	920	3	2,430	2,680	6.4
289–A–1	Regular	15	5,260	960			2,830		
135–B–1	Regular	13	3,240	600			2,680		
228–B–1	Regular	13	3,630	670	690	10	2,390	2,730	8.5
327–B–1	Regular	13	4,280	800			3,120		
69–C–1	Regular	14	4,010	730			2,210		
102–C–1	Regular	14	4,140	750	770	5	2,050	2,100	3.3
156–C–1	Regular	14	4,520	820			2,040		
585–A–1	Medium	15	7,010	1,060			4,690		
601–A–1	Medium	15	7,450	1,140	1,050	6	5,820	4,780	14.0
715–A–1	Medium	15	6,320	960			3,830		
443–B–1	Medium	13	4,990	800			5,430		
445–B–1	Medium	13	4,310	690	910	24	3,890	4,680	11.0
476–B–1	Medium	13	7,760	1,230			4,710		
679–C–1	Medium	14	5,430	830			4,340		
680–C–1	Medium	14	5,610	860	920	11	4,930	4,820	6.7
744–C–1	Medium	14	7,050	1,070			5,180		
1,148–A–1	Fine	15	5,790	930			7,270		
1,149–A–1	Fine	15	9,450	1,500	1,170	19	4,360	5,880	17.0
1,151–A–1	Fine	15	6,770	1,070			6,010		
997–B–1	Fine	13	5,360	860			6,300		
998–B–1	Fine	13	6,180	990	910	6	5,280	5,840	6.3
999–B–1	Fine	13	5,560	890			5,940		
854–C–1	Fine	14	3,270	520			5,180		
934–C–1	Fine	14	2,840	450	790	52	5,040	5,250	3.4
1,052–C–1	Fine	14	9,010	1,410			5,520		

TABLE 17.—*Collected Data.—Continued*

Second burn

Mark	Grind	Burned to Cone	Cross-Breaking, Load in Pounds	Modulus of Rupture: Modulus, Pounds per Square Inch	Modulus of Rupture: Mean	Modulus of Rupture: Average Deviation, Per Cent. of Mean	Crushing Strength, Brick Laid Flat: Crushing Strength, Pounds per Square Inch	Crushing Strength, Brick Laid Flat: Mean	Crushing Strength, Brick Laid Flat: Average Deviation, Per Cent. of Mean
275–A–2	Regular	15–15	5,820	1,080			2,230		
288–A–2	Regular	15–15	5,160	980			2,750		
290–A–2	Regular	15–15	3,480	620	910	15	2,990	2,680	8.6
293–A–2	Regular	15–15	5,140	940			2,760		
138–B–2	Regular	13–13	5,620	1,040			2,770		
218–B–2	Regular	13–13	4,730	860	1,010	10	2,540	2,760	5.4
330–B–2	Regular	13–13	6,070	1,120			2,970		
10–C–2	Regular	14–14	5,420	980			2,690		
14–C–2	Regular	14–14	4,730	880	920	3	2,490	2,680	4.6
110–C–2	Regular	14–14	4,950	920			2,850		
567–A–2	Medium	15–15	10,440	1,600			5,780		
599–A–2	Medium	15–15	10,730	1,630	1,680	5	7,010	6,030	11.0
628–A–2	Medium	15–15	11,800	1,810			5,290		
467–B–2	Medium	13–13	9,350	1,460			7,030		
472–B–2	Medium	13–13	10,390	1,630	1,310	24	5,200	6,270	11.0
579–B–2	Medium	13–13	5,360	840			6,590		
572–C–2	Medium	14–14	10,340	1,570			5,600		
658–C–2	Medium	14–14	10,620	1,620	1,680	7	5,130	5,570	5.2
699–C–2	Medium	14–14	12,150	1,860			5,970		
824–A–2	Fine	15–15	4,440	680			7,870		
848–A–2	Fine	15–15	4,830	770			7,850		
850–A–2	Fine	15–15	12,450	1,950	1,270	43	6,700	6,890	14.0
970–A–2	Fine	15–15	10,550	1,670			5,150		
961–B–2	Fine	13–13	5,420	860			5,610		
1,019–B–2	Fine	13–13	6,770	1,080	1,070	13	7,390	6,290	12.0
1,042–B–2	Fine	13–13	7,820	1,280			5,870		
805–C–2	Fine	14–14	13,650	2,160			8,560		
836–C–2	Fine	14–14	8,050	1,260	1,590	24	7,150	7,830	6.2
1,012–C–2	Fine	14–14	8,730	1,360			7,780		

TABLE 17.—*Collected Data.—Continued*

Third burn

Mark	Grind	Burned to Cone	Cross-Breaking, Load in Pounds	Modulus of Rupture: Modulus, Pounds per Square Inch	Modulus of Rupture: Mean	Modulus of Rupture: Average Deviation, Per Cent. of Mean	Crushing Strength, Brick Laid Flat: Crushing Strength, Pounds Per Square Inch	Crushing Strength, Brick Laid Flat: Mean	Crushing Strength, Brick Laid Flat: Average Deviation, Per Cent. of Mean
39–A–3	Regular	15–15–15	9,260	1,680			3,320		
53–A–3	Regular	15–15–15	4,060	740			3,000		
144–A–3	Regular	15–15–15	5,770	1,040	1,200	25	2,940	3,110	4.5
249–A–3	Regular	15–15–15	7,330	1,340			3,170		
174–B–3	Regular	13–13–15	7,190	1,310			3,000		
202–B–3	Regular	13–13–15	5,460	990	1,010	20	2,590	2,830	5.7
280–B–3	Regular	13–13–15	3,890	720			2,910		
66–C–3	Regular	14–14–15	6,290	1,170			2,650		
153–C–3	Regular	14–14–15	6,380	1,160	1,110	7	3,490	2,990	11.0
179–C–3	Regular	14–14–15	5,470	990			2,840		
427–A–3	Medium	15–15–15	4,910	770			6,570		
428–A–3	Medium	15–15–15	6,550	1,030			6,520		
439–A–3	Medium	15–15–15	7,840	1,230	1,210	25	6,890	6,550	2.8
501–A–3	Medium	15–15–15	11,680	1,820			6,220		
431–B–3	Medium	13–13–15	3,040	470			4,860		
568–B–3	Medium	13–13–15	10,520	1,610	1,320	43	6,220	6,380	17.0
622–B–3	Medium	13–13–15	12,280	1,880			8,070		
513–C–3	Medium	14–14–15	8,280	1,270			4,620		
672–C–3	Medium	14–14–15	12,790	1,950	1,580	16	4,520	4,830	7.0
674–C–3	Medium	14–14–15	9,770	1,510			5,360		
1,028–A–3	Fine	15–15–15	5,780	910			7,390		
1,046–A–3	Fine	15–15–15	6,340	1,000			6,920		
1,057–A–3	Fine	15–15–15	6,900	1,100	1,240	28	8,000	7,060	9.0
1,066–A–3	Fine	15–15–15	12,130	1,930			5,920		
846–B–3	Fine	13–13–15	13,900	2,180			8,680		
882–B–3	Fine	13–13–15	10,510	1,650	1,730	17	8,610	8,120	9.0
906–B–3	Fine	13–13–15	8,470	1,350			7,010		
880–C–3	Fine	14–14–15	6,800	1,070			7,070		
974–C–3	Fine	14–14–15	3,300	520	680	38	6,240	6,600	11.0
1,074–C–3	Fine	14–14–15	2,790	440			5,860		

TABLE 18.—*Summary*

Cross-Breaking and Crushing Tests

Burn	Burned to Cone	Modulus of Rupture, Brick Tested Edgewise, 6-in. Span						Crushing Strength, Brick Laid Flat					
		Regular Grind		Medium Grind		Fine Grind		Regular Grind		Medium Grind		Fine Grind	
		Mean	a.d.*	Mean	a.d.	Mean	a.d.	Mean	a.d.	Mean	a.d.	Mean	a.d.
1	15	920	3	1,050	6	1,170	19	2,680	6	4,780	14	5,880	17
	14	770	5	920	11	790	52	2,100	3	4,820	7	5,250	3
	13	690	10	910	24	910	6	2,730	8	4,680	11	5,840	6
2	15–15	910	15	1,680	5	1,270	43	2,680	9	6,030	11	6,890	14
	14–14	920	3	1,680	7	1,590	24	2,680	5	5,570	5	7,830	6
	13–13	1,010	10	1,310	24	1,070	13	2,760	5	6,270	11	6,290	12
3	15–15–15	1,200	25	1,210	25	1,240	28	3,110	5	6,550	3	7,060	9
	14–14–15	1,110	7	1,580	16	680	38	2,990	11	4,830	7	6,600	11
	13–13–15	1,010	20	1,320	43	1,730	17	2,830	6	6,380	17	8,120	9

* a.d. represents the average deviation in per cent. of mean.

Discussion of Cross-Breaking and Crushing Tests

Average Deviation and Index of Uniformity.—It is highly desirable not only that the brick should be mechanically strong, but also that different specimens of the same lot should be of uniform strength. For this reason, in Table 19, obviously the lot of brick of "regular grind, mark C–3," is a more satisfactory material, so far as cross-breaking is concerned, than the "fine grind, mark A–2," notwithstanding the fact that the mean modulus of rupture of the former is 1,110 lb. per square inch, of the latter, 1,270 lb. per square inch. Since, plainly, mean values in themselves are insufficient as a basis of comparison for the strength of such brick, the "average deviation" of the single observations from the mean (as developed in the principles of precision of measurements) has been calculated as a measure of uniformity of the different specimens, for each lot of brick tested.[71]

The deviation of single observations from the mean is due to two factors: (1) lack of precision of the measurements; and (2) lack of uniformity of different specimens of the material. In the series of tests herein described, variations due to the former factor amount to less than 2 per cent. and are practically negligible as compared with variations due to the latter. For this reason, the "average deviation" can be considered a true measure of uniformity. A very low "average deviation" is an indication of a high degree of

[71] Similarly, "average deviations" are given by the Mechanical Engineering Department of the Massachusetts Institute of Technology, in reporting results obtained in testing the strength of timber.

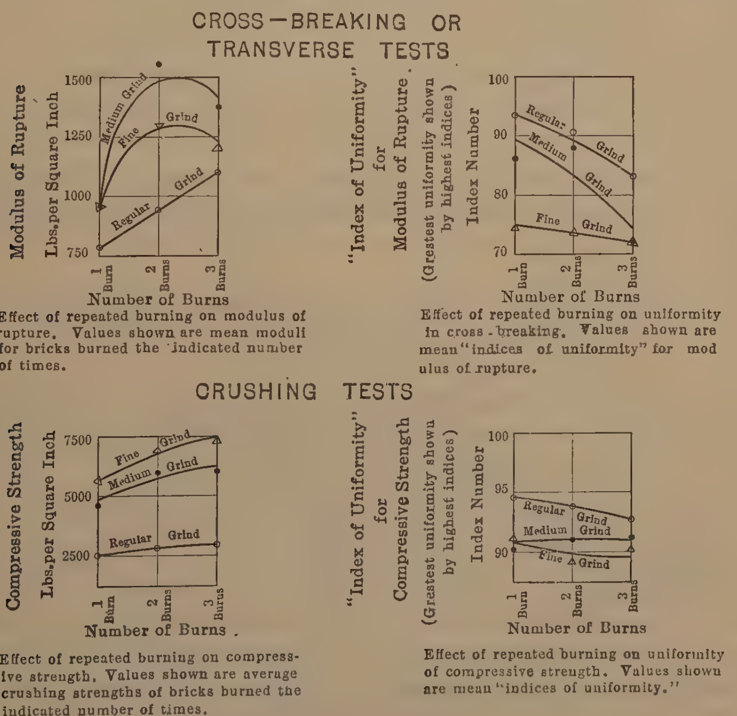
concordance in the values found for different specimens of the same material; a high "average deviation" indicates a lack of concordance.

The method of derivation of the "average deviation" for two specific cases is shown in Table 19.

TABLE 19

Grind	Mark	Modulus of Rupture	Mean Modulus	Deviation from Mean	Average Deviation or "a.d."	a.d. in Per Cent. of Mean
Fine...........	824–A–2	680		590		
Fine...........	848–A–2	770		500		
Fine...........	850–A–2	1,950	1,270	680	540	43.0
Fine...........	970–A–2	1,670		400		
Regular........	66–C–3	1,170		60		
Regular........	153–C–3	1,160	1,110	50	80	7.2
Regular........	179–C–3	990		120		

In the graphical representation of the relative uniformity of different lots of specimens (Fig. 14) use has been made of an "index of uniformity,"



Effect of repeated burning on modulus of rupture. Values shown are mean moduli for bricks burned the indicated number of times.

Effect of repeated burning on uniformity in cross-breaking. Values shown are mean "indices of uniformity" for modulus of rupture.

Effect of repeated burning on compressive strength. Values shown are average crushing strengths of bricks burned the indicated number of times.

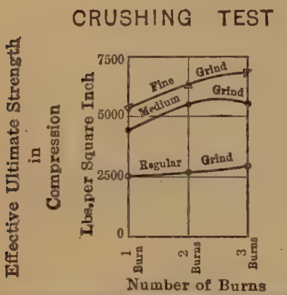
Effect of repeated burning on uniformity of compressive strength. Values shown are mean "indices of uniformity."

FIG. 14.

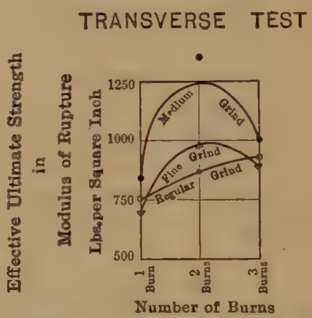
calculated from the average deviation (a.d.) thus: "Index of uniformity" = 100 − (a.d. in per cent. of mean). In the above table, the "index

of uniformity" for modulus of rupture is 57 on this scale for the first lot, 92.8 for the second.

Effective Ultimate Strength.—In comparing the strength of different lots of brick, their relative uniformity must be considered. As shown above, the mean ultimate strength is not a satisfactory basis for comparison. Since the "average deviation" is a measure of uniformity, it seems logical to consider the value obtained by subtracting the average deviation from the mean strength, as a more practical comparative standard. This value may be called the "effective ultimate strength;" and to this, rather than to the mean, the factor of safety should be



"Effective ultimate strengths" were calculated by deducting average deviations from mean compressive strengths. Average values are shown.



"Effective ultimate strengths" were calculated by deducting average deviations from mean moduli of rupture. Average values are shown.

FIG. 15.

applied in calculating a safe working load for materials of which the load-carrying capacity is an important consideration. The effect of repeated burning upon the "effective ultimate strength," as so defined, is shown in Fig. 15.

Effect of Variation in Method of Manufacture.—The effects of the variables considered upon the strength of the test brick are represented graphically in Figs. 14 and 15.

The mean modulus of rupture of medium-grind brick is greatest; of regular-grind, least. The burning temperatures which produce the strongest brick in cross-breaking are as follows:

	First Burn	Second Burn
Regular-grind..........	Cone 15	Cone 15
Medium-grind..........	Cone 15	Cone 14 and 15
Fine-grind.............	Cone 15	Cone 14

Bricks of regular grind increase in cross-breaking strength after each burning; after the first burn, specimens burned at cone 15 are considerably stronger than those burned at cone 13 or 14, after the second burn this is no longer true. Modulus of rupture of medium- and fine-grind brick, in most cases, reaches a maximum on the second burn.

Crushing strength of regular-, medium- and fine-grind brick are to each other approximately as 41:84:100. There is an increase in strength after

each burn; a slight increase for regular grind, a considerable increase for medium and fine grinds.

Uniformity of strength of a given lot of brick diminishes with repeated burning; slightly in compression, considerably in cross-breaking. The most uniform brick are those of regular grind; fine-grind brick are fairly uniform in crushing, very non-uniform in cross-breaking strength.

The figures indicate that the medium-grind brick is the most satisfactory in point of strength, the regular-grind least so. While the compressive strength of the medium-grind is somewhat lower than that of the fine-grind, its cross-breaking strength is greater and it is a much more uniform material. In cross-breaking, the fine-grind shows little if any advantage over the regular-grind, for while its mean strength is high, it lacks uniformity; in compression, it is far stronger than the regular-grind.

There is, in all cases, an advantage in burning the brick a second time; for the medium and fine grinds, a third burn is usually harmful, decreasing the cross-breaking strength considerably, and increasing the compressive strength very slightly. Brick of regular grind increase in strength on each burn up to the third, slightly in compression, considerably in cross-breaking.

The alternate expansions and contractions incurred in repeated burning may tend to cause the formation of minute cracks in the brick and to have an unfavorable effect upon their strength. For that reason it is probable that stronger brick would be produced by maintaining the maximum temperature a greater length of time during a single burn than by resorting to repeated burning.

Microscopic Study of Constitution of Brick [72]

Method in General

Quantitative determinations of the proportions of quartz, tridymite and cristobalite in specimens of the various lots of brick were made by the Rosiwal micrometric method.[73] The measurements were all made upon powdered material, the minerals being identified through comparison of indices of refraction by immersion in liquids of definite indices. The Zeiss mechanical stage, and a lens system giving a magnification of about 390 diameters, were used.

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[72] The microscopic work was performed under the direction of Dr. C. H. Warren, without whose instruction and assistance the writer would not have ventured to undertake a study of this character.

[73] A. Johannsen: *Manual of Petrographic Methods*, 291. McGraw-Hill, New York, 1914. Lincoln and Rietz: Determination of the Relative Volumes of the Components of Rocks by Mensuration Methods, *Economic Geology* (1913), 8, 120–139.

The indices of refraction of quartz are 1.544 to 1.533; cristobalite, 1.484 to 1.487; tridymite, 1.469 to 1.473;[74] calcium silicates, higher than those of quartz.[75] Separate portions of each powder were studied in two liquids whose indices of refraction were respectively 1.495 and 1.479. The higher-refracting quartz and compounds not identified were distinguished in the former liquid from the lower-refracting cristobalite and tridymite. A large part of the higher-refracting material had the refractive index and general properties of quartz.

The proportion of tridymite was determined by immersion of the powder in a liquid with refractive index of 1.479, which is between those of tridymite and cristobalite, but extremely close to them. On account of the latter fact, variations in room temperature caused some difficulty, since the index of the liquid was diminished 0.0005 to 0.0006 for each degree temperature rise. All material with an index lower than 1.479 was reported as tridymite, and the amount of cristobalite obtained by difference.

Sampling

A sample of about 300 grams from each lot of test specimens (100 grams from each brick of the lot) was crushed in a small Blake crusher and a disk grinder, to pieces approximately $\frac{1}{16}$ in. in diameter. After cutting down to ± 20 grams with a split shovel, the sample was reground until all passed through a 140-mesh sieve. For the microscopic study, that fraction was reserved (about 5 per cent. of the whole) which passed through a 240-mesh and was caught on a 260-mesh sieve. The size of these grains as determined microscopically varied from 0.05 to 0.06 mm. in diameter.

Identification

The principal optical property used in identifying the minerals was the index of refraction, by means of the Becke effect. The single grains were commonly made up of two, sometimes all three minerals, and the correct positions of the boundaries between tridymite and cristobalite could be told only by crossing the nicols and rotating the stage.

The more highly refracting material (chiefly quartz in the first burns) was easily recognized by its high relief in the liquid used, and by well-defined Becke lines. Tridymite was distinguished by its low index, low double refraction, and by the lath-shaped and wedge-shaped outlines of its crystals. Identification was confirmed by the facts that the lath-shaped sections had parallel extinction and negative elongation, and that

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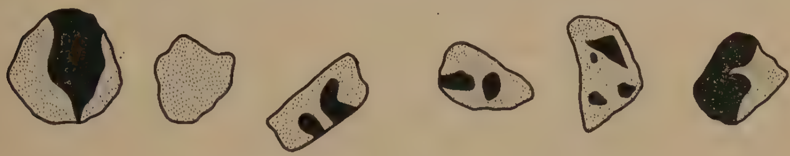
[74] See Table 2.

[75] Rankin and Wright: The Ternary System $CaO–Al_2O_3–SiO_2$, *American Journal of Science*, ser. 4 (1915), **39**, 74.

the index of refraction of the faster ray, as determined by the immersion method in sodium light, was 1.469.[76]

The low relief, the index of refraction, and the extremely low double refraction, which could usually be detected only with the aid of the sensitive tint plate, served to identify cristobalite. In the liquid with index 1.479, the Becke lines were faint, but their directions of motion could be



Quartz shown in black; cristobalite dotted

FIG. 16.—SKETCHES OF GRAINS OF POWDERED BRICK. FROM BRICK OF REGULAR GRIND, BURNED ONCE AT CONE 14. × 218 APPROXIMATELY.

detected upon close observation so long as the room temperature was kept at the proper point.

The lack of homogeneity of the grains led to the introduction of certain unavoidable quantitative errors. Quartz could, as a rule, be recognized even on the interior of the grains by the clearness of the Becke lines and the high relief; yet numerous small particles of residual quartz

Quartz shown in black; cristobalite dotted
tridymite cross-hatched

FIG. 17.—SKETCHES OF GRAINS OF POWDERED BRICK. FROM BRICK OF REGULAR GRIND, BURNED FOUR TIMES AT CONE 14–14–15–14. × 218 APPROXIMATELY.

were often so mingled with the cristobalite that an accurate quantitative separation of the two was impossible. The double refraction of tridymite was the only means of determining with exactness the boundaries between tridymite and cristobalite, but in sections of certain orientation could not be used. The presence of numerous microscopic cracks caused anomalous effects, and in many cases interfered with the Becke tests.

The texture of representative grains is shown in Figs. 16 and 17.

Precision

A coarser powder than that used was found to give less accurate results. The very fine portion, which went through the 260-mesh sieve,

[76] See C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4 (1913), **36**, 351.

was exceedingly difficult to study. Since the latter fraction comprised about 85 per cent. of the original powder, it is evident that any great variation in its composition from that of the study sample will very seriously affect the results. For that reason, check determinations were made in a few cases of the amount of quartz in both the fraction which had passed through the 260-mesh sieve, and that which had been caught upon it. Results so obtained averaged 2 per cent. lower in the former case than in the latter, a difference which lies within the error of observation.

Errors introduced into the tridymite determinations by discarding the very fine material are probably serious, yet it was not found possible to obtain concordant results with this portion of the powder. As shown in Fig. 19, tridymite forms most rapidly in the fine-grained groundmass of the brick, and comparatively slowly in the larger grains. A much larger proportion of the groundmass than of the grains will undoubtedly crush down in the grinding so as to pass through a 260-mesh sieve. For that reason, the amount of tridymite in the fraction of the powder reserved for study, is probably considerably less than in the brick itself.

TABLE 20.—*Results of Microscopic Study—Tabulated Data Constitution of Test Brick*

Lot	Mark	Grind	Burned to Cone	Number of Burns	Constitution in Per Cent. by Volume		
					$n > 1.495$	$n < 1.479$	$n > 1.479$, < 1.495
					Quartz (+Silicates)	Tridymite	Cristobalite
A–1	277, 279	Regular	15	1	26	4	70
C–1	102, 156	Regular	14	1	25	4	71
B–1	135	Regular	13	1	40	1	59
A–1	715, 730	Medium	15	1	24	6	70
A–1	1,150, 1,151	Fine	15	1	26	4	70
C–1	934, 1,052	Fine	14	1	26	5	69
A–2	275, 288, 290, 293	Regular	15–15	2	19		
A–3	39, 53, 144, 249	Regular	15–15–15	3	16	19	65
C–3	66, 153, 179	Regular	14–14–15	3	14	20	64
B–3	174, 202, 280	Regular	13–13–15	3	18	10	
A–3	427, 428, 439, 501	Medium	15–15–15	3	13	17	70
A–3	1,028, 1,066, 1,057, 1,046	Fine	15–15–15	3	13	17	70
C–4	12	Regular	14–14 15–14	4	12	30	58
C–6*	74	Regular	14–14–15 14–15–14	6	14	44	42
B–6*	326	Regular	13–13–15 14–15–14	6	12	45	43

* Analysis of No. 74 and No. 326 by V. Dolmage.

The figures given as the proportions of quartz and tridymite in the test brick represent in each case results of measurements taken on 150 to 250 grains. As a check on the work, percentages were calculated for each 75 to 80 grains. The first two results usually agreed within 3 to 4 per cent.; if not, an additional 80 to 100 grains were studied. The figures so obtained are believed to be truly comparative as showing the direction and approximately the velocity of the transformations occurring, notwithstanding the constant errors due to the difficulties abovementioned, and to the personal equation. The latter errors are believed to be small, since results obtained on the same powder by Mr. Victor Dolmage (who kindly assisted with many of the measurements) and by the writer, usually checked within 3 per cent.

Effect of Variations in Grind and Burning Temperature, and of Repeated Burning, upon Constitution of the Brick

The effect of repeated burning on the constitution of brick of regular grind, burned several times, is shown in Fig. 18. The analyses of this brick after repeated burnings are shown in Table 21.

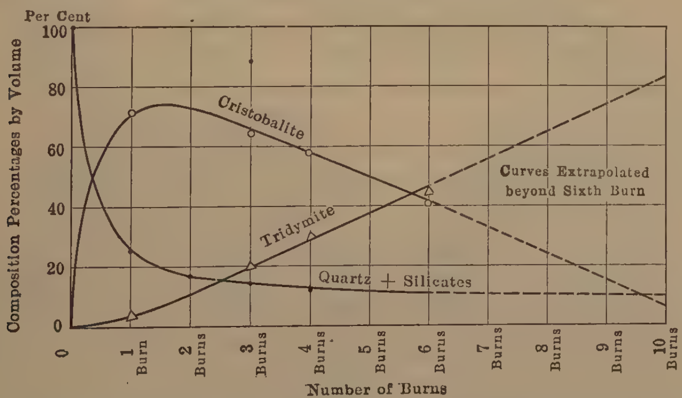


FIG. 18.—EFFECT OF REPEATED BURNING UPON THE CONSTITUTION OF SILICA BRICK. BRICK OF REGULAR GRIND. BURNING TEMPERATURE, CONE 14–14–15–14–15–14.

TABLE 21

Mineral	Volume, Per Cent. Number of Burns					
	1	2	3	4	5	6
Quartz (+Silicates).....	25	....	12	14	....	14
Cristobalite............	71	....	58	64	....	42
Tridymite..............	4	....	30	20	....	44

It is evident that in the first burn the larger part of the quartz is transformed into cristobalite, and that on repeated burning the remaining quartz is very slowly transformed.[77] The cristobalite at first very slowly and later more rapidly, inverts to tridymite. These results are apparently in accordance with the conclusions of Grum-Grzimailo and Endell, that silica brick heated a sufficient length of time at the proper temperature go slowly but completely over into tridymite.

Time and temperature of burning appear to be the controlling factors. Bricks burned a single time at cones 14 and 15 show essentially the same amount of transformation; those burned at cone 13 show considerably less. The effect of variations in grind is comparatively slight.

Study of Thin Sections

Examination of thin sections cut from certain specimens of test brick show that the inversions proceed most rapidly in the groundmass, most slowly in the larger grains. This is illustrated in Fig. 19, a sketch made from thin section of brick of fine grind, burned three times at cone 15. The groundmass and the rims of the grains are made up of cristobalite and tridymite, with a little residual quartz; the interior of the grains consists of cristobalite and quartz, with a very little tridymite.

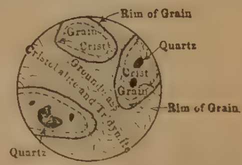


FIG. 19.—SKETCH MADE FROM THIN SECTION OF BRICK OF FINE GRIND, BURNED THREE TIMES AT CONE 15.

Photomicrographs of the thin sections are shown in Figs. 20 to 23 inclusive. The dark areas represent principally cristobalite, the lighter areas, quartz and tridymite. From a comparison of the first two figures with the second two, the increased amount of tridymite in the later burns is at once evident.

THE RELATION OF THE CONSTITUTION OF SILICA BRICK TO THE PHYSICAL PROPERTIES

The Effect of the Addition of Lime

It has already been shown that fineness of grind is not the controlling factor in the formation of tridymite, since that mineral evidently forms nearly as rapidly in the brick of coarse (or regular) grind, as in those of finer grinds. A seeming contradiction exists between this fact and the fact that the inversion to tridymite takes place more rapidly in the finegrained groundmass of the brick than in the larger quartzite fragments (see Fig. 19).

The relative concentrations of lime in the groundmass of brick of

[77] Compare with results obtained by Seaver, shown in Table 12.

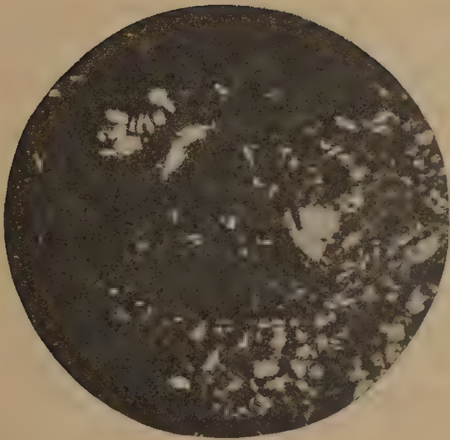


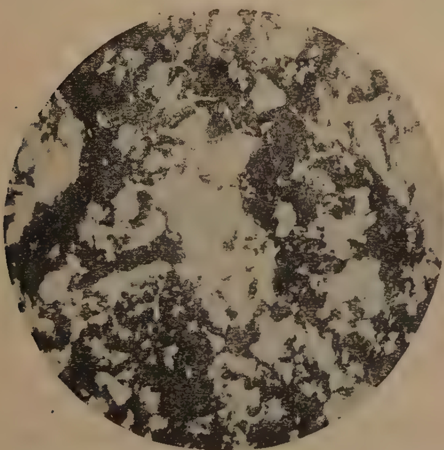
Fig. 20. Fig. 21.

Fig. 20.—Silica Brick of Regular Grind, Burned once at Cone 15 (Whitehead). Crossed Nicols. × 120.

The white areas represent residual quartz grains of irregular size and shape, surrounded by a dark field of cristobalite. Little, if any, tridymite is to be seen.

Fig. 21.—Silica Brick of Fine Grind, Burned once at Cone 15 (Whitehead) Crossed Nicols. × 120.

The figure shows residual quartz grains surrounded by cristobalite. There are a few narrow, elongated white areas in the field, which may represent tridymite.



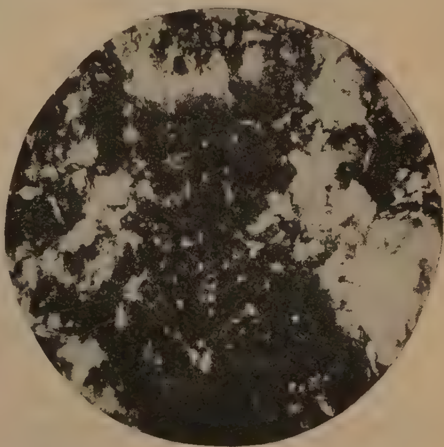


Fig. 22. Fig. 23.

Fig. 22.—Silica Brick of Regular Grind, Burned Three Times at Cone 15 (Whitehead). Crossed Nicols. × 120.

Tridymite and residual quartz are shown by the white areas; cristobalite by the dark areas.

Fig. 23.—Silica Brick of Regular Grind, Burned Three Times at Cone 15 (Whitehead). Crossed Nicols. × 120.

The larger white areas indicate places in which the conversion to tridymite is nearly complete. The smaller white areas in the darker portion of the field represent residual quartz and tridymite, surrounded by cristobalite (shown in black). Figs. 22 and 23 are taken from different parts of the same thin section.

various grinds probably has some bearing in this connection. It will be recalled that in the manufacture of the brick, 2 per cent. of lime is added to ground quartzite containing about 2 per cent. of impurities (mainly Al_2O_3 and Fe_2O_3). Since all of this enters the groundmass, the proportion of lime therein is much higher than 2 per cent.

The greatest concentration of lime in the groundmass will occur in that brick with the least proportion of fines (*i.e.*, the regular grind) in which, therefore, conditions will be most favorable for the formation of flux compounds during the burning. The catalytic action of the flux compounds probably explains the observed inversion of cristobalite to tridymite, which as Fenner has shown, does not form except in the presence of a flux. According to this point of view, an increase in the amount of lime in the brick should accelerate the formation of tridymite.

Specific Volumes of the Silica Minerals

As an aid in interpreting the results of the microscopic study, use has been made of the specific volume curves of Fig. 24, based upon the following data:

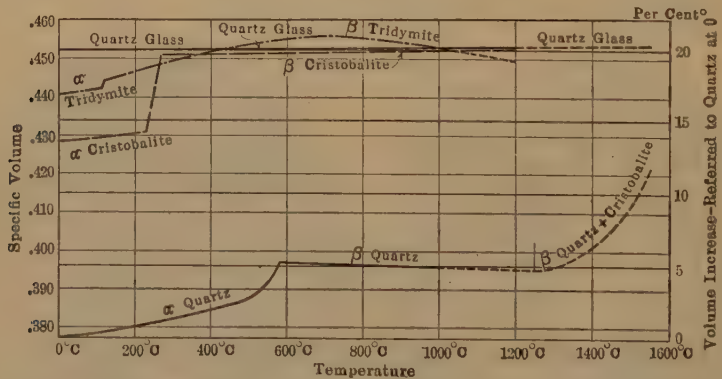


FIG. 24.—SPECIFIC VOLUMES OF THE SILICA MINERALS AND QUARTZ GLASS.

The quartz curve is a reproduction of that shown in Fig. 4, taken from the work of Day, Sosman and Hostetter.

The specific volumes at 20° of quartz glass, tridymite and cristobalite were calculated from their specific gravities, using the values shown below:[78]

	Specific Gravity
Quartz glass	2.210 (Dana, Endell)
Tridymite	2.270 (Fenner)
Cristobalite	2.333 (Fenner)

[78] See Table 3.

The quartz-glass curve was obtained by computing the volume expansion from the linear expansion[79] as given by LeChatelier, and applying these figures to the specific volume at 20°.

The curve for cristobalite is based upon the work of Rieke and Endell, and of LeChatelier. Both show that the expansion of cristobalite from 0° to 230° is slight, and that the volume changes at 230° to 270° is considerable,[80] due to the α to β cristobalite inversion. Rieke and Endell find that the specific volume and coefficient of expansion of cristobalite above 300° are nearly the same as those of quartz glass; the specific volumes, as computed from the expansion data of LeChatelier[79] are considerably lower than those of quartz glass. The view of Rieke and Endell is taken in plotting the cristobalite curve of Fig. 24.

The tridymite curve was calculated from the thermal expansion as given by LeChatelier (Fig. 3). The specific volume of tridymite from 500° to 1,000° is seemingly a little greater than that of quartz glass. While this is probably incorrect, the difference between the specific volumes of the two materials is doubtless small.

The curves of Fig. 24 have been used hereinafter in discussing the probable effect of the changes in constitution upon certain properties of silica brick. It is believed that the inaccuracies of these curves are merely slight errors of degree; and that, while conclusions drawn from them can not be numerically exact, they will serve to indicate the general trend of the changes considered.

The Phenomena of Expansion

Permanent Expansion, Temporary or Thermal Expansion, and the Effect of Rapid Temperature Changes.—The permanent expansion of silica brick after the first burning (in which ±5 per cent. of tridymite and ± 70 per cent. of cristobalite are formed) amounts to 10 to 11 per cent. by volume. The gradual decrease of quartz and increase of tridymite caused by repeated burning, theoretically requires a slight additional expansion after each burn until equilibrium is reached, since the complete transformation of quartz to cristobalite is accompanied by a volume increase of 13.6 per cent.; quartz to tridymite, 16.8 per cent.

At 1,250° C. the specific volumes of cristobalite and tridymite are approximately 14 to 15 per cent. greater than that of quartz. Since the residual quartz of silica brick gradually goes over into the other forms of silica at furnace temperatures, an underburned brick will expand considerably after being placed in use.

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[79] Fig. 3.
[80] Fig. 3; also Fig. 5.

Tridymite expands upon heating much less than does cristobalite. Its specific volume at 20° is 2.8 per cent. greater than that of cristobalite; at 1,000°, nearly the same as that of cristobalite. The gradual transformation of cristobalite into tridymite, shown to occur upon repeated burning of silica brick, probably takes place also in furnace linings. This inversion is accompanied by little, if any, change in volume at furnace temperature.

From the data of Fig. 24, specific volumes at different temperatures were computed for two materials assumed to consist of mixtures of quartz, tridymite, and cristobalite; expansion curves derived from these figures are shown in curves I and II, Fig. 25. Curve I is based upon an assumed composition of 25 per cent. quartz, 70 per cent. cristobalite, 5 per cent. tridymite; curve II on one of 12 per cent. quartz, 43 per cent. cristobalite,

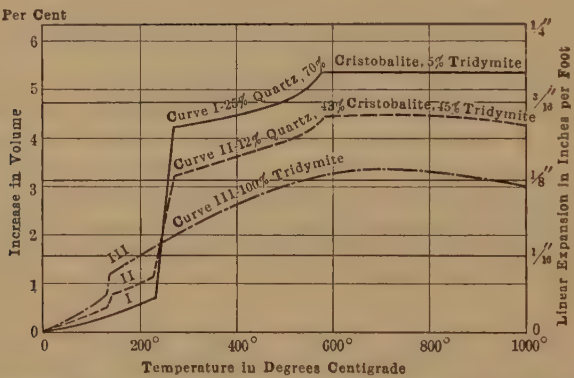

FIG. 25.—THEORETICAL EXPANSION CURVES FOR MIXTURES OF THE SILICA MINERALS.

Composition of Curve I corresponds closely to brick of regular grind *burned once* at cone 14.

Composition of Curve II corresponds closely to brick of regular grind *burned six times* at cone 14–14–15–14–15–14.

42 per cent. tridymite. These figures represent approximately the compositions of brick of regular grind, after one and after six burnings respectively. Curve III is taken from the expansion data of LeChatelier.[81]

Fig. 25 is intended merely to indicate the probable character of the expansion curve of silica brick, and of the effect upon the same of the observed changes in constitution. Obviously, the theoretical curves shown will not correspond precisely with those that would be obtained by actual measurements, being affected by inaccuracies in the data from which they were computed, and, in addition, modified by changes in

[81] Fig. 3.

porosity of the brick, the action of flux compounds, and the lag in expansion due to the temperature gradient between the exterior and interior of the brick.

These curves indicate that the temporary or thermal expansion of silica brick is decreased by repeated burning, and that an all-tridymite brick would have an expansion amounting to about 60 per cent. of that of an ordinary commercial brick, as represented in Curve I.

It is well known that silica brick crack or spall badly if subjected to rapid temperature changes. The phenomenon is probably due chiefly to the rapid expansion at 230° to 270°, accompanying the α to β cristobalite inversion. This effect is diminished by repeated burning. The calculated volume increase at 230° to 270° amounts to 3.5 per cent. for brick burned once, 2.0 per cent. for brick burned six times. An all-tridymite brick should show very little tendency to crack upon rapid heating or cooling, since the only sharp break in the tridymite expansion curve, occurring at 130°, is attended by a volume increase of but 0.3 per cent.

For several reasons, therefore, a large proportion of tridymite in silica brick is desirable, provided the brick are not to be heated continuously above 1,470° C.

1. Tridymite has the least thermal or temporary expansion of the three silica minerals.

2. Tridymite has the greatest specific volume of the three minerals, and can not show any permanent expansion after repeated burning below its melting point.

3. A tridymite brick could probably be subjected to fairly rapid changes of temperature with little danger of cracking.

Comparison of curves I and II seems to show that, while repeated burning produces very desirable effects, the advantages derived up to the sixth burn are entirely disproportional to the additional expense involved, on account of the extreme slowness with which tridymite is formed. Moreover, if used continuously at a temperature exceeding 1,470°, a tridymite brick would slowly revert to cristobalite.

Since the time element appears to be the most important factor in the formation of tridymite, it appears that long-continued burning at the proper temperatures should be more satisfactory than repeated burning. The maximum temperature is maintained 40 hr.[82] in the burning of silica brick; maintaining this temperature for 2½ weeks should presumably give rise to the same changes in constitution as 10 separate burnings under the ordinary conditions of manufacture, and produce brick consisting almost wholly of tridymite (see Fig. 16).

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[82] K. Seaver: *Op. cit.*, 136.

Burning Temperature

In order to make a brick containing the maximum proportion of tridymite, the burning temperature should be that at which the inversion to that mineral proceeds most rapidly. It appears that this must occur below 1,470°, since tridymite is not known to form under any conditions above that point. While the inversion proceeds more rapidly at cones 14 and 15 than at cone 13, no conclusions can be drawn from the data herein given as to the most favorable temperature for burning.

Suggestions for Future Study

It seems desirable that the theoretical advantages of a high-tridymite brick should be investigated by comparing the thermal expansion of an ordinary commercial brick with that of a brick high in tridymite. The effect of rapid temperature changes might be compared by means of spalling tests.

If it should be proven that the production of a tridymite brick would be advantageous, the possibility of making such a brick on a commercial basis for certain purposes might well be considered. A small furnace, in which close temperature regulation is possible, should preferably be employed; and the results thereby obtained, if promising, should later be tried out on a larger scale.

The temperature at which the formation of tridymite proceeds most rapidly in silica brick, and the inversion-velocity at that temperature, are as yet undetermined. The possibility of accelerating the inversion by a slight increase in the amount of lime used or by the addition of some other catalyzing material, is worth considering.

It is not impossible that by long-continued heating at the proper temperature, with possibly a slight additional amount of lime, a tridymite brick might be produced on a commercial basis. The results of tests so far made, however, indicate that this is extremely doubtful.

SUMMARY

A brief review is given of the literature pertaining to the silica minerals and the silica refractories.

Microscopic study of the quartzites most important in the American refractories industry shows that they consist essentially of interlocking quartz crystals of random orientation, without a groundmass or cement. This is not in accordance with Wernicke's conclusion that no quartzites are suitable for the manufacture of high-grade refractories except those

which consist of quartz grains in a cement of amorphous silica or cryptocrystalline quartz.

Physical tests and microscopic studies were made of silica brick manufactured under varying conditions of grind, and burned one to four times at slightly varying temperatures. The grinds considered were (1) "regular," or coarse, (2) "shape," or medium, and (3) fine; the burning temperatures ranged from cone 13 to cone 16.

In all cases, a second burning increases the strength of the brick: a third burning has a harmful effect upon the strength in some cases; in others, causes little change. Brick of shape grind are the strongest, those of regular grind the weakest. With a single burning, a temperature of cone 15 produces a stronger material than cone 13 or 14. Uniformity of strength of different specimens of a given lot diminishes upon repeated burning. The most uniform brick are those of regular grind; the least uniform, those of fine grind. Attention is called to the insufficiency of average values as a proper basis for the comparison of strength of non-uniform materials, and for practical purposes the use of certain derived values, termed the "index of uniformity" and the "effective ultimate strength" is suggested.

Upon repeated burning, the quartz, of which the brick is made, is transformed through cristobalite into tridymite. By microscopic methods, estimates were made of the degree of transformation in various bricks. The figures obtained, while not numerically exact, indicate the general trend of the changes considered. The powder studied from brick burned a single time under the ordinary conditions of manufacture consists by volume of approximately 25 per cent. quartz + silicates, 70 per cent. cristobalite and 5 per cent. tridymite; after a sixth burn the composition is approximately 12 per cent. quartz + silicates, 43 per cent. cristobalite, and 45 per cent. tridymite. The inversions proceed more rapidly in the groundmass of the brick than in the larger grains.

It is considered desirable that silica brick should contain a large proportion of tridymite because (1) tridymite has the least thermal expansion of the silica minerals; (2) it will show no permanent expansion after repeated burning; (3) a tridymite brick could probably be subjected to fairly rapid changes of temperature with little danger of injury.

The advantages in these respects theoretically derived up to the sixth burn are believed to be wholly disproportional to the additional expense involved.

SUPPLEMENTARY DATA

Spalling Tests

General Discussion.—It has been shown in the preceding pages that a relation probably exists between the amount of cristobalite in a silica

brick, and its tendency to spall upon rapid change of temperature; and that since the percentage of cristobalite decreases on repeated burning, the spalling tendency should likewise diminish. It was desired to confirm this conclusion through experimental data.

In spalling tests as ordinarily conducted, brick are alternately heated and cooled rapidly a definite number of times between fixed temperature limits; they are usually cooled in the air, but sometimes by plunging into water. The loss of weight in per cent. is taken as a measure of the spalling. This method is not altogether satisfactory, since it does not indicate the internal condition of the brick, which may possibly weaken considerably in the test without much spalling, or lose surface spalls without much weakening; and if the hot test pieces are water-cooled, the formation of steam in the pores may disrupt the brick and lead to inaccurate conclusions.

For the purposes of this study it was decided to make use of a milder heat treatment than that usually employed; one sufficiently severe to cause internal cracks, yet not severe enough to cause spalls to chip off. For each lot of brick studied, the cross-breaking strength was determined upon a certain number of specimens which had been thus treated, and also upon an equal number which had not. The loss in mean strength, expressed in per cent., is considered to represent the "comparative spalling tendency."

Spalling Test in Detail.—The test was carried out as follows: The brick to be subjected to the heat treatment were placed in a gas-fired test kiln, 16 at a time, and heated at a uniform rate of 15° C. per hour (so as to cause no injury to the specimens during the heating) until 600° C. was attained. They were kept at that temperature for 3 hr. and at the end of that time were withdrawn and placed on edge on inverted steel pallets about 2 ft. apart, in a place protected from drafts, and allowed to cool in the air of the laboratory. They were not moved until the following day.

This treatment had the desired effect. No pieces spalled off, nor were the brick visibly shattered; yet when any two brick were struck together they gave a dead ring.

In Table 22 are shown the cross-breaking strengths before and after this heat treatment, of brick of regular grind burned a number of times varying from one to six. The burning temperatures are those shown in Table 14.

In Table 23, these values are summarized and the loss in strength, assumed to represent the comparative spalling tendency, is shown. Figs. 26 and 27 are graphic representations of the data given in Table 23.

TABLE 22.—*Cross-Breaking Tests Made to Determine Spalling Tendency*

Brick Tested on Edge; Supports 6 In. Apart; Load Applied at Midspan

Cross-Breaking Strength of Brick *Not* Subjected to Heat Treatment						Cross-Breaking Strength of Brick Subjected to Heat Treatment					
No. of Burns	Mark	Lot	Cross-breaking, Load in Pounds	Modulus of Rupture: Modulus, Pounds per Square Inch	Modulus of Rupture: Mean	No. of Burns	Mark	Lot	Cross-breaking, Load in Pounds	Modulus of Rupture: Modulus, Pounds per Square Inch	Modulus of Rupture: Mean
1	83	A	4,970	910		1	268	A	1,730	310	
1	279	A	4,820	890		1	295	A	2,480	443	
1	289	A	5,260	960		1	299	A	2,290	412	
1	69	C	4,010	730	848	1	56	C	2,080	373	339
1	71	C	4,790	877		1	73	C	1,900	341	
1	102	C	4,140	750		1	97	C	780	138	
1	156	C	4,520	820		1	166	C	2,000	357	
2	275	A	5,820	1,080		2	18	A	3,430	605	
2	288	A	5,160	980		2	88	A	2,360	421	
2	290	A	3,480	620		2	125	A	3,120	564	
2	293	A	5,140	940		2	240	A	2,030	365	
2	138	B	5,620	1,040		2	196	B	2,440	434	
2	218	B	4,730	860	919	2	22	B	1,600	288	444
2	330	B	6,070	1,120		2	324	B	2,350	420	
2	335	B	3,260	606		2	351	B	2,360	425	
2	10	C	5,420	980		2	96	C	2,310	414	
2	14	C	4,730	880		2	108	C	2,300	419	
2	33	C	5,650	997		2	151	C	2,730	489	
2	110	C	4,950	920		2	318	C	2,740	490	
3	39	A	9,260	1,680		3	70	A	2,850	505	
3	53	A	4,060	740		3	123	A	2,900	518	
3	144	A	5,770	1,040		3	263	A	2,390	425	
3	249	A	7,330	1,340		3	303	A	2,620	468	
3	82	B	6,250	1,100		3	51	B	2,770	495	
3	174	B	7,190	1,310	1,120	3	208	B	3,230	572	489
3	202	B	5,460	990		3	267	B	2,800	494	
3	280	B	3,890	720		3	276	B	3,020	531	
3	66	C	6,290	1,170		3	32	C	3,040	538	
3	78	C	6,480	1,170		3	152	C	1,960	355	
3	153	C	6,380	1,160		3	155	C	2,730	491	
3	179	C	5,470	990		3	171	C	2,650	478	
4	41	A	5,160	947		4	149	A	3,000	547	
4	85	A	6,070	1,100		4	247	A	1,830	325	
4	91	A	3,900	691		4	305	A	2,800	500	
4	238	A	5,600	1,010		4	310	A	1,720	306	
4	188	B	5,170	923		4	89	B	3,040	549	
4	219	B	4,750	848	883	4	140	B	2,150	384	458
4	291	B	4,220	764		4	195	B	2,280	406	
4	329	B	5,480	990		4	350	B	2,050	366	
4	8	C	6,790	1,200		4	22	C	2,950	534	
4	170	C	4,420	789		4	77	C	2,580	471	
4	175	C	3,300	594		4	130	C	2,850	521	
4	313	C	4,150	737		4	148	C	3,230	586	
5	67	A	4,290	769		5	19	A	2,450	434	
5	104	A	4,650	830		5	76	A	2,270	407	
5	147	A	7,360	1,320		5	132	A	3,020	543	
5	158	A	5,730	1,040		5	173	A	2,900	516	
5	186	B	6,270	1,150		5	80	B	3,010	528	
5	215	B	4,920	886	1,050	5	216	B	3,350	601	558
5	224	B	6,000	1,060		5	308	B	3,800	691	
5	230	B	5,350	961		5	342	B	2,600	466	
5	257	C	5,950	1,090		5	122	C	2,820	510	
5	262	C	6,920	1,240		5	145	C	3,840	688	
5	306	C	5,950	1,080		5	165	C	3,410	611	
5	338	C	6,650	1,200		5	297	C	3,860	695	
6	4	A	5,620	1,010		6	87	A	3,570	638	
6	201	A	5,230	935		6	126	A	3,170	573	
6	266	A	4,780	850		6	157	A	3,250	572	
6	325	A	5,600	1,010		6	159	A	2,810	504	
6	136	B	5,580	1,010		6	30	B	2,610	468	
6	211	B	5,750	1,030	951	6	185	B	2,640	471	522
6	250	B	5,000	908		6	232	B	1,940	349	
6	255	B	3,680	668		6	320	B	2,750	479	
6	6	C	5,150	916		6	91	C	3,300	600	
6	13	C	3,980	704		6	94	C	2,990	549	
6	34	C	6,590	1,190		6	160	C	2,870	517	
6	150	C	6,560	1,180		6	317	C	3,010	549	

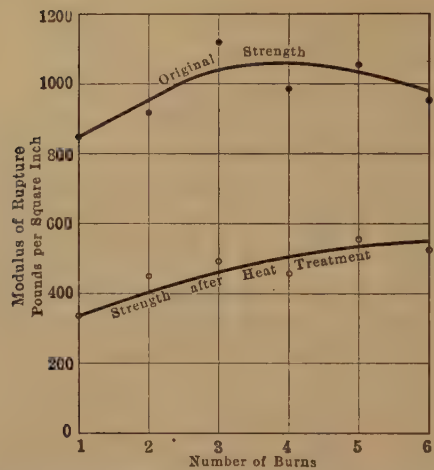

FIG. 26.—VARIATIONS IN THE CROSS-BREAKING STRENGTH OF BRICK OF REGULAR GRIND UPON REPEATED BURNING, AND THE EFFECT UPON THE STRENGTH OF A CERTAIN HEAT TREATMENT, CONSISTING OF SLOW HEATING TO 600° C. FOLLOWED BY COOLING IN THE AIR.

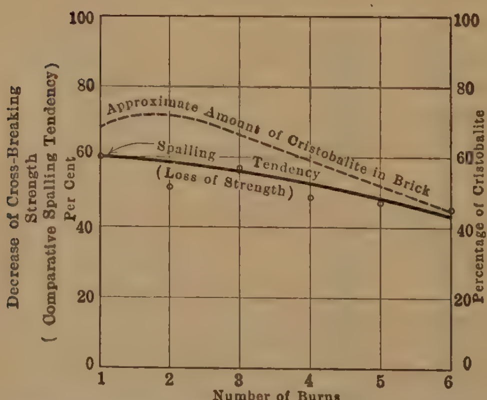

FIG. 27.—DECREASE IN CROSS-BREAKING STRENGTH EXPRESSED IN PER CENT. DUE TO HEAT TREATMENT DESCRIBED. THIS PERCENTAGE DECREASE IS ASSUMED TO REPRESENT THE COMPARATIVE SPALLING TENDENCY.

TABLE 23.—*Summary of Cross-breaking Tests Showing Loss of Strength due to Heat Treatment*

Burn	Modulus of Rupture, Pounds per Square Inch		Loss of Strength due to Heat Treatment	
	Before Heat Treatment	After Heat Treatment	Pounds per Square Inch	Per Cent. (= Comparative Spalling Tendency)
1	848	339	509	60.0
2	919	444	475	51.8
3	1,120	489	631	56.2
4	883	458	425	48.1
5	1,050	558	492	46.9
6	951	552	429	45.1

Results.—The experimental data verify the conclusion that the spalling tendency of silica brick diminishes on repeated burning. In Fig. 27 there is a marked correspondence between the curve representing approximately the cristobalite content of the brick and that representing the spalling tendency. The latter, however, does not decrease as rapidly as the former. It is evident that, while reburning the brick diminishes

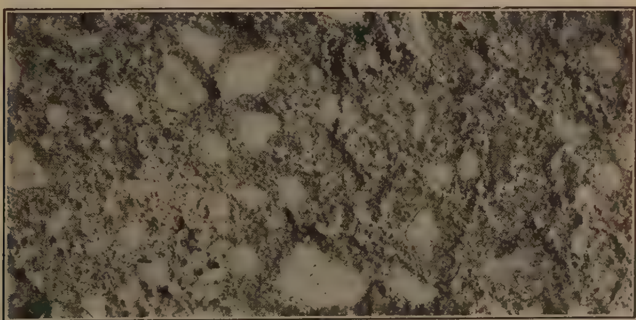

FIG. 28.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF REGULAR GRIND, BURNED ONCE IN SILICA BRICK KILN. × 4.

the spalling tendency, the change therein from 60 per cent. on a first burn to 45 per cent. after a sixth is too slight to be of any commercial importance.

A number of first-quality fire-clay brick, subjected to the same heat treatment, lost about 4.5 per cent. in strength. This result agrees with the well-known fact that the spalling tendency of clay brick, at the temperature of this test, is but a fraction of that of commercial silica brick.

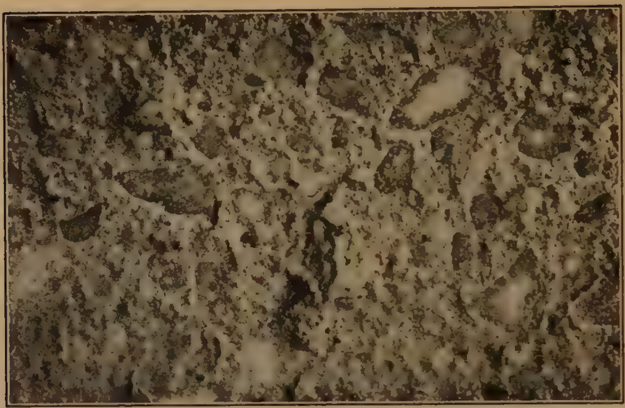

FIG. 29.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF REGULAR GRIND, BURNED SIX TIMES. × 4.

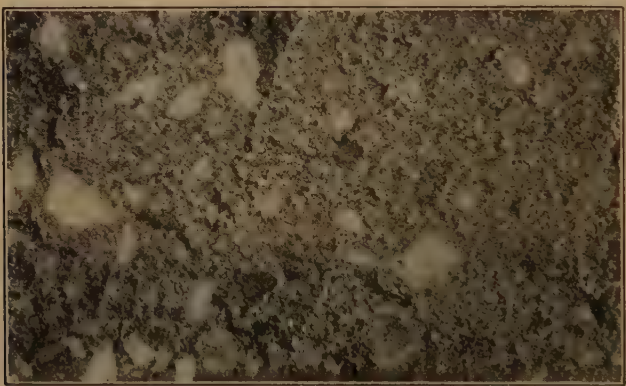

FIG. 30.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF SHAPE GRIND, BURNED ONCE. × 4.

FIG. 31.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF SHAPE GRIND, BURNED SIX TIMES. × 4.

Changes in Texture of Brick Due to Burning

The photomicrographs shown in Figs. 28 to 31 bring out very clearly the change in the texture of the brick brought about by repeated burning. After a single burning the larger grains of quartzite appear nearly white; after six burnings the grains stand out much more distinctly in the pictures and have become darker and apparently more dense. A number of grains are seen in which the center is still white, while the margins are altered in the manner described.

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DISCUSSION

J. W. RICHARDS, So. Bethlehem, Pa.—The paper is certainly an exceedingly valuable one and it gives detailed information which should be utilized in the following way. There are probably twenty different circumstances or conditions under which silica bricks are used. Sometimes they are used under high pressure, at others under low pressure, at times for low temperatures and others for very high temperatures. If the methods of this paper are followed out, and if the precise mechanical and thermal properties are determined and can be reproduced, the industry can furnish the proper brick for any particular purpose. There is at present only an imperfect classification made as to properties and uses, but we need a much more scientific classification of the uses to which the bricks are to be put and then should manufacture the bricks suited particularly for each of those purposes. There is one property which is not mentioned in the paper and which is of considerable importance, that is, the electrical conductivity. There may be other properties besides, which a good research laboratory can determine, and when we have listed all the various properties and noted the way in which they can be reproduced, the maker can then specify his exact requirements and the corresponding quality of brick can be selected or made to meet them.

H. D. HIBBARD, Plainfield, N. J.—About a year ago I had some experience in another country, in melting steel, where silica bricks made in that country failed, but in a different way from those in America. Here, when the bricks get too hot, they are partly melted and string down from the roof, but there when the heat was too high pieces dropped off from them, and I would like to ask why they acted that way.

W. F. Rochow, Pittsburgh, Pa.—An underburned silica brick if burned rapidly will expand permanently to about the same degree as a properly burned silica brick, but would be weakened. It is possible that the difficulty you describe could have been caused by too rapidly burned brick.

H. D. Hibbard.—It seemed to me that the trouble was that the lime was not thoroughly mixed with the ganister so that in the cementing material between the grains of ganister the lime was very high, perhaps 10 or 20 per cent. and the cementing material was therefore too easily fusible. Then when it got hot enough, it was softened, lost its strength and let the brick drop to pieces. Since then in thinking it over, it appeared probable that the trouble was in insufficient grinding and mixing of the materials in the pans. Thus the cementing material might, if it had a high enough percentage of lime, have a fusion point as low, perhaps, as 1,200° C.; if it got more silica worked in by continued grinding it might rise to 1,300, 1,400, 1,500 or perhaps 1,600°. The total or ultimate percentage of lime in the brick does not give the composition of the cementing material or the mortar between the silica grains on which the integrity of the brick really depends. It is like a chain, the strength is that of the weakest link, and the resistance of a silica brick to heat depends on the fusibility of the most fusible part, which, in this case, was the cementing material between the grains.

W. F. Rochow.—That is a possible explanation. There are three eutectics of mixtures of lime and silica, and a silica brick having a much higher percentage than 3 per cent. of lime will be lacking in refractoriness while one having less than, say, 1.4 per cent., will be mechanically weak.

H. D. Hibbard.—The point I wanted settled, if possible, was whether the lime and silica which formed the mortar between the grains had the proper degree of refractoriness or not, and if not, whether it could have been cured by prolonged mixing and grinding?

W. F. Rochow.—Such a condition would be possible, but it is not at all likely with brick manufactured in this country in accordance with the present methods, because after the quartzite rock is ground and mixed with the lime as milk of lime, it is usually allowed to stand for some time before it is used, so that the milk of lime would have a tendency to spread uniformly throughout the mass even if the mixing could not have been thorough in the grinding pan.

H. D. Hibbard.—Without any mixing effect?

W. F. Rochow.—No, not without any mixing. Thorough mixing

in the grinding pan, however, is readily accomplished without great difficulty.

J. B. Umpleby, Washington, D. C.—I have been particularly interested in this paper because I know something of the work of Dr. Fenner on the stability of the silica series in the Geophysical Laboratory in Washington, and although I am not sure, I suspect that in his original investigation, he did not foresee the application of the work to the manufacture of silica brick. It is probably one more instance of an abstract study having a definite commercial application unforeseen by the man who originally starts the investigation. I cannot but wonder if much more of the work being done at the Geophysical Laboratory on the two and three component systems will not have a definite application apart from adding to our knowledge of mineralogic phenomena.

The Genesis of Asbestos and Asbestiform Minerals*

BY STEPHEN TABER, PH. D.,† COLUMBIA, S. C.

(New York Meeting, February, 1917)

CONTENTS

Introduction

The term asbestos, as commonly used, includes half a dozen minerals all having a well-developed fibrous structure, but differing in chemical composition and in some of their physical properties. In its strict application the name is limited to the fibrous varieties of the monoclinic amphiboles. Commercially, however, the most important of the asbestiform minerals is chrysotile, a fibrous variety of serpentine. About 95 per cent. of the asbestos used in manufacturing is chrysotile, and it commands a much higher price than any of the other fibrous minerals now on the market.

Although the production of asbestos has increased rapidly in recent years, comparatively little has been published concerning its origin. The present paper is preliminary in its nature, and therefore does not pretend to exhaust the subject. The ideas herein developed are the result of field investigation, laboratory experiments in the growth of fibrous

* Manuscript received by the Secretary on June 26, 1916.
† State Geologist and Professor of Geology at the University of South Carolina.

crystals, and the examination of asbestos specimens from the more important producing districts, as well as a careful study of the work of previous investigators.

Asbestiform Minerals

Chemical and Mineralogical Relations

The table given below contains the group name and species of the various fibrous minerals to which the term asbestos is commonly applied. The name of each mineral is accompanied by its chemical formula as given in Dana's *System of Mineralogy.*

Asbestiform Minerals
- Amphibole Group
  - Orthorhombic Section
    - Anthophyllite $(Mg,Fe)\,SiO_3$
  - Monoclinic Section
    - Tremolite $CaMg_3(SiO_3)_4$
    - Actinolite $Ca(Mg,Fe)_3(SiO_3)_4$
    - Crocidolite $NaFe(SiO_3)_2.FeSiO_3$
- Serpentine $H_4Mg_3Si_2O_9$
    - Chrysotile
    - Picrolite

All of the minerals listed in the table have both fibrous and non-fibrous varieties. The fibrous varieties never show the outward form of crystals except as pseudomorphs, but the fact that they are crystalline is proved by their optical properties and other evidences of regular internal molecular structure. Well-developed euhedral crystals of tremolite and actinolite are quite common, especially where these minerals have developed in metamorphic limestones or have grown in cavities. Euhedral crystals of anthophyllite are very rare, they have never been reported for crocidolite, and serpentine has no crystal form of its own, though frequently occurring as a pseudomorph after other minerals.

Tremolite and actinolite usually assume more or less elongated prismatic forms. Often the crystals occur in aggregates of parallel prisms or in groups having a radial arrangement. There is every gradation in structure from the normal prismatic forms through the slender and columnar into the fibrous. The habit of anthophyllite is similar, except that radiating groups are relatively more abundant and the structure is more inclined to be fibrous. Crocidolite is usually fibrous but is sometimes massive or earthy.[1] Serpentine is generally massive in appearance, frequently fibrous, and sometimes foliated. Examined under the micro-

[1] E. S. Dana: *A System of Mineralogy,* Ed. 6 (1914), 400.

scope, even the massive varieties commonly show a more or less finely fibrous structure, though occasionally the serpentine is in the form of minute scales. The amphiboles possess a well-developed prismatic cleavage intersecting at angles of 54° to 56°, while the chrysotile variety of serpentine is said to have a prismatic cleavage of 50°.

In chemical composition these minerals are characterized by the absence of aluminum and by the presence of magnesium in all except crocidolite, which contains instead a notable percentage of sodium. Tremolite and actinolite are distinguished by the presence of calcium as an essential element. The fibrous amphiboles are all normal metasilicates, and serpentine is a hydrous orthosilicate. These minerals are all secondary, being formed from other minerals, and for the most part they are confined to metamorphic rocks, which have had in some instances an igneous and in others a sedimentary origin.

Physical Properties

The commercial value of asbestos and the uses to which the different varieties may be put are dependent upon the physical properties; fineness, length and flexibility of fiber, tensile strength, and heat- and acid-resisting properties.

All of the different varieties of asbestos may be split up into exceedingly fine fibers, and when the finest obtainable fibers, having a diameter of 0.002 mm. or less, are examined under the microscope, they are usually seen to be made up of still smaller fibers. The number and fineness of the smallest fibers distinguishable under the microscope increase with every increase in the power of magnification, and there is no apparent limit to this subdivision.

Bunches of amphibole asbestos, of the slip-fiber type, are sometimes found with a length of 2 or 3 ft., but when these bunches are split up into smaller and smaller bundles, the length of the bundles tends to decrease with their diameter. This indicates that the long bundles probably consist of shorter interlaced and overlapping bundles of fibers. In chrysotile veins the fibers seldom have a length of over 2 or 3 in., and the bulk of the material mined averages less than ½ in. The longer fibers are commonly cut by planes of jointing, sometimes almost invisible, and sometimes marked by the presence of foreign material.

Chrysotile fibers are the most flexible, but fibers of crocidolite are almost if not equally as good in this respect. The other varieties of amphibole asbestos are, on the whole, less flexible, and, while this property varies greatly in each of the different species, the fibers of anthophyllite seem to be somewhat more flexible than similar fibers of tremolite or actinolite. Picrolite is similar to chrysotile except that the fibers are coarser and more brittle. The latter mineral is of no commercial im-

portance. Little is known concerning the tensile strength of the different species of asbestos, but the experiments of Haussman[2] indicate that crocidolite has the greatest tensile strength, and chrysotile probably ranks second.

Chrysotile easily withstands temperatures of 2,000° to 3,000° F., while with some varieties a temperature of 5,000° F., produces no visible effect.[3] At red heat it gives up water and becomes brittle. Anthophyllite under the same conditions remains practically unaltered. Tremolite and actinolite fuse at somewhat lower temperatures, while crocidolite fuses so easily that it is useless for many purposes where asbestos is commonly employed.

Chrysotile is readily attacked by relatively weak acids, being decomposed with the separation of silica; the amphibole varieties, especially tremolite and anthophyllite, are very resistant even when subjected to the action of concentrated acids.

Types of Asbestos

Three types of asbestos fiber are recognized—cross-fiber, slip-fiber and mass-fiber. Cross-fiber asbestos occurs in veins with the fibers extending transverse to the strike of the vein. Usually the fibers are approximately perpendicular to the inclosing walls, frequently they are more or less oblique and occasionally they are curved or abruptly bent. Slip-fiber is found along fault planes, often accompanied by slickensides, and the direction of the parallel fibers records the direction of displacement. The amount of displacement is usually small. Slip-fiber asbestos is commonly distributed in thin layers that are not as a rule continuous for any considerable distance, but occasionally it is found in masses a foot or more in thickness. All gradations are to be found between the cross-fiber and slip-fiber types. Mass-fiber asbestos occurs in fibrous bundles or groups varying in size and orientation. The fibers may be parallel but are usually divergent and often radiating. In the type occurrence at Sall Mountain, Georgia, mass-fiber asbestos makes up practically the entire rock mass. Gradations between the mass-fiber and slip-fiber types probably exist, although, so far as the writer is aware, none has been described.

Anthophyllite has been reported as occurring in all three ways, and it is the only variety of asbestos known to occur as mass-fiber. The asbestiform varieties of tremolite and actinolite are practically limited to the slip-fiber type although two occurrences of true asbestos in the

[2] J. F. L. Haussman: *Handbuch der Mineralogie* (1847), 734.

[3] Fritz Cirkel: Chrysotile-Asbestos, Its Occurrence, Exploitation, Milling and Uses, Ed. 2., *Canada Department of Mines, Mines Branch, Report* No. 69 (1910), 30.

form of cross-fiber have been described.[4] Crocidolite has been reported only in cross-fiber veins, but there is every reason for believing that it may be found also as slip-fiber. Very little is known as yet concerning the occurrence of crocidolite. Most of the commercially important deposits of chrysotile are of the cross-fiber type, but slip-fiber is also very common.

Origin of the Fibrous Structure

Views of Previous Investigators

The wonderfully fibrous structure of the asbestos minerals has aroused interest and curiosity ever since they were first discovered, and yet little by way of explanation of this peculiar structure is to be found in geologic literature. Why these minerals sometimes possess an asbestiform structure and sometimes do not is a question on which few geologists have advanced an opinion.

Merrill[5] expresses the belief that the fibrous structure in the case of anthophyllite "and of the true asbestos as well, is due, in many instances at least, to a process of shearing—is, in fact, an exaggerated form of the process of uralitization."

Heddle[6] and Merrill[7] have described instances in which the fibrous structure of anthophyllite has been accentuated by weathering. Hopkins[8] apparently adopts this view in part for the deposits at Sall Mountain, Georgia.

Heddle[9] argued that chrysotile must be pseudomorphic, since serpentine is a mineral incapable of assuming a chrystalline form, but, after describing the chrysotile found near "Hesta Ness, which terminates the south side of the bay of Gruting, in Fetlar, Shetland," he suggests that "As the magnetite here is itself saturated with a serpentinous basis, there is a possibility that the fibrous structure of the chrysotile may be the result of its protrusion, by an exfiltration process, through the interstices of the granular magnetite, which interstices acted upon the mineral in a manner similar to the holes in a draw-plate."

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[4] M. F. Heddle: Chapters on the Mineralogy of Scotland, *Transactions of the Royal Society of Edinburgh* (1877–78), **28**, 503–504 and 531. Also quoted by G. P. Merrill in Notes on Asbestos and Asbestiform Minerals, *Proceedings of the U. S. National Museum* (1895), **18**, 286–287.

[5] G. P. Merrill: Notes on Asbestos and Asbestiform Minerals, *Proceedings of the U. S. National Museum* (1895), **18**, 289.

[6] M F. Heddle: *Op. cit.*, 502 and 504.

[7] *Op. cit.*, 288.

[8] O. B. Hopkins: Report on the Asbestos, Talc and Soapstone Deposits of Georgia, *Geological Survey of Georgia, Bulletin* No. 29 (1914), 106.

[9] M. F. Heddle: *Op. cit.*, 535–536.

Dana,[10] in his *Systematic Mineralogy*, gave the following explanation of the origin of fibrous structure: "When a solution is spread thinly over a large surface, minute crystalline points encrust the whole, and if the solution be gradually supplied, as crystallization goes on, it is obvious that the minute points may elongate into crowded prisms of fibres, producing a fibrous structure. Such a structure is common in narrow seams in rocks, and the fibres are usually elongated across the seam."

Cirkel[11] quotes this explanation and then adds: "It seems that this paragraph refers to chrysotile; because the crystals of the same are elongated across the seam in the precise mode described." Dana's hypothesis is probably the correct explanation of the fibrous structure seen in the stalactitic, mammillary, and similar forms that have been deposited in open spaces from a thin layer of solution, but there is no evidence that any of the varieties of asbestos have been formed in this way.

The hypothesis advanced by Cirkel[12] is that, "chrysotile may have been an extreme example of crystallization which took place under conditions of high temperature and extreme pressure."

While the writer has not had the opportunity of making an exhaustive search of the literature, he believes that the various explanations of fibrous structure listed above include the more important hypotheses so far advanced by scientific investigators.

Discussion of the Evidence

Microscopic examination shows that bundles consisting of thousands of small fibers of chrysotile or amphibole asbestos behave as crystal units, an entire bundle exhibiting the same optical properties as any one of the component fibers. The fact that all of the asbestiform minerals have prismatic cleavage suggests that the fibrous structure may be due to an abnormal development of the cleavage or at least that the separation of the fibers takes place along cleavage planes. This hypothesis is supported by the prismatic form shown by most of the smallest obtainable fibers of amphibole asbestos, when they are examined under the microscope, but fibers of chrysotile usually possess irregular polygonal or rounded cross-sections. The highly fibrous structure of the asbestiform minerals is not, however, a crystallization phenomenon in the sense that it is due solely to the inherent physical properties of the crystal molecule, for all minerals having asbestiform varieties occur in non-fibrous as well as fibrous forms. Moreover, there are intermediate gradations between

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[10] J. D. Dana: *Systematic Mineralogy: Section II, Theoretical Crystallogeny* (1850), 124.

[11] Fritz Cirkel: *Op. cit.*, 93.

[12] *Loc. cit.*, 93.

the non-fibrous and the asbestiform varieties of serpentine and of the amphiboles with the possible exception of crocidolite.

Why is it, then, that these minerals sometimes have a fibrous or asbestiform structure, while at other times they are non-fibrous? The answer to this question, the writer believes, is to be found chiefly in the physical conditions under which the minerals were formed.

The primary minerals of igneous rocks never show an asbestiform structure even in the case of minerals possessing perfect prismatic cleavage, and this statement is true in general for all minerals that have grown in free contact with supersaturated solutions. The latter generalization will be objected to by those who believe that cross-fiber veins have been deposited from solutions circulating along open fractures, but the writer has found no evidence supporting such an hypothesis. This question, however, is later discussed in detail under the origin of cross-fiber veins.

There are many minerals that have fibrous varieties. Most of them do not have a prismatic cleavage, some have no cleavage at all, and at least one such mineral, limonite, is always amorphous and never crystalline. The fibrous varieties of these minerals differ from asbestos chiefly in having fibers that are coarser, more brittle and not so easily separable. With fibers of equal size the flexibility and tensile strength are probably determined very largely by chemical composition.

In the case of those minerals that do not have a prismatic cleavage and do not normally have a columnar or prismatic habit, the fibrous structure is obviously due to physical conditions which have prevented crystal growth except in one direction. If a mineral having perfect prismatic habit and cleavage develops under physical conditions that limit growth to a direction parallel to the principal axis, then the fibrous structure may be accentuated to such an extent as to make the mineral truly asbestiform. A careful comparison of the occurrences of all the common fibrous minerals indicates that the commercially important deposits of the asbestiform minerals have been formed under the same physical conditions that result in the development of a well-defined fibrous structure in other minerals.

That the peculiar structure of asbestiform minerals is usually due to the accentuation of a normal prismatic habit and cleavage through the limitation of crystal growth by physical conditions is the author's thesis. Recently he conducted a series of laboratory experiments with the object of determining the essential conditions for the development of fibrous crystals. Some of these experiments have been referred to in another paper,[13] and it is planned to publish a detailed account of the others in the near future. All questions connected with the problem have not been

[13] Stephen Taber: The Growth of Crystals under External Pressure, *American Journal of Science*, Ser. 4 (1916), **41**, 532–556.

cleared up, but some of the facts established have an important bearing on the present discussion.

Conclusions

The shape of a growing crystal is controlled by one or more of three independent factors, namely: (1) the tendency to assume a regular polyhedral form because of the forces of surface tension and molecular orientation; (2) the relative and absolute magnitude of the external forces resisting growth in different directions; and (3) the accessibility of the material from which the crystal is built.

1. Some crystalline substances normally have a prismatic or columnar habit because of the intermolecular forces controlling the development of crystal faces, and under favorable conditions slender acicular or hair-like crystals may result; but a highly fibrous or asbestiform structure is never developed even in the case of minerals having perfect prismatic cleavage, without the assistance of one of the other two factors.

2. If crystals are under unequal pressure, growth may be limited to the direction of least pressure. When crystals grow through the addition of new material from solutions, growth takes place only where the solutions are supersaturated with respect to the crystal surfaces with which they are in contact, and growth must continue regardless of resisting forces as long as supersaturation is maintained. But pressure increases the solubility of all substances that go into solution with decrease in volume, and therefore any increase in pressure must be accompanied by a corresponding increase in the concentration of the solution, if growth is to continue. Crystals subjected to unequal pressure may even go into solution on the surfaces that are under the greater pressure, while at the same time deposition is taking place along the line of least pressure; and if the normal habit of the crystals is columnar, then those crystals that are oriented with their longer axes parallel to the least pressure will tend to grow at the expense of those that are less favorably oriented.

In the same way unequal pressure may control the direction of growth and therefore the shape of crystals, when crystallization from a state of fusion takes place, with increase in volume. There are, however, only a few substances, such as ice, that solidify with expansion in volume. Unequal pressure may also determine the shape of secondary minerals formed without going into solution, for, when the alteration of one mineral to another is accompanied by increase in volume, pressure tends to prevent the alteration, and therefore the effect of unequal pressure may be to limit alteration and the growth of the new mineral to the direction of least pressure.

Unequal pressure always sets up shearing stresses, and in the case of cleavable minerals these stresses are relieved most easily by slipping

along cleavage planes, thus making the cleavage more pronounced. The effect of shearing on minerals with prismatic cleavage may therefore be a factor of some importance in producing a fibrous structure.

The development of fibrous structure is undoubtedly in some instances to be attributed to growth under unequal pressure, but this is not the commonest cause of the phenomenon.

3. The study of fibrous minerals and their occurrence in nature, as well as the results of laboratory experiments in the growth of fibrous crystals, leads to the conclusion that in most cases the development of fibrous structure has been due primarily to the fact that the material from which the growing crystals were built was accessible only in one direction. This is the most probable reason why secondary minerals are so frequently fibrous. For example, when serpentine is formed from olivine, the alteration sets in from the exterior of the crystal and from cracks, and the resulting serpentine is usually in the form of microscopic fibers that develop normal to these surfaces. The fibrous form is gradually assumed by the microscopic crystals, because growth takes place only at their base where they may receive additions of new material as alteration of the olivine progresses. In a partly altered olivine the microscopic veinlets of serpentine are frequently similar in appearance and structure to the larger veins of cross-fiber chrysotile.

The efficacy of this method of producing fibrous structure in crystals, when growth takes place through the addition of new material from supersaturated solutions, has been proved experimentally.[14] Cups of porous porcelain were partly immersed in concentrated solutions of copper sulphate, alum and other salts. The solutions were drawn up through the capillary pores allowing evaporation to take place from the exposed surfaces. After a day or two, crystal growth began with the formation of irregular spots or thin crusts on the upper surfaces of the cups, and these gradually increased in size and thickness. Later, groups of fibrous crystals could be observed under the crusts slowly pushing them outward. The fibers continually increased in length as long as the material for growth was available, and at the end of 8 months some were over 2 cm. in length.

In these experiments crystallization also occurred at a few favorable places within the walls of the cups, and, as growth continued, the crystals developed sufficient pressure to produce rupture. The fractures thus formed were gradually extended and widened by the growth of fibrous veins, closely resembling in structure the cross-fiber veins of chrysotile and of asbestos, as well as the similar veins of fibrous calcite, gypsum and other minerals. In some of the experiments where alum was used, it was possible to change the color of the solution by adding chrome alum in varying amounts, and thus produce banded veins.

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[14] Stephen Taber: *Op. cit.*, 545–546.

Cavities or open spaces were usually present under the central portions of the larger crusts and within the veins, and these openings were sometimes partly lined with crystals normal in habit instead of fibrous. This lack of uniformity is to be explained by the fact that supersaturation of the solution was produced and maintained by evaporation, and this process was retarded or prevented where the crusts protected the solution from contact with the air. Better results were obtained when supersaturation was induced by cooling, so that practically uniform concentration could be obtained over the entire crystallizing surface.

The experiments so far conducted by the writer indicate that fibrous crystals may be produced in the manner described above only from substances that go into solution with decrease in volume.[15] Most, if not all, of the rock-forming minerals belong to this class.

The development of the fibrous structure is probably aided by small adjustments or slips along the surfaces of the fiber prisms and cleavage planes when the latter are present, since under the conditions of growth it is not conceivable that new material can be added at the end of each prism of a large group continuously and at the same uniform rate. Indeed observation shows that such slipping does take place in the groups of crystals grown in the laboratory, and both megascopic and microscopic examination of fibrous minerals furnishes evidence of similar slipping.

The fibers obtained in the experiments just described are brittle and therefore rather difficultly separable, but occasionally it is possible to procure individual fibers having a length of several millimeters and a thickness of less than 0.001 mm. Such fibers of copper sulphate and alum are flexible and somewhat elastic. The columnar or fibrous ice crystals found in clayey soils grow in practically the same way as the fibrous crystals described above, and Merrill has observed the growth of fibrous incrustations of gypsum on the walls of caves,[16] but did not attribute their fibrous structure to the manner of growth.

Origin of Cross-fiber Veins

Many different theories, some of them rather hypothetical, have been advanced by geologists to explain the origin of veins of cross-fiber asbestos. These theories have been reviewed recently in three publica-

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[15] Attempts were made to obtain similar results with ammonium chloride and other salts that go into solution with expansion in volume, but no veins were formed and the crusts were enlarged only through the addition of new material on the outer exposed surfaces, the solutions reaching these surfaces through capillary pores in the crystalline mass. In some instances tubular, hair-like crystals were formed, 0.01 mm. or less in diameter, and these apparently grew only at their outer ends, the new material being furnished by solutions drawn up through the capillary tubes.

[16] G. P. Merrill: On the Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings of the U. S. National Museum* (1894), **17**, 77–81.

tions, and it is not necessary, therefore, to give them in detail here. That no general agreement has been reached, is shown by the diversity of opinion expressed in the conclusions drawn by the authors of the three monographs just cited.[17] Most of the theories are limited to chrysotile veins in serpentine and could not be applied to veins of crocidolite and other varieties of amphibole asbestos. Since the origin of chrysotile is closely related to the origin of the massive serpentine with which it is associated, the origin of chrysotile veins will be considered separately.

Chrysotile Veins

All of the different theories attempting to explain the origin of chrysotile veins fall under one or the other of the following classes:

1. The veins were deposited in open fissures: (*a*) by circulating solutions; or (*b*) through the segregation of material; or (*c*) infiltration of serpentinous solutions from the walls.

2. The veins were formed by the replacement of the wall rock along small fissures that served as channels for the circulation of solution.

3. The veins are portions of the serpentine that have crystallized *in situ*, the crystals growing outward from preëxisting fractures through which water entered to alter the rock to serpentine.

All of these theories presuppose the presence of fractures, and much ingenuity has been used in accounting for the fractures. They have been attributed to: (1) the contraction of an igneous magma upon cooling and solidifying; (2) dynamic causes; (3) exfoliation, possibly aided by the increase in volume that accompanies the alteration of peridotite to serpentine; (4) shrinkage resulting from the loss of silica or other constituent; and (5) a partial dehydration of the serpentine.

It seems to the writer that there are serious objections to all of the theories outlined above, for, while some of them may explain many of the phenomena connected with the occurrence of asbestos veins, not one offers a complete and adequate explanation of all of the known facts. It is conceivable that some chrysotile veins may have been formed in open fissures, but it is mechanically impossible that all of them could have been formed in this way. Dresser[18] has pointed out the absurdity of this theory as applied to the chrysotile deposits of southern Quebec, Canada, where the veins, in places occupying over 10 per cent. of the entire rock, run in all directions from vertical to horizontal, and occasionally reach

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[17] Fritz Cirkel: *Op. cit.*

J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey, Memoir* No. 22 (1913).

O. B. Hopkins: *Op. cit.*

[18] J. A. Dresser: *Op. cit.*, 65.

a length as great as 100 ft. Pratt[19] and Diller[20] have described chrysotile veins in the Grand Canyon, Arizona, over 4,000 ft. below the rim, which extend horizontally parallel to the bedding of the inclosing rocks for distances of 150 ft. or more.

Moreover, there is evidence, in many instances at least, that the formation of the chrysotile veins and the alteration of the inclosing rock to form serpentine are contemporaneous processes. In the Black Lake district of Quebec, where the alteration of peridotite to serpentine is not complete, the chrysotile veins are bordered on both sides by bands of massive serpentine, and the width of the veins is proportional to the width of the inclosing bands. By careful measurements Dresser[21] found the ratio of the chrysotile vein to the entire band of serpentine to be 1:6.6. The alteration of peridotite to serpentine is accompanied by such an increase in the volume of the rock that it would be impossible for fissures to remain open during the process.

Little if any evidence has been offered in support of the replacement hypothesis. Where chrysotile veins occur in limestone, as they do in Arizona, it is possible for some replacement of the inclosing rock to take place, but this occurrence is exceptional. Nearly all chrysotile veins are found in massive serpentine, having a chemical composition that is practically identical with that of the vein material, and no reason has yet been given to explain why serpentine should replace serpentine of the same chemical composition. Moreover, chrysotile veins do not possess any of the characteristics that commonly distinguish replacement veins from other veins. They have their own peculiar structure and never show any trace of an inherited structure. Serpentine frequently occurs as a pseudomorph after other minerals, but pseudomorphs are never found in chrysotile veins. Replacement veins are characterized by great irregularity in width and lack of sharp boundaries, while chrysotile veins frequently show remarkable uniformity in width, and always have well-defined walls.

The theory that "the veins are crystallized portions of the serpentine walls, and that the crystals (fibers) have grown outward from the original crevices which are now represented by partings of iron ore found near the center of the veins,"[22] was evidently advanced in order to avoid some of the more obvious objections to the preceding hypothesis, but it completely fails to explain many of the characteristic phenomena of chrysotile veins. If the fibers grew outward in the way here postulated, some of

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[19] J. H. Pratt: Asbestos, *Mineral Resources*, 1904, *U. S. Geological Survey* (1905), 1137–1140.

[20] J. S. Diller: Asbestos, *Mineral Resources*, 1907, *U. S. Geological Survey* (1908), pt. II, 720–721.

[21] J. A. Dresser: *Op. cit.*, 59–60.

[22] J. A. Dresser: *Op. cit.*, 66.

them would unquestionably penetrate the massive serpentine for greater distances than others, thus giving irregular boundaries to the veins, but chrysotile veins have sharply defined boundaries and are easily separated from the wall rock.

In discussing the deposits of southern Quebec, Dresser states: "the facts are self-evident that zones of the country rock have been altered to serpentine and proportionate parts of these have taken the form of asbestos veins."[23] But, if the veins are merely "portions of the serpentinized bands which have crystallized *in situ*"[24] why is the ratio of chrysotile to massive serpentine limited? Why are the veins limited to a width of 1, 2 or, in rare instances, 3 in.?

This theory does not satisfactorily explain the angular inclusions of massive serpentine that frequently mark the central parting of chrysotile veins and are also to be found irregularly distributed through them. It affords no explanation of the occasional presence of more than one parting or of bends in the fibers such as are shown in Figs. 4, 5 and 6. It does not explain the gradation of cross-fiber into slip-fiber as illustrated in Figs. 3 and 7. Most of the objections here given are equally applicable to the other theories of vein formation so far discussed, and, in view of all the facts, the conclusion is inevitable that none of these theories furnishes a true explanation of the origin of chrysotile veins.

As already noted in this paper, fibrous veins with structural features analogous to those of chrysotile veins have been produced in the laboratory, where their formation and growth could be carefully observed. The results obtained from these experiments, supplemented by a study of fibrous veins in different kinds of rock, have led the writer to certain definite conclusions. These conclusions have been formulated into a theory of vein formation that seems to explain satisfactorily the phenomena connected with chrysotile veins. Briefly stated, this theory is that all cross-fiber veins are formed through a process of lateral secretion, the growing veins making room for themselves by pushing apart the inclosing walls; and that the fibrous structure is to be attributed largely to the physical conditions which have limited crystal growth to a single direction. In the case of the asbestiform minerals, the fibrous structure is accentuated by a normal prismatic habit and cleavage.

In individual occurrences it may be difficult or impossible to determine why the fibrous mineral was taken into solution, the cause of its redeposition and the other details of origin; but, in the case of serpentine, pressure due to expansion in volume is probably the controlling factor.

Serpentine is a secondary mineral resulting, as a rule, from the altera-

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[23] J. A. Dresser: Asbestos in Southern Quebec, *Trans.* (1915), **50**, 957.

[24] J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey, Memoir* No. 22 (1913), 65.

tion of preëxisting silicates of magnesia. The alteration is seldom one of simple hydration, for usually some material is added or subtracted, and in many cases the constituents out of which serpentine is formed are derived from several different minerals. Serpentine is therefore found in rocks that are either of sedimentary or of igneous origin. Locally it may be the dominant mineral thus forming serpentine rocks. Such rocks are derived chiefly from igneous rocks of the pyroxenite-peridotite family which contain a high magnesian content. Although serpentine is secondary after many minerals, the most important source is olivine followed by the magnesian pyroxenes and amphiboles.

The alteration to serpentine is accompanied by an increase in the volume of the rock mass, which results in the development of pressure whenever there is resistance to expansion. This pressure can not be explained, however, by attributing it to a chemical reaction taking place with increase of volume, as, for example, when plaster of Paris combines with water and sets, because, in many cases where serpentine is formed from anhydrous minerals, there is an actual decrease in the total volume of the system, if the volume of the water is taken into consideration. For instance, when serpentine is formed from olivine according to the equation:

$$3Mg_3FeSi_2O_8 + 6H_2O + O = 3H_4Mg_3Si_2O_9 + Fe_3O_4.$$

Neglecting the oxygen, the volume changes in this reaction are as follows: 281.3 parts of olivine combine with 108 parts of water to give 321 parts of serpentine and 44.5 parts of magnetite. Hence there is an increase in the volume of solid amounting to approximately 30 per cent., while there is a decrease of about 6 per cent. in the volume of the system as a whole. The pressure resulting from this reaction is, therefore, due to the fact that water in a liquid or gaseous state is able to penetrate the rock mass through capillary and subcapillary openings while the solid serpentine formed does not escape in like manner.

The water may penetrate the rock through fractures, along the contact between mineral grains and along the cleavages of minerals. As the alteration proceeds, readjustments in the rock mass, due to increase in volume, result in the formation of new fractures. Some of the water possibly travels for short distances through a process of diffusion or trading, by which a molecule of serpentine gives up its water to a neighboring anhydrous molecule and then takes up more water.

The maximum pressure that may be developed locally as a result of the alteration must be enormous, since, in most cases, it is limited solely by the resistance offered to expansion. Pressure alone can have no direct effect in preventing a reaction that takes place with decrease in volume; and, whether the reaction will take place or not is determined entirely

by the partial pressure of the water vapor and the vapor pressure of the serpentine.

The development of pressure in this way is nicely illustrated by the following laboratory experiment:

A porous porcelain cup was partly filled with anhydrous cupric chloride ($CuCl_2$), after which the open end was sealed with paraffine. The cup was then placed, together with a beaker containing water, under a bell jar, and allowed to stand undisturbed. Water vapor was absorbed through the capillary pores of the cup with the formation of the hydrous salt ($CuCl_2.2H_2O$). At the end of 5 days small fractures were observed, and these gradually increased in length and width as the volume of the salt continued to increase. The fracturing of bottles by salts that take up water of crystallization from the atmosphere is a common phenomenon in chemical laboratories.

The pressure developed as a result of the increase in rock volume when serpentine is formed is important as a factor in the formation of chrysotile veins, chiefly because the solution of the serpentine and its redeposition in the form of chrysotile seem to be determined very largely by pressure. The pressure is at a maximum where serpentine is being formed, and here the mineral is most readily taken into solution. The separation of serpentine from solution is accompanied by expansion in volume, and therefore takes place only where the forces opposing expansion are not prohibitive.

All existing fractures or joints, whether dynamic in origin or due to contraction on cooling, are favorable places for the deposition of chrysotile, because as a rule less force is required in pushing apart the walls of existing fractures than is necessary in forming new ones. The alteration of a rock to serpentine begins along the existing fractures which divide the rock mass into blocks of variable size. As the alteration slowly penetrates inward from the surfaces of a block, strains are set up between the expanded outer shell and the unaltered central part. These strains tend to produce exfoliation fractures, and, although the resisting pressure is probably sufficient in most cases to prevent the formation of open fissures, this pressure is locally reduced by the tendency to fracture, thus permitting the separation of serpentine from solution and the growth of new veinlets of chrysotile.

The chrysotile deposits of Thetford, Canada, furnish good illustrations of veins that have originated in this way,[25] and Merrill has described veins in the serpentine at Montville, N. J., that are difficult of explanation under any other hypothesis. At the latter locality the serpentine has been formed through the hydration of nodules of lime-magnesian

[25] J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey Memoir*, No. 22 (1913), 58–59 and Fig. 7.

pyroxene occurring in dolomite, and the narrow chrysotile veins are in a general way parallel to the surface of the original nodule of pyroxene.[26]

Since the pressure due to expansion prevents the formation or maintenance of appreciable open spaces, water must reach the unaltered rock, at least partly, through very small capillary and subcapillary openings. Under these conditions circulation is extremely slow and the movement is probably limited almost entirely to a single direction, *i.e.*, toward the unaltered portion of the rock. It is not necessary to assume the presence of solutions circulating toward the chrysotile veins in order to explain their growth, as the transfer of serpentine through the short distance from the place of its formation to the walls of the growing vein is probably due chiefly to diffusion of the material through the solution.

If pressure due to expansion is the principal factor in bringing about solution of the serpentine, there must be a close relation between the increase in volume of the rock mass and the volume of material removed in solution and redeposited as chrysotile. This probably explains why the percentage of cross-fiber chrysotile in serpentine is less than the increased volume of the rock, and why the width of chrysotile veins in partly altered rock sometimes bears a definite ratio to the thickness of the inclosing bands of massive serpentine. When the rock is homogeneous and the alteration proceeds uniformly, a definite and limited proportion of the resulting serpentine is removed in solution, for the local pressure is relieved by the transfer of this excess material.

When the formation of serpentine is accompanied by little or no increase in the volume of the rock because of the complete removal of a portion of the products, no chrysotile may be formed. It is possible that in individual cases the resistance to expansion and various other factors prevent the formation of chrysotile veins.

The difference in the specific gravities of massive serpentine and chrysotile is unquestionably due to the difference in their modes of origin. According to Dana[27] the specific gravity of massive serpentine varies from 2.50 to 2.65, while that of chrysotile is only 2.219.

While it seems probable that pressure is the controlling factor in the solution of serpentine and its redeposition as chrysotile, this is not essential to the writer's theory of the origin of cross-fiber veins. The solution of mineral matter and its redeposition in fibrous veins may result from one or more of several different causes. In the laboratory experiments referred to above, supersaturation was induced by either evaporation or cooling. The essential conditions for the growth of cross-fiber veins are: (1) the growing fibers must be in contact at their base with supersaturated solutions; and (2) that the solutions must reach the grow-

[26] G. P. Merrill: On the Serpentine of Montville, New Jersey. *Proceedings of the U. S. National Museum* (1888), **11**, 105–112.

[27] E. S. Dana: *System of Mineralogy*, Ed. 6 (1914), 269–271.

ing veins through the wall rock. So long as the veins are in contact with supersaturated solutions, growth will continue and the walls will be pushed apart regardless of resisting forces; but the greater the resisting pressure, the greater the concentration must be in order to produce supersaturation.

The force that enables growing veins to make room for themselves by pushing apart their inclosing walls is not due to the tendency of a crystalline substance to assume a regular polyhedral form, for the columnar or fibrous structure of most minerals occurring in cross-fiber veins is not a crystallization property, but is caused by the conditions of growth, as already explained. Under suitable conditions, the fibrous structure may develop in substances that crystallize in any of the systems of crystallization. In most cases it is not the normal habit, and therefore is unstable. The writer believes that the force is due chiefly to the expansion in volume which accompanies the separation of most solids from solution. When a substance separates from solution with increase in volume, the pressure developed depends on the magnitude of the forces resisting expansion, and may be many times the force required to crush the substance. It is altogether improbable that pressure alone could expel solutions occupying subcapillary pores in rocks, and, in serpentine and other rocks found inclosing cross-fiber veins, the openings are almost entirely subcapillary in size. As previously stated, the transfer of material is probably due to diffusion rather than to circulating solutions.

Structural Features of Cross-fiber Veins

Nearly all of the structural features characteristic of the cross-fiber veins found in rocks have been reproduced in the course of laboratory experiments. The diagrammatic sketches shown in Figs. 1 to 7 are all drawn to scale from veins of chrysotile or from veins of other fibrous minerals, but, so far as the phenomena that they illustrate are concerned, they might just as well have been drawn from some of the veins produced in the laboratory.

Cross-fiber veins commonly show a central parting or break in the continuity of the fibers (see Fig. 4), but in some veins the fibers extend from wall to wall without interruption. The central parting is formed whenever fibers start to grow from both walls of a fracture, and apparently this always happens when veins develop along preëxisting fractures, unless growth is limited to one side only. In laboratory experiments, growth is frequently limited to one side of a vein by the development of the fissure in such a direction as to cut the other side off from the solution furnishing material for growth. That this may also occur in the case of chrysotile veins is indicated by the occasional presence of the parting very close to one of the walls, instead of near the center. Since growth

in such veins has been very largely limited to one side, it is more than probable, in some instances, that growth may be entirely confined to one side. In some of the laboratory veins, the absence of a parting is due to simultaneous growth at both ends of the fibers—a fact proved by changing the color of the solutions. This occurs only when vein growth and the inception of fracturing are contemporaneous. The absence of partings is most common in the small lenticular chrysotile veins that frequently narrow and pinch out without intersecting other veins, thus indicating that they were not formed along preëxisting joints. The absence of partings is also more noticeable where the inclosing serpentine is practically free from mineral impurities.

Occasionally two or more partings may be observed in the same chrysotile vein, and these are sometimes symmetrically arranged with respect to the center. Such partings may result from a pause in the process of growth or from a slight displacement of the walls along their contact with the vein. A parting may frequently be observed separating two stages of growth in the fibrous or needle-like ice columns found on clayey soils after a cold night, and a similar parting in fibrous gypsum has been noted by Merrill.[28]

When cross-fiber veins are examined closely, it may be seen that the partings are never plane surfaces. The specimen of chrysotile[29] sketched in Fig. 1 shows how very irregular the partings may be in individual cases. The same phenomenon can be observed on a smaller scale under the microscope, where the fibers look as though they had been pushed past one another, causing the ends to interpenetrate along the line of parting. The inequality in the length of fibers is due to the more rapid growth of these fibers that are most favorably situated for receiving additions of new material. The stresses resulting from unequal growth are relieved by slipping along cleavage planes and prism boundaries, and this tends to accentuate the development of the fibrous structure. Additional evidence of the unequal growth of fibers in chrysotile veins is furnished by the similar displacement of bands roughly paralleling the walls and marked by a slight difference in color from the rest of the vein material.

The partings in chrysotile veins are commonly marked by the presence of granular crystals of magnetite and chromite and angular inclusions of the massive serpentine wall rock, as in Figs. 2 and 7. In size, the fragments of wall rock range from the smallest grains to masses that are larger than the veins themselves. In other words, there is every gradation between branching veins and veins with inclusions of wall rock.

[28] G. P. Merrill: On the Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings U. S. National Museum* (1894), **17**, 81.

[29] This specimen of chrysotile, from the deposits on Ash Creek near Globe, Ariz., was furnished through the courtesy of the U. S. Geological Survey.

The inclusions are also irregularly distributed through the veins without reference to partings, and occasionally a large number of fragments may be seen strung out in a broken train that may extend from the wall to a

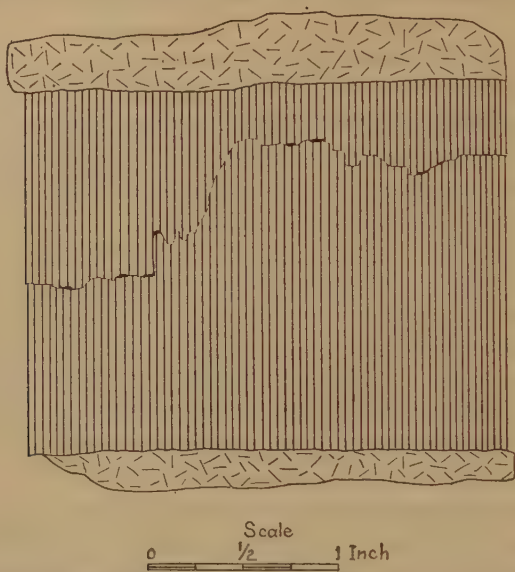

FIG. 1.—CHRYSOTILE VEIN FROM NEAR GLOBE, ARIZ. PARTING IS VERY IRREGULAR AND MARKED BY OCCASIONAL INCLUSIONS OF MASSIVE SERPENTINE WALL ROCK.

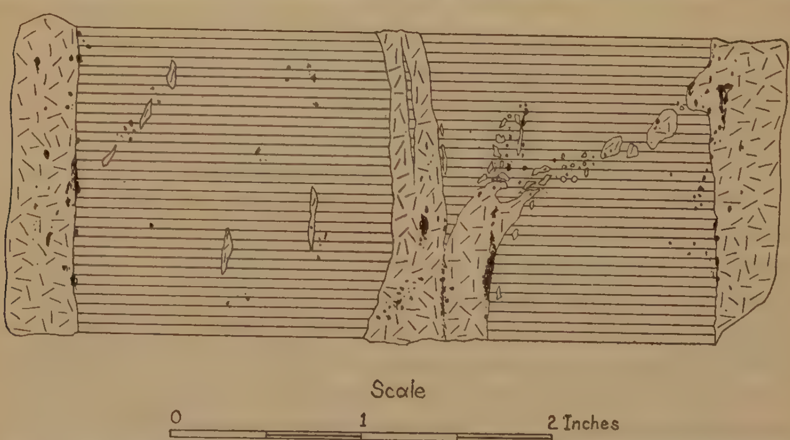

FIG. 2.—CHRYSOTILE VEIN FROM THETFORD, CANADA, CONTAINING CENTRAL INCLUSION OF MASSIVE SERPENTINE AND SMALLER INCLUSIONS OF SERPENTINE AND MAGNETITE EXTENDING IN A BROKEN TRAIN TOWARD ONE WALL. THERE IS A VEINLET OF CHRYSOTILE IN THE LARGE INCLUSION.

central parting, as in Fig. 2. Some of the larger inclusions contain veinlets of chrysotile.

Cross-fiber veins grown in the laboratory are frequently branching,

and they also contain numerous inclusions of wall material distributed through the veins in exactly the same way as in veins of chrysotile and other fibrous minerals, such as crocidolite, gypsum and calcite. The inclusions that mark a central parting represent fragments formed when rupture occurred, and their position is due to the growth of the vein on both sides as new material was added through the walls. When growth

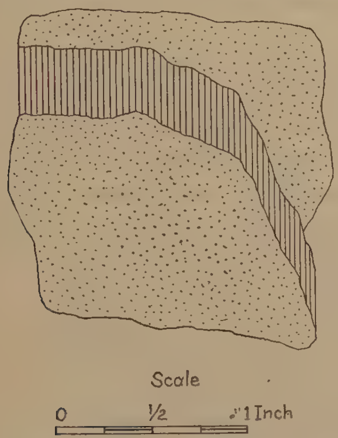

FIG. 3.—VEIN OF FIBROUS CALCITE IN LIMESTONE FROM ST. LAWRENCE COUNTY, NEW YORK.

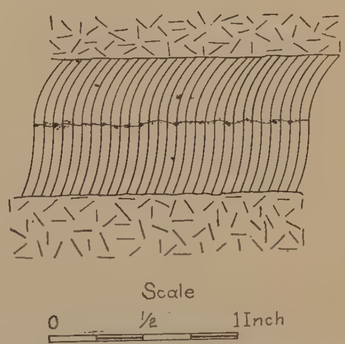

FIG. 4.—CHRYSOTILE VEIN FROM LOWELL, VERMONT, WITH CURVED FIBERS.

is more rapid on one side, the line of parting, together with the inclusions, is closer to the opposite side. Vein matter occasionally begins to crystallize out along an incipient fracture or line of weakness close to the vein, and in this way a fragment is gradually separated from the wall and included in the growing vein.

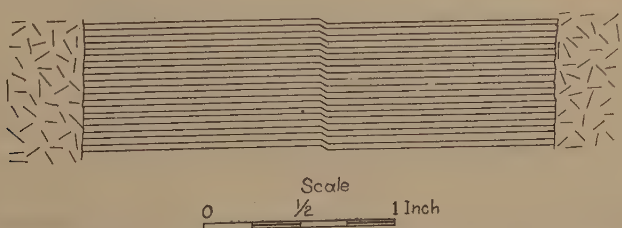

FIG. 5.—CHRYSOTILE VEIN WITH FIBERS SHOWING TWO ABRUPT BENDS.

The fibers are always parallel to one another, and, in most veins, they are approximately normal to the walls (see Fig. 2), but frequently they are more or less oblique. Laboratory experiments prove that the fibers always extend in the direction in which the walls move as they are pushed apart by vein growth. In most veins the fibers are normal to the walls because the walls are usually forced directly apart, but when the walls

have also a lateral displacement because of the simultaneous growth of adjacent non-parallel veins or other causes, the fibers grow in the direction of the resultant motion. If the course of a vein is not straight, the fibers may be normal to the walls at one place and oblique at another, as in the calcite vein from St. Lawrence County, New York, sketched in Fig. 3. In this way cross-fiber veins may grade imperceptibly into slip-fiber veins.

As long as the relative motion of the walls of a growing vein is in a straight line the fibers are straight; any change in the direction of motion is immediately recorded by the slowly lengthening fibers. If the change in the direction of relative motion is gradual and continuous, the fibers are curved, as in Fig. 4, and if the change is abrupt, it results in the development of sharp bends. In the chrysotile vein shown in Fig. 5, the initial movement of the walls had a lateral component, which soon

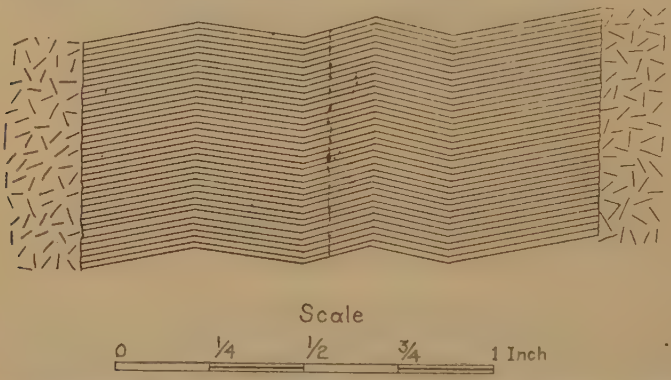

Fig. 6.—Chrysotile Vein with Fibers Showing Four Abrupt Bends and a Central Parting.

disappeared as the vein continued to grow. The structure here suggests that the formation of the vein fracture was due, at least in part, to shearing stresses. Most of the fibers in this vein extend from wall to wall without a break, and their tensile strength does not seem to be appreciably affected by the bends.

Sometimes the fibers of a vein record several changes in the relative movement of the walls, and this gives a banded appearance due to the unequal reflection of light where the fibers run in different directions. The specimen of chrysotile from Thetford, Canada, sketched to scale in Fig. 6, is a beautiful example of this kind of banding. The beginning of the vein is recorded by a parting within the central band, and there are numerous inclusions of chromite and massive serpentine distributed along the line of parting. The movement of the walls during the first stage of vein growth is recorded by the direction of the fibers in the central band, and during the enlargement of the vein there were two sudden

changes in the relative motion of the walls with the formation of two additional bands on both sides of the central band, thus making five in all.

The structural phenomena discussed in the preceding paragraphs all furnish evidence tending to confirm the writer's theory of the origin of cross-fiber veins, and they are difficult or impossible of explanation under any other hypothesis of vein formation so far advanced.

Veins of Crocidolite and Other Fibrous Amphiboles

All of the asbestiform minerals belonging to the amphibole group have been reported as occurring in cross-fiber veins, though the number of such instances is small, and, as yet, little information has been published concerning the details of individual occurrences. For this reason, it is not possible at the present time to discuss, except in a general way, the factors controlling the origin, solution and redeposition of the minerals, but the data now available indicate that these veins must also have been formed through a process of lateral secretion, and that they have made room for themselves by forcing apart the inclosing wall rock. Crocidolite is the only asbestiform mineral other than chrysotile known to occur in cross-fiber veins in commercial quantities, and only one occurrence of this mineral, so far reported, is of any importance.

Crocidolite has long been known to occur in West Griqualand, South Africa, and most specimens found in mineral collections have come from that source. The deposits are located in a range of quartzose schists called the asbestos mountains. The mineral is characterized by a dull lavender-blue color, due to the presence of considerable iron protoxide. In places it is altered by oxidation and infiltration of silica to a compact siliceous stone with a bright yellow color, and chatoyant luster, which has given it the popular name of *tiger-eye*. A brief account of this occurrence is given by Cirkel[30] in his monograph on chrysotile asbestos. He states that the crocidolite "is generally found in veins, seldom less than 2 in., and more often 4 in. and 5 in. wide, formed of closely compacted parallel fibers which run from wall to wall of the vein without break or fault. Several veins have been found, regular in extent, and the fiber always lies at right angles to the sides of the deposit. The inclosing rock is a dark brown shale."

A considerable number of specimens of crocidolite from South Africa have been examined megascopically and also under the microscope, by the present writer. All of the structural phenomena characteristic of

[30] Fritz Cirkel: Crysotile-Asbestos, Its Occurrence, Exploitation, Milling, and Uses, Ed. 2, *Canada Department of Mines, Mines Branch, Report* No. 69 (1910), 239–240. The quotations here given appear to have been taken from a paper by H. T. Olds, Notes on Blue Asbestos, read before the Institution of Mining and Metallurgy, London, *Transactions* (1899), 7, 122–123.

chrysotile veins may be found in veins of crocidolite. Specimens in the collections of the University of South Carolina show that the vein fibers are sometimes oblique instead of perpendicular to the walls, partings are frequently present and may be marked by inclusions of hematite, magnetite, quartz and fine-grained wall rock; bent and curved fibers are also common. Most of these features may likewise be observed in specimens of tiger-eye. In view of all the facts stated above, it must be concluded that, although consisting of different material, the crocidolite veins have grown in the same way as those of chrysotile.

Anthophyllite and true asbestos are rarely found in cross-fiber veins because the conditions prevailing at the time of their formation are not, as a rule, favorable for the solution and transportation of these minerals to a new place of deposition. According to Hopkins,[31] the veins of cross-fiber anthophyllite found in Georgia "are in every way similar to chrysotile, the fibers being perpendicular to the inclosing walls, and sometimes continuing from the one to the other, but more commonly jointed one or more times." This description indicates that their origin is similar to that of other cross-fiber veins.

Origin of Slip-fiber

All asbestiform minerals, with the possible exception of crocidolite, are known to occur as slip-fiber, and true asbestos seldom occurs in any other way. The origin of the slip-fiber type is a question that has aroused much less diversity of opinion than the origin of the cross-fiber type. Merrill[32] and others have advocated the efficacy of pressure and shearing in the production of a fibrous structure in minerals. The common association of slip-fiber with slickensided surfaces and other evidences of faulting have apparently convinced most investigators that shearing is the essential factor in its formation. The various ways in which unequal pressure may bring about the development of a fibrous structure have been previously discussed, but there is reason for believing that all of the so-called slip fiber has not been formed directly as a result of unequal pressure.

Cirkel believed that the slip-fiber found in the Broughton district, Quebec, had been formed from cross-fiber chrysotile as "the result of the secondary readjustment, which took place immediately after the crystallization of the fiber in veins. Both rock and veins must have been in a semi-magmatic condition during this period, and pressure may have aided this process of physical alterations of the mass in a marked degree."[33]

[31] O. B. Hopkins: *Op. cit.*, 104–105.

[32] G. P. Merrill: On the Serpentine of Montville, New Jersey, *Proceedings of the U. S. National Museum* (1888), **11**, 105, and Notes on Asbestos and Asbestiform Minerals, *Proceedings U. S. National Museum* (1895), **18**, 289.

[33] Fritz Cirkel: *Op. cit.*, 94–95.

It is possible if not probable that some slip-fiber has been formed from cross-fiber, but there is absolutely no reason for postulating that the rock was "in a semi-magmatic condition."

Dresser[34] reached a somewhat different conclusion as to the origin of the Broughton deposits. He notes that the serpentine at Broughton was probably derived from pyroxenite, while that at Thetford, where cross-fiber predominates, was derived from peridotite. After calling attention to the fact that the asbestos is limited to "the sheared and shattered" portions of the rock mass, he argues that there is some connection between the two. He further[35] suggests that when the pyroxenite was altered to serpentine, "the upper portions may have had a development of asbestos in the form of 'mass-fibre' or asbestos irregularly distributed through the rock, possibly due to a greater action of magmatic waters near the top of the sills; and that this fibrous structure weakened

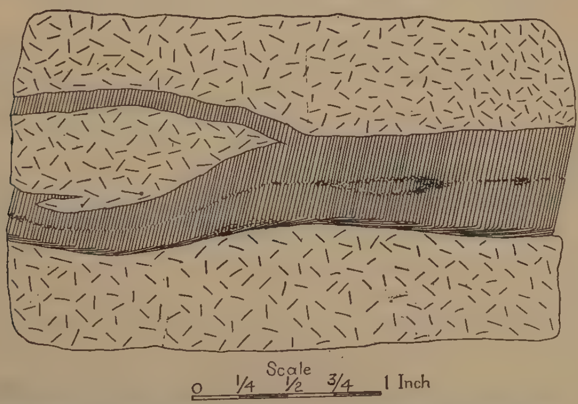

Fig. 7.—Chrysotile Vein from Lowell, Vermont, Showing Gradation of Cross-Fiber into Slip-Fiber along the Lower Wall.

the resisting power of the rock and the shear zone was thus localized." The chief objection to the latter hypothesis is, that while chrysotile is common both as cross-fiber and as slip-fiber, it is not known to occur as mass-fiber.

The investigations of the present writer indicate that much so-called slip-fiber has really been formed in the same way as cross-fiber, as is evidenced by the presence of partings, inclusions and fibers that are oblique rather than parallel to the inclosing walls. In fact, every gradation may be found between cross-fiber veins in which the fibers are nor-

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[34] J. A. Dresser: A Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey Memoir* No. 22 (1913), 69.

[35] J. A. Dresser: *Idem.*, 70.

mal to the walls, and slip-fiber veins in which the fibers are approximately parallel to the walls. In some instances the gradation may be observed in a single vein, as in the specimen[36] from near Lowell, Vt., sketched in Fig. 7. In this vein some of the fibers terminate at the abrupt bend, but others continue without a break, although they are less flexible where approximately parallel to the vein walls than where they are more nearly normal.

Origin of Mass-fiber

Asbestos of the mass-fiber type, so far as known, is always anthophyllite. Several occurrences of this kind are found in the vicinity of Sall Mountain, Georgia, which may be considered the type locality, as the deposits have been worked on a small scale for several years. More recently similar deposits have been opened up about 14 miles southeast of Kamiah, Idaho.

The deposits at Sall Mountain are roughly elliptical in shape, and the largest, which has been practically mined out, had a length of about 75 ft., a width, near the middle, of 50 ft., and apparently pinched out at a depth of 50 ft. It is estimated that considerably over 90 per cent. of the rock mass is realized as fiber.[37] The fibers are arranged in small groups or bundles, and range up to an inch in length, though averaging only about ½ in. The fibers show a strong tendency to form spherical bunches with radial structure, but because of mutual interference these bodies are, as a rule, only imperfectly developed, and therefore, in most cases, the rock consists of a mass of fibrous bundles and sheaves oriented in all directions. Occasionally, however, cross-fractures show well-formed rosettes of radiating fibers. Individual fibers sometimes appear jointed or broken. They are low in tensile strength and brittle, readily breaking into short lengths so that none of the material is of spinning grade. Hopkins states that, because of lack of flexibility, the fibers are broken so many times during the milling process "that they are exceptionally ¼ in. long, while the bulk is $\frac{1}{10}$ in. and less."[38]

According to Diller, the rock found near Kamiah, Idaho, "is very like that mined at Sall Mountain, Georgia, except that in Idaho the fibers are somewhat coarser and the radial groups larger."[39]

Hopkins, who studied and described the Georgia deposits in great

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[36] This specimen was obtained through the courtesy of the U. S. National Museum.

[37] J. S. Diller: Asbestos, *Mineral Resources*, 1907, *U. S. Geological Survey* (1908), pt. II, 717.

[38] O. B. Hopkins: Report on the Asbestos, Talc and Soapstone Deposits of Georgia, *Geological Survey of Georgia, Bulletin* No. 29 (1914), 88.

[39] J. S. Diller, Asbestos: *Mineral Resources*, 1909, *U. S. Geological Survey* (1910), pt. II, 729.

detail, concludes that the origin of the mass-fiber is due largely to the alteration of enstatite.[40]

Since this alteration involves an increase in volume, it probably takes place in the katamorphic zone. The change of enstatite to anthophyllite is paramorphic, and therefore the presence of solutions is not essential, as in the case of reactions involving the addition or subtraction of material. This probably accounts for the difference in the origin and structure of serpentine and of mass-fiber anthophyllite. The alteration of a rock mass to serpentine begins along fractures and all accessible surfaces, gradually penetrating toward the unaltered interior; while the alteration to anthophyllite begins at a large number of centers, more or less evenly distributed through the rock, and then spreads out radially from these centers. The lack of flexibility in mass-fiber anthophyllite is attributed largely to the fact that physical conditions have not controlled and limited its growth to the same extent that they have controlled the formation of the cross-fiber and slip-fiber types. The separation of the fibers is made easier by the hydration of the anthophyllite in the belt of weathering.

Discussion

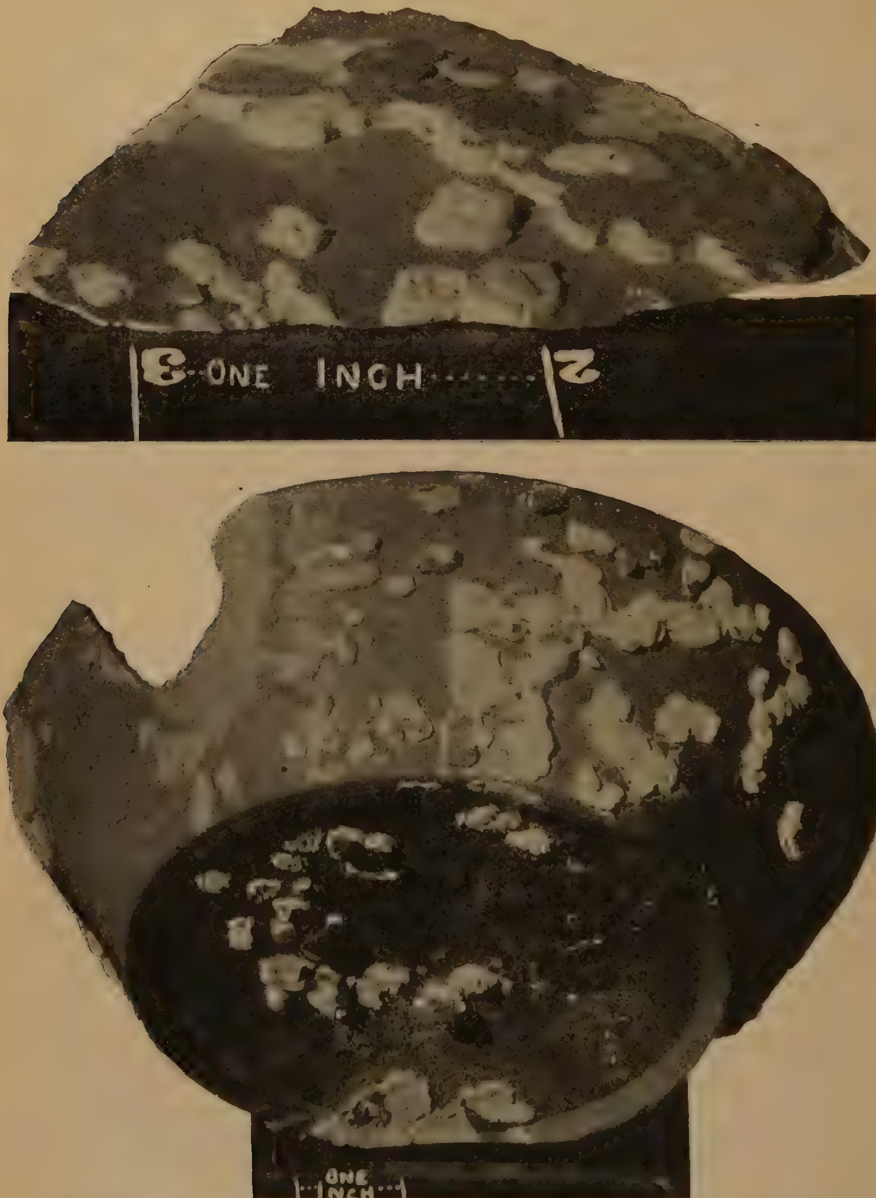
John C. Branner, Stanford University, Cal. (communication to the Secretary*).—Wideawake teachers of geology are constantly on the lookout for good illustrations of veins, especially where the processes of formation are either clearly shown or suggested. And I long ago found out that one who would learn nature's laws must despise nothing.

It thus happened that in 1910, Dr. George J. Peirce of the department of botany at Stanford University called to my attention a cracked and broken yellow earthenware dish, 8½ in. across the top and 2½ in. deep, that, to the ordinary observer, seemed to be of no further use, and quite ready for the waste can. The striking thing about this dish was that the thin glaze over much of the outside surface and over the upper margin of the inside surface had been lifted in small scales, varying in diameter from ½ cm. to 1 cm., and thrust outward and held there by some sort of white fibrous mineral. The process looked very much like one we are familiar with in the Southern States where we often see in the winter pebbles and soil thrust out from a clay bank or lifted from a clay soil and held there supported by what is popularly known as needle-ice.

Upon inquiry I learned the following facts in regard to the history of this particular dish: In the summer of 1910, it had been partly filled with a concentrated salt solution taken from one of the salterns of the

[40] O. B. Hopkins: *Op. cit.*, 104.

* Received Jan. 24, 1917.



Asbestiform Mineral Formed on the Outside of a Glazed Earthenware Dish of Porous Material. The Mineral Crystallized beneath the Glaze and Lifted It off in Scales. The Upper Photograph Shows, Enlarged, the Fragment Broken from the Lip of the Dish.

Stauffer Chemical Co. at Redwood City, Cal. A qualitative analysis of the original brine showed:

Present: iron, magnesium, sodium, potassium, chlorides, sulphates, nitrates (trace), bromides, carbonates, borates (trace), and considerable amounts of organic matter.

Absent: calcium, lithium and iodides.

The solution in the dish was a saturated one made of the unpurified commercial salt just as it is shoveled out of the salterns at Redwood. This salt was dissolved in distilled water to which a little more than ½ per cent. of agar-agar had been added. Inasmuch as more of the agar-agar had been added than was intended or desired, the solution was left standing undisturbed in the dish on a windowsill in one of the botancial laboratories at Stanford University from the time the preparation was made in July, 1910, until Nov. 20, 1910, when the contents were found to have completely evaporated and the dish to have been shattered and chipped off by the crystallization of the salt in process of evaporation, as may be seen in one of the accompanying photographs.

Furthermore, it is evident that the nature of the dish itself had much to do with what happened to it. The dish is salt glazed both inside and out, and the glaze, examined under a magnifying glass of low power, is seen to be cracked in angular mosaic-like blocks about 1 mm., in diameter. The body of the dish, where it is broken, is 5 mm. thick, is of a light cream color, and is *porous*.

The accompanying photographs give some idea of what happened. The solution inside of the dish penetrated the cracks of the inner glaze and the porosity of the body of the dish permitted the solution to pass through it, and to evaporate through and from the cracks of the outer glaze. Evaporation of the water caused the crystallization of the minerals in solution beneath the outer glaze, and the formation of the crystals lifted off the scales, as they grew at their inner ends, just as needle-ice thrusts outward the earth or stones beneath which it starts to form.

The minerals are all needle like and parallel—in other words, they are asbestiform. The length of these asbestiform minerals varies considerably; the longest noted are 1½ mm. in length. Somewhat longer crystals were broken off by the handling of the dish before it came to my attention.

It is worthy of note that chipped pieces were formed on the bottom of the dish, all over the outside of it, and on the inside of it above the level at which the fluid formerly stood. A thick scum was evidently formed over the surface of the solution shortly after it was made, and this seems to have prevented evaporation below the level of that protective covering inside of the dish.

In the photographs some of the crystals seem to have no chips of the glaze on their summits. This is due to their having been broken off in

handling. In some cases where the glazed chips do not show I have removed the crystals and have found them springing from shallow bare pits, showing that they too had started beneath the glazed chips.

The upper photograph shown in the accompanying plate is of the piece broken from the nick seen in the photograph of the dish. The cracks about this fragment were filled with the white asbestiform mineral, but it is quite possible that these were incipient cracks that have simply been enlarged by the deposition of mineral matter.

Starting from the left edge of the nick in the dish and 1 in. (25.4 mm.) below its lip, another crack runs nearly halfway round the vessel and finally ends on the lip itself. This crack is now filled with the white asbestiform mineral. I suspect that this is also an incipient crack that has been enlarged by the deposition of the mineral.

The specimen is preserved in the department of geology at Stanford University, California.

The application of the theories of Dr. Taber in regard to the origin of asbestiform mineral veins seems to explain satisfactorily all the phenomena shown in this interesting illustration.

In conclusion, I beg to express my high appreciation of the great value of the contributions of Dr. Taber to the subject of vein formations. Such a paper as his is a great credit to the American Institute of Mining Engineers, and it should remind our young men especially that the solutions of some of our greatest problems in geology lie within their easy reach if only they will put themselves in that scientific and receptive attitude of mind which enables us to sit humbly at the feet of nature.

John A. Dresser, Montreal, Canada (communication to the Secretary*).—The features of Mr. Taber's paper from which I shall dissent are questions of fact rather than of theory.

Regarding the origin of chrysotile veins, Mr. Taber points out that if they were formed by fibers growing outward from a central fracture the fibers would tend to grow unequally in length and thus give the veins irregular boundaries against the massive serpentine wall rock, which he implies is not the case. In this Mr. Taber's observations of the occurrences of Southern Quebec at least are clearly in error. Microscopic evidence shows abundantly that the characteristic boundary is finely irregular and might be described as a minutely jagged line. It is scarcely, if at all, less irregular than the line of boundary between the serpentine wall rock and the adjacent peridotite. This characteristic, of course, affords strong evidence of the growth of the vein at the expense of the wall rock.

Inclusions of massive serpentine in the central partings of chrysotile veins are not infrequent. I have never observed that they are characteristically angular, or that any occur in other parts of the vein. They

* Received Dec. 2, 1916.

might be found in "poor" veins in which the chrysotilization is far from complete.

The question of "mass fiber" chrysotile, which Mr. Taber says does not exist, may be only one of definition. The serpentine of the East Broughton mines is largely fibrous. For instance, in some of the mines 90 per cent. or more of the rock is passed through the concentrating mills. Perhaps 10 to 15 per cent. of this amount is recovered and the balance rejected because the fiber is too short. But the rejected dump material differs from that saved only in the length of fiber. In places chrysotile at the Broughton mines appears as slip fiber, and occasionally veins are found, but the great mass of the product of these mines is derived from rock which on crushing and screening separates into fibrous particles. For such material it is difficult to find a more applicable term than "mass fiber," and there seems to be no obvious reason why the term should be restricted to anthophyllite.

R. P. D. Graham, Montreal, Canada (communication to the Secretary*).—For some time past I have been interested in the mineralogy of the Black Lake-Thetford area in the Province of Quebec, and during the past summer completed a short memoir, now in the press,† on the origin of the serpentine and chrysotile which are so abundant in that locality. I was, therefore, especially interested to read Mr. Taber's paper on the genesis of asbestos and asbestiform minerals, and gladly avail myself of the invitation to discuss one feature of it.

While it is possible that Mr. Taber's interesting experiments in the laboratory production of fibrous crystals may throw some light on the mode of genesis of certain fibrous minerals, I do not consider that he is justified in drawing the conclusions he does from these experiments regarding the manner in which chrysotile veins increase in width during their formation; the conditions in the laboratory experiments and in the field are totally different in at least one very important particular.

Mr. Taber writes that he was successful in producing fibrous crystals only with substances which go into solution with decrease in volume (*i.e.*, crystallize with expansion in volume). If a supersaturated solution of such a substance enters the pores of a vessel (such as the cups used in the experiments, and there crystallizes, it would naturally tend to rupture the vessel and produce cracks; any further crystallization of solution entering such cracks or fissures, either directly or through the pores, must obviously widen them by "pushing apart the inclosing walls." I do not wish to discuss here the possible reasons for the fibrous habit of the crystals or for their transverse ateitude to the cracks; these features may be due to fresh supplies of mattrial reaching the crystals only at their extremities, they may be the result of a differential pressure

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* Received Dec. 2, 1917.

† *Economic Geology* (1917), **12**, No. 2.

exerted upon the growing crystals owing to the tendency of the cracks to open, or there may be other and more complex causes. So far as the widening of the veins is concerned, however, the important point seems to be that in Mr. Taber's experiment the inclosing walls are not in any way chemically related to the solutions which bathe them, and are not capable of reacting with them. The walls, therefore, are quite inert and they are necessarily permanent; once cracks are formed, there is only one way in which they can possibly widen, and that is by the receding of their walls from one another.

Turning now to the chrysotile veins, Mr. Taber is in agreement with the view that "the alteration of a rock to serpentine begins along the existing fractures which divide the rock mass into blocks of variable size" (p. 76) and further, that this alteration and the formation of chrysotile veins are contemporaneous processes (p. 73). It would seem to follow that at the very outset serpentinization should be most complete in the layer of rock immediately adjacent to the fracture; here also the pressure developed (due to the reaction) should be at a maximum, and consequently the serpentine formed most readily soluble. If the pressure were not relieved, the serpentine would remain in solution until it reached the necessary concentration, when it would be precipitated; on the other hand, if the pressure were relieved, the separation would take place from solutions of lower concentration. In either case, the original inclosing wall has been destroyed and the serpentinizing waters proceed to attack and destroy successive new layers or zones of rock further and further removed from the original fracture.

I agree with Mr. Taber that the determining factors responsible for the fibrous habit, parallelism and transverse attitude of the serpentine in the veins, have probably been (1) crystallization under differential pressure (due to the tendency of the fractures to open) and (2) the supply of material at one extremity only of the crystals (that furthest removed from the original fracture), and I further agree with his view that the veins have increased in width during growth by the recession of the inclosing walls. I am of opinion, however, that this growth has taken place at the expense of the walls, which, as serpentinization with its contemporaneous chrysolitization progressed, have receded because they were continually destroyed, and not because they were pushed apart by the growing fibers.

While the above refers more particularly to serpentinization along pre-existing fractures, the same argument would apply to the minor chrysotile veins, which are sometimes regarded as following subsidiary strain fractures produced through the expansion which attended the alteration. It seems reasonable to presume that the waters which were responsible for the serpentinization would find their way to such cracks directly from the main fractures rather than by the necessarily very much slower passage through pores in the massive rock.

Two further points in Mr. Taber's paper may be referred to, although they are not very material to his theory. On p. 75 he states that the pressure developed as a result of the alteration of peridotite, etc., to serpentine "cannot be explained by attributing it to a chemical reaction taking place with increase of volume, as, for example, when plaster of Paris combines with water and sets." The two reactions, however, are entirely similar, so far as the nature of the volume changes involved are concerned.

Finally, the writer has reason to believe that, at least in the Black Lake-Thetford occurrence, chrysotile and ordinary "massive" serpentine have the same specific gravity, and not the values 2.50 to 2.65 and 2.219 respectively, as usually stated in the textbooks.

George P. Merrill, Washington, D. C. (communication to the Secretary*).—That the subject of Professor Taber's paper is one of interest to me has been made apparent by his numerous quotations from my writings of several years ago. Referring to my paper of 1895, which related primarily to amphibolic forms, I will say only that the idea he now expresses, to the effect that the asbestiform structure is usually due to the accentuation of a normal prismatic habit and cleavage through the limitation of crystal growth by physical conditions, is essentially the view put forward by myself in the paper he quotes, and I have as yet seen no good reason for changing this view. What I have to say today, however, relates more particularly to the asbestiform serpentines, and this mainly for the reason that Professor Taber seems to have overlooked my paper of 1905 in the *Bulletin of the Geological Society of America.*[1] Perhaps I should state first of all in this connection that my knowledge of the field relations of this form of the mineral is limited to but a few hours each at the Thetford, Canada, and Montville, N. J., localities, and the views which I then put forward I regarded as more in the nature of suggestions than as theories or final conclusions. It is, therefore, with a feeling of some satisfaction that I have noticed even a tendency toward agreement on the part of the several workers, including Professor Taber, who have since written. My paper had in view, however, the explanation of the formation of vein cavities as well as their filling, and as it is here that our views become divergent, I shall have to ask the privilege of quoting briefly from myself, as follows:

"The writer's own opinion, founded on the facts at present available, is that the crevices are due to shrinkage such as is incidental to the change of a highly hydrated colloidal substance into a less hydrated and more solid form, and perhaps also to a loss of silica."

While this was acknowledged as not wholly satisfactory, it seemed to

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* Received Jan. 5, 1917.

[1] (1905), **16**, 131–136.

me, and still seems, as free from objections as any that have been since advanced. Cirkel, the well-known Canadian authority, for instance, in 1910 wrote in his report to the Canadian Department of Mines: "The most rational explanation, and the one that seems to gain most support, is that the formation of cracks was caused by cooling and shrinkage of the rock masses, similar to the formation of cracks caused by shrinkage of a gelatinous mass of iron carbonate in the so-called serpentine nodules of clay ironstone, as suggested by Merrill." It is also, he added, probable that the introduction of the granitic dikes so frequently met with in the serpentine masses has caused or facilitated to a great extent the formation of numerous fissures in the immediate proximity of these intrusions by rapid dehydration due to the agency of heat. The same view was accepted in part by Dr. A. P. Lowe and others. Dr. Dresser, however, as shown by the quotations of Professor Taber, was inclined to differ, and, in part at least, for the reason that he was able to point out veins at least 100 ft. in length, which he could not conceive it possible could have remained as open fissures while filling was taking place. This latter idea is one that I had not taken the trouble to elaborate, since the fact that veins starting as mere lines may increase in width during the process of filling has become long since a part of the general textbook knowledge on the subject of vein formation. I might add, however, that even a length of 100 ft. of open cavity is not impossible, since so far as has thus been determined the vein-bearing portions of the serpentinous rock do not extend to great depth. Indeed, Dresser states distinctly that in the Broughton area the serpentine containing asbestos occurs only in the upper portions of the sills. Further, no idea is given as to the width of the 100-ft. vein which he mentions. This would certainly have an important bearing upon its ability to withstand pressure and remain an open cavity.

With Professor Taber I shall have to differ very materially as to the origin of the vein cavities, inasmuch as I cannot accept his ideas as to the origin of serpentine, nor, wholly, those put forward by the Canadian geologist whom I have quoted. My own views on this part of the subject are still essentially the same as expressed in an article on the use of the terms *rock weathering*, *serpentinization* and *hydrometamorphism*, which appeared in the *London Geological Magazine* for August, 1899, and was subsequently reprinted in the *American Geologist* for the same year. I there stated the opinion that serpentinization is a deep-seated process due to waters or vapors coming from considerable depth, and possibly, in the case of igneous rocks, even constituents of the magmas at the time of their intrusion. I have yet to learn of a single instance in which the serpentinization of a rock mass has been found to be superficial or connected in any way for a certainty with meteoric waters, the action of which is almost invariably accompanied by oxidation. The views which

I have since suggested as to the filling of the vein cavities, and which Professor Taber seems to accept in part, were to the effect that the mineral-bearing solutions permeated through the wall cavities, crystallization beginning at the outer walls, and as the fibers grew at the base, pushing out into the interior, sometimes from one wall and sometimes from both. The fragmental matter often found along an irregular longitudinal line in the interior of the vein, I considered particles of matter pushed off from the wall as crystallization proceeded, in the same manner as the fibrous gypsum in limestone,[2] and as the hoar frost in the moist ground. That as the crystals continued to grow in length there was a possibility of interference, I felt was shown by the crimpings where the fibers seemed to meet from the opposite walls of the vein. That this could not be due to any lateral movement of the walls seemed to me sufficiently evident from the fact that these were mere gash veins sometimes of such slight length that an appreciable movement was simply impossible. A feature which seems to have escaped the attention of Professor Taber and others who have argued that the cavities could not have been open cavities, is this: It is impossible to conceive of so many veins within a given area beginning as mere lines and with walls being gradually forced apart by the growing crystals without considering the compressive results that would be produced on the intervening massive serpentine portions. So far as I have observed, there is no evidence of any such crushing effect as this would call for, and, while I am not wholly committed to the open-fissure idea, it would seem to me that this phase of the subject should not be overlooked.

Not to carry the discussion too far, I will simply add that Professor Taber's conclusions, as given in his paper and the one since published in the *Proceedings of the National Academy of Sciences* (December, 1916), as to the origin of cross-fiber asbestos and the effect of "a normal prismatic habit and cleavage" on the amphibolic forms are with the exceptions mentioned substantially in agreement with those I have expressed in the papers above quoted, and as yet I see no reason for changing my views.

Stephen Taber, Columbia, S. C. (communication to the Secretary*).—The fibrous salt crystals described by Dr. Branner are interesting, and I am glad that he has put this additional evidence on record. It confirms the results that I obtained in growing fibrous crystals in the laboratory. Much similar evidence of value to science in interpreting the laws of nature has doubtless been lost through failure to appreciate little things.

Mr. Dresser points out that the boundary between chrysotile veins

[2] See my paper: On Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings of the U. S. National Museum* (1894), **17**, 77–81.

* Received Mar. 25, 1917.

and massive serpentine is finely irregular when examined under the microscope. This fact I have observed in studying chrysotile veins and also, more recently, in studying veins of fibrous calcite and fibrous gypsum. For the latter veins the statement holds true when the walls are of shale as well as when they are of the same material as the veins, but in no instance have I observed any evidence of replacement. The description of the vein walls given in my paper (pp. 73 to 74) was megascopic, not microscopic. The fact that chrysotile veins have sharply defined boundaries and are easily separated from the wall rock is evidenced by the small amount of waste adhering to the "crude asbestos" which has been shipped in such large quantities from some of the Canadian mines.

Mr. Dresser has good ground for his statement that there seems to be no obvious reason why the term "mass fiber" should be restricted to anthophyllite. I have never examined the serpentine at the East Broughton mines; but in studying other serpentines under the microscope I have noted that much of the so-called "massive serpentine" is really finely fibrous (see pp. 63 to 64). The texture of such rock is, however, altogether different from that of mass-fiber anthophyllite, and I believe that the fibers have developed in a different way (see pp. 70 and 87).

Mr. Graham raises certain objections on the ground that the laboratory production of fibrous crystals was limited to substances which go into solution with decrease in volume. Since my paper was written, however, I have succeeded in obtaining fibrous crystals of ammonium nitrate, a salt that dissolves with expansion in volume.

Recent studies[3] of fibrous calcite and gypsum veins indicate that the composition of the walls, whether similar to that of the veins or different, is of no importance, and that for these veins the fineness of the fibers is determined by the spacing of the pore spaces in the walls. There is no evidence of replacement or recrystallization along the walls, and where one wall contains angles or other irregularities, there are corresponding irregularities in the opposite wall, such that the two surfaces would fit closely together if placed in contact.

Mr. Graham is of the opinion that vein growth "has taken place at the expense of the walls," which "have receded because they were continually destroyed, and not because they were pushed apart by the growing fibers;" but he does not explain why serpentine should replace serpentine of the same composition and of "the same specific gravity." The lower figures given by textbooks for the specific gravity of chrysotile

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[3] Stephen Taber: The Origin of Veinlets in the Silurian and Devonian Strata of Central New York. This paper will appear in an early number of the *Journal of Geology*.

as compared with massive serpentine are possibly due to the difficulty of making accurate determinations in the case of such finely fibrous material.

Dr. Merrill's paper of 1905 in the *Bulletin of the Geological Society of America*[4] was not overlooked, since at my request he had kindly furnished me with a reprint. I regret that direct reference was not made to this paper, as was done with his other publications. This omission, as previously explained (pp. 71 to 72), was owing to the fact that the many theories advanced by geologists to explain the origin of cross-fiber veins had been reviewed recently in three monographs (see footnote 17, p. 72). Dr. Merrill's theory is outlined in each of these publications and was included in my classification of theories.

Dr. Merrill is still of the opinion that the veins were formed in open crevices "due to shrinkage such as is incidental to the change of a highly hydrated, colloidal substance into a less hydrated and more solid form, and perhaps also to the loss of silica, as suggested by Prof. Kemp."[5] No evidence in support of this opinion is given in Dr. Merrill's paper or in his discussion of my paper. He compares the hypothetical cracks in serpentine with "the shrinkage cracks which appear in clay on drying, or, better yet, those which result from the shrinkage of a gelatinous mass of iron carbonate, as in the so-called septarian nodules of clay-iron stone;"[6] but he cites no evidence tending to prove that the latter were formed in the manner postulated. If the vein spaces in septaria result from evaporation of water and the shrinking of a gelatinous mass, the cracks would presumably begin at the surface and diminish in size as they approached the center. Now the reverse of this is true; the veins are largest near the center and seldom reach the surface (see Plate 34 accompanying Dr. Merrill's paper). This suggests that the growth of the veins may have accompanied the growth of the concretions as a whole. The view that such veins were not deposited in open cavities is supported by the fact that they sometimes contain detached angular fragments of the wall material.

Concerning the origin of the serpentine, I purposely made no assumption as to whether the source of the water was magmatic or meteoric, for this was not essential to the development of my theory. The equation (p. 75) was introduced merely as an illustration of the fact that the formation of serpentine might be accompanied by a net decrease in volume if the volume of the water is taken into consideration. If serpentinization is a deep-seated process, as advocated by Dr. Merrill, it is all the more difficult to explain how his cavities could remain open.

Heddle seems to have first suggested the possibility that the fibrous

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[4] G. P. MERRILL: On the Origin of Veins in Asbestiform Serpentine. *Bulletin of the Geological Society of America* (1905), **16**, 131–136.

[5] *Idem.*, 135–136.

[6] *Idem.*, 136.

structure in chrysotile might be a result of its growth outward from the interstices between mineral grains (p. 66).

The increase in the volume of a rock mass when chrysotile veins are formed is due, according to my theory, to the introduction of water and formation of serpentine rather than to the growth of the veins; and the resulting compressive effects have been generally recognized by geologists.

It is not necessary to reply here to the other points raised by Dr. Merrill, since those who are interested may draw their own conclusions after reading the original papers.

The theories advocated in my paper were supported by much evidence, both observational and experimental; they furnish the only plausible interpretation of the established facts; and, therefore, I believe that my conclusions must stand until new facts are adduced tending to disprove them.

The nature of the force that enables growing veins to make room for themselves by pushing apart their inclosing walls was not discussed in detail as I had treated that phase of the subject in a previous publication. Since that time, however, I have obtained additional experimental data, and my present views on the subject are given in a paper on "Pressure Phenomena Accompanying the Growth of Crystals," which will appear in an early number of the *Proceedings of the National Academy of Sciences.*

The Conservation of Phosphate Rock in the United States*

BY W. C. PHALEN,† PH. D., WASHINGTON, D. C.

(New York Meeting, February, 1917)

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* Published with the permission of the Director of the U. S. Geological Survey.

† Mineral Technologist, U. S. Bureau of Mines.

INTRODUCTION

Nobody will dispute the fact that the conservation in every legitimate manner of our valuable high-grade phosphate-rock deposits is a present-day problem of importance.

The table and curve, given herewith, show that during the past 10 years the exportation of high-grade rock has averaged very close to half the total output of the entire country. The bulk of this exportation is from Florida, for obvious reasons. It is plain that the deposits in this State, more particularly, are being wastefully depleted under a system of selecting the cream of the product for exportation to Europe, leaving the comparatively low-grade rock, running from 65 to 70 per cent., and under, in bone phosphate of lime, for our own fertilizer manufacturers to work up when all the best rock is gone.

In this paper the writer has endeavored to make a point of describing in some detail the important methods of production and conservation in Tennessee. These ought to prove of educational value to our American agriculturists and fertilizer manufacturers, and should result in a demand for the highest-grade rock, both for direct application to the soil and for use in making acid phosphate. It is certainly evident that the European

manufacturer is alive to the situation and is demanding the highest-grade rock from the Pacific Islands and Florida. The cost of transporting low-grade rock in this country, and the acid phosphate resulting from it, is another factor which should appeal to the self-interest, if to no higher motive, of the American producer, and user as well. This factor, as well as the important one of keeping our high-grade rock at home, is the fundamental reason for a change in our policy with reference to our high-grade phosphate rock.

PRODUCTION AND EXPORTATION OF PHOSPHATE ROCK

Since the beginning of phosphate-rock mining in the United States, there has been a total output of 48,457,906 tons, more than half of which has been produced in the past 10 years. During this 10-year period there has been an exportation of nearly 11,000,000 tons, or about 43 per cent. of the marketed production in the same period.[1] The exported material does not represent average grades, but the highest-grade rock, running 77 per cent. and more in phosphate of lime (bone phosphate) and 3 per cent. and less in iron and alumina.

Production and Exports of Phosphate Rock with Ratio between Them, 1905–1914

Year	Production, Long Tons	Exports, Long Tons	Percentage
1905	1,947,190	934,940	48.0
1906	2,080,957	904,214	43.4
1907	2,265,343	1,018,212	45.0
1908	2,386,138	1,188,411	49.8
1909	2,338,264	1,020,556	43.6
1910	2,654,988	1,083,037	40.8
1911	3,053,279	1,246,577	40.8
1912	2,973,332	1,206,520	40.6
1913	3,111,221	1,366,508	44.0
1914	2,734,043	964,114	35.0
Total...........	25,544,755	10,933,089	43.0

METHODS OF CONSERVATION

Introductory Note

There appear to be many differences of opinion among soil chemists and agriculturists as to the form in which phosphorus shall be applied

[1] Figures for 1915 are not included in these computations, for the reason that the phosphate-rock industry was in an abnormal condition during that year and also during the latter part of 1914. The production of phosphate rock during 1915 was 1,835,667 long tons, and the exports 253,549 tons, or 13.8 per cent. of the production

to the soil. One group argues for the use of superphosphate, especially where quick returns are desired, and another for the use of ground rock or

FIG. 1.—MAP SHOWING THE LOCATION OF THE PHOSPHATE-ROCK DEPOSITS OF THE UNITED STATES.

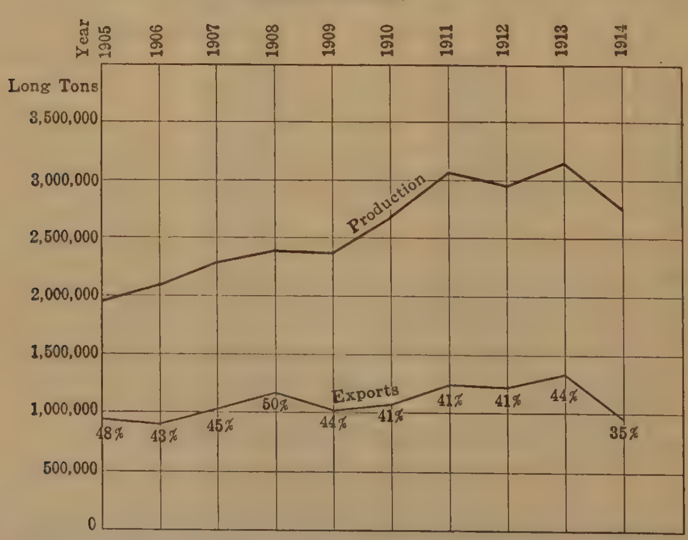

FIG. 2.—CURVE SHOWING PRODUCTION AND EXPORTS OF PHOSPHATE ROCK, 1905–1914. FIGURES ON EXPORT CURVE INDICATE PERCENTAGE OF PRODUCTION LEAVING THE COUNTRY.

"floats," especially where permanent results are the object. Without considering the merits of either side of this question, it is certain that the conservation of our phosphate resources must be of interest to both.

The conservation of phosphate rock will be considered in the following pages, under the heads of (1) mechanical, and (2) chemical methods of conservation, and (3) the use of substitutes, either natural or manufactured, as a source of phosphate of lime or phosphorus.

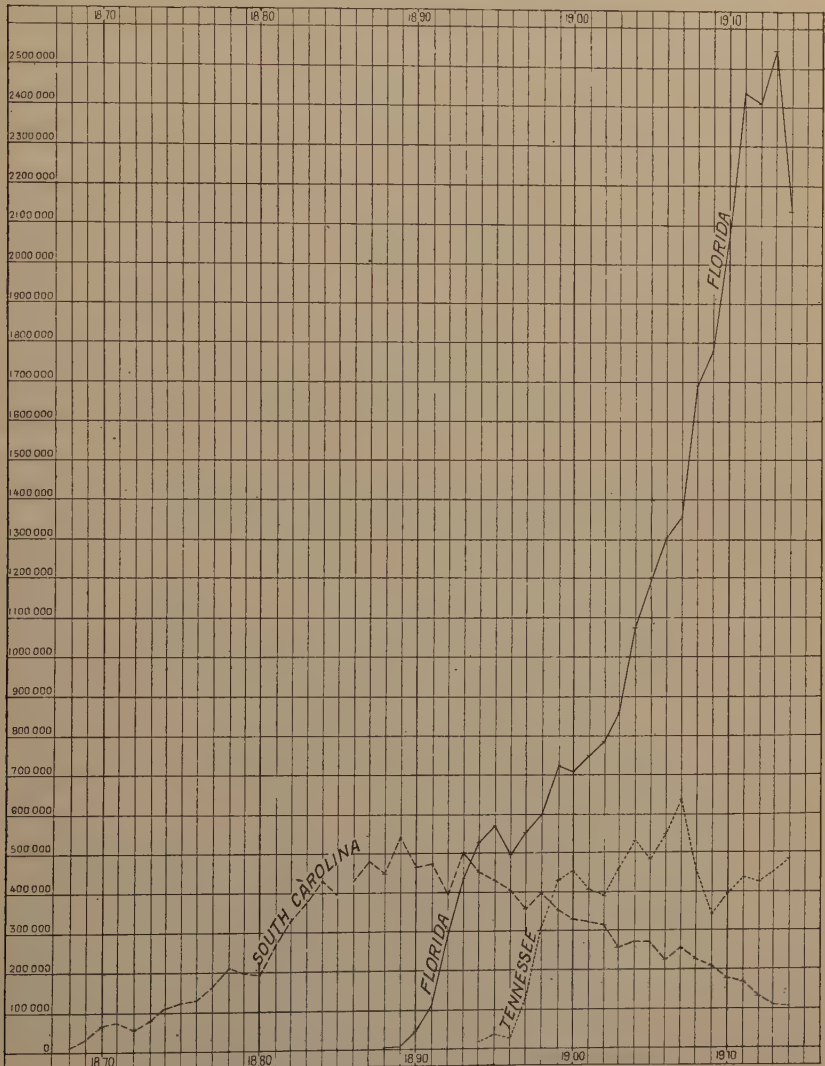

FIG. 3.—DIAGRAM SHOWING MARKETED PRODUCTION OF PHOSPHATE ROCK IN SOUTH CAROLINA, FLORIDA AND TENNESSEE IN LONG TONS FOR YEARS INDICATED AT TOP AND BOTTOM.

MECHANICAL METHODS OF CONSERVATION

It is well known that in the early days of phosphate mining in the more important phosphate fields of the United States, there was a large

waste of good material. In many places this waste is still going on. In certain places the material once thrown aside is in such condition that it may be reworked and is actually being reworked. In other places, it is lost beyond all hope of recovery in so far as can be told at the present time. The devising of methods to prevent or reduce such losses and yet maintain the grades set by commercial standards is one of the problems that has faced and is now facing the phosphate-rock miner; and in the following pages will be given descriptions of the methods by which the problem is being met in Tennessee, which may serve as a type of what is going on in the other important phosphate-producing States.

In speaking of the early wasteful methods employed in Tennessee, W. H. Waggaman of the Bureau of Soils[2] says: "For years the richest of brown rock in the Mount Pleasant region was worked by hand, and only when these deposits were considered to be nearly exhausted did the operators seem to realize the crudity, wastefulness, and inefficiency of the methods they were using. Even now a few small firms and farmers are employing the pioneer method of shaking out all rock not held by the tines of a potato fork and drying the larger pieces in the sun or on ricks of wood."

The same writer, in speaking of the waste connected with the mining of hard rock in the Florida phosphate field,[3] says: "In order to meet the present demand for a high-grade product, a vast amount of phosphatic material is thrown aside. The marketed product is probably not more than 15 per cent. of the total material mined. The remainder, consisting of sand, clay, and the phosphates of lime, iron, and aluminum, is washed out upon a waste pile. This discarded material varies in its content of phosphoric acid, but seldom if ever contains less than 10 per cent. The actual amount of phosphoric acid discarded, therefore, is almost twice as great as the quantity saved. This enormous amount of low-grade phosphate will no doubt eventually be used."

In speaking of the waste connected with the mining of pebble phosphate in Florida, the same writer makes the following observations:[4]

"The percentage of phosphoric acid washed away in preparing pebble phosphate for the market is fully as great as that wasted in mining hard-rock phosphate. In the vicinity of the large phosphate washers many acres are covered to a considerable depth with this detritus. The author made no determination of the exact composition of this finely divided material, but it resembles the wash from the hard-rock mines. . . . An analysis, however, was made of the material having a diameter greater than 2½ in. discarded before the material containing the phosphate goes through the washing process. The composition of a sample thrown

[2] *U. S. Bureau of Soils, Bulletin* No. 81 (1912), 9.

[3] *U. S. Bureau of Soils, Bulletin* No. 76 (1911), 14.

[4] *Loc. cit.*, 21–22.

from the picking table at one of the plants was also determined." These two analyses are given in Table 1.

TABLE 1.—*Chemical Analyses of Discarded Material Connected with Preparation of Florida Pebble Phosphate*

Description	Analysis				
	SiO_2, Per Cent.	Al_2O_3, Per Cent.	Fe_2O_3, Per Cent.	P_2O_5, Per Cent.	$Ca_3(PO_4)_2$, Per Cent.
Clay balls, 2½ in. in diameter, containing phosphate pebbles from discard.........	44.10	5.24	1.78	17.60	38.54
Material from picking table..	50.58	5.18	12.96	7.20	17.68

In the Mount Pleasant phosphate field at the present time, and probably in other parts of the Tennessee brown-rock phosphate areas, changes are being made that will result in leaving little or no wasted phosphate rock in the ground. Some phosphate is going into the waste ponds, but, without doubt, the time will come when all this material will be reworked, and even now some companies are working or are planning to work these old tailings. The modern mining and milling methods of the last decade are revolutionizing the industry and incidentally conserving this valuable fertilizer material. They are in striking contrast with the crude and wasteful methods formerly employed in the brown-rock field. Though the large operators are using up-to-date methods, even now some of the small operators are employing the old-fashioned hand methods which in the past resulted in the loss of much valuable rock.

The object, of course, in preparing phosphate rock for market is to remove as much of the clay, chert, and limestone as possible from it. Theoretically, it is possible to remove all these impurities, but this is not practicable, especially in the case of the clay. There is no sharp division between the finest phosphate sand and the clay, and it would obviously be wasteful to carry the process of obtaining the fine sand up to or beyond the point where the cost would offset the value of phosphate obtained. This is one of the practical considerations connected with the modern conservation of phosphate rock which perhaps has not always been given just and deserved consideration.

The point beyond which it is not practicable to carry the preparatory treatment is not fixed, and standards vary from time to time, and probably at a given time, among individuals and corporations. Thus, in the phosphate-mining industry as practiced in Tennessee in the early 90's, rock was discarded which has a high value today, and the former apparent lapses from the highest standards have in the course of time proven to be not lapses at all, but simply conditions imposed by the trade and the

times. In other words, the phosphate once discarded is now being utilized. The open-cut or surface method of mining brown rock, as practiced in the Tennessee field, with which the writer is more especially familiar, is peculiar in this respect, and the generalizations made do not cover any other classes of mining, and certainly will not apply to underground mining in general.

Grades of Brown Phosphate Rock

Most of the rock from the Mount Pleasant field is shipped in three grades, namely, those containing 72, 75, and 78 per cent. of calcium phosphate. Five per cent. of iron oxide and alumina is the maximum allowed, and this is usually referred to as "I and A" in the trade and in commercial analyses. At one time only 78 per cent. rock was shipped from the Mount Pleasant field, and rock of this grade is still known as export rock. The guaranteed content in phosphate of lime, "bone phosphate," or BPL, as it is commonly referred to in the trade, next fell to 75 per cent., and at the present time many of the companies are finding it difficult to ship this grade exclusively, and the life of the 75 per cent. rock is limited. Every per cent. of iron oxide and alumina less than the 5 per cent. limit is regarded as equivalent to an additional 2 per cent. of calcium phosphate, for it is considered that in the subsequent treatment of the phosphate in the manufacture of fertilizers the harmful effect of 1 per cent. of iron oxide and alumina offsets the good effect of 2 per cent. of calcium phosphate. If there is more than 5 per cent. of iron oxide and alumina, the superphosphate becomes gummy and farmers find it difficult to drill it into the land.

Preparation of Phosphate Rock for Market, with Phases of Conservation Involved

There are many stages to be considered under the heading of preparation of phosphate rock for market, but they may all be subdivided into three major operations as follows: (1) Removal of overburden, (2) mining, and (3) milling, in which is included drying. The present methods of utilizing and thus protecting from loss, or conserving, the brown phosphate-rock supplies, especially in the Mount Pleasant field, naturally are included under the above headings, and, therefore, will be described in connection with them so far as this can be done.

Removal of Overburden

The overburden of the brown rock in the Mount Pleasant field varies from 1 ft. up. Usually it is less than 20 ft., but a thickness of 30 ft. is known, although that is excessive in places where mining is now in prog-

ress. The methods of removal of overburden are diverse. Under exceptional conditions, the old-time crude and expensive hand methods have to be resorted to, but in most places, and especially where virgin ground is being opened, operations are conducted in the most up-to-date fashion. Where the overburden is not very thick or hard, it may simply be plowed up and removed with scrapers, or it may be loosened with dynamite and then removed with scrapers. A favorite method of getting rid of the overburden, used especially in ground that is being reworked, is to first "hog" or undercut it, pry it off with bars, and then scrape or carry it away.

The drag-line excavator and the steam shovel are types of up-to-date machinery used in removing overburden in this field. The hydraulic method is also used. Both the overburden and the rock itself are removed by this last-named method, which is simplicity itself in action and which requires a minimum of labor in operation, usually one man to handle the hydraulic gun and two to keep the sluices clear. As practiced at the plant of the Blue Grass Phosphate Co., in the southern part of the Mount Pleasant field, the rock is mined out in small areas and the overburden from one area is washed into the mined-out cavity next to it. The resultant topography is level. The land is thus conserved for farming purposes for future generations and, indeed, greatly improved, for the phosphate sand and rock, formerly below the subsoil, is thoroughly incorporated in the soil and the fertilizing value of the phosphate thus rendered available in time.

Methods of Mining

Much of the mining in the Mount Pleasant field has to be done by hand on account of the method of occurrence of the brown rock. The steam shovel has not proved successful because it cannot discriminate as to grade of rock mined, with the result that much clay and flint get into the product, and have to be subsequently removed. The cantilever adjunct to mining, which is employed at the plant of the Hoover and Mason Phosphate Co., is unique, there being only one in this field. The hydraulic method of mining is used at two plants and has many advantages, as pointed out under the preceding topic. These mechanical methods of mining and removing overburden, which have cheapened operating costs, have played the major part in conserving Tennessee brown rock.

Reworking Deposits.—The brown phosphate rock occurs in blanket form. From the base of this blanket, so-called deep "cutters" project like a network of roots, into the underlying limestone, the irregular projections of the latter, upward and into the phosphate rock, being known as "horses." In the early days of mining, all the phosphate rock occurring between the limestone "horses" was left, owing to the difficulty in mining it. Moreover, nearly everything that went through the tines of

a phosphate fork or a 2-in. screen was discarded. The latter material, for this reason, has come to be known as "screenings" or "throwbacks," and at certain plants this is now being worked. The screenings or throwbacks can be easily distinguished from the normally occurring plate or lump rock by its mixed-up or heterogeneous appearance, the lump rock being scattered irregularly throughout the mass. From the fact that the throwbacks represent material that has been worked over and from which the larger lump rock has been removed, it follows that the proportion of muck, sand, and clay is larger than in unworked territory.

On the Ruhm Phosphate Co.'s property, in the northeastern part of the Mount Pleasant field, the worked-over phosphate deposits are located on one of the terraces that surround the town.[5] This location at the top of a hill is such that the ore can be handled easily by gravity, and the hydraulic method of mining is therefore employed. The rock appears practically at the surface in quantity, and in addition to the muck left from early operations it contains much small lump rock, discarded in the days of early mining. The limestone horses often get in the way and have to be blasted out, but this is not difficult, owing to the loose or platy character of the phosphatic limestone associated with the brown-rock deposits. In addition to the hydraulic method, which can be employed only in certain favorable locations, hand mining is also employed. Where hand mining is practiced, the ore is usually screened on the spot where the miner is at work. The fine material passes through the screen and is saved and washed; and the coarse rock is hauled away and dried by burning on ricks of wood in the open, thus saving rehandling in the mill. The lump rock, as mined, usually contains from 20 to 21 per cent. of moisture, and drying it in this way reduces the moisture to 1 per cent. or less.

At the Century plant of the Federal Chemical Co., on the Williamsport Pike, northwest of Columbia, screenings are now being worked on an extensive scale.

The accompanying analyses show the content in phosphoric acid and phosphate of lime, of material left in earlier mining operations but now being worked at the plants mentioned above. It is certain that these analyses do not exactly represent the standard in phosphate content of a great deal of the material which is being reworked in this field. It is doubtful whether any hand sample can adequately represent the normal content in calcium phosphate of this material, owing to the difficulty of properly apportioning the lump rock and the muck. It is more than likely, however, that very low-grade material can and will be profitably worked, provided some of the cheaper methods of operating can be brought to bear upon it. There is also given in Table 2, an analysis of a sample of phosphate muck. This material is found in larger proportion

[5] The field observations were made in the Fall of 1914.

in the throwbacks than in the normally occurring rock, for the reason that in the original mining operations only the lump rock was saved. The writer has been told that the recovery of the phosphate in the muck handled is about 50 per cent. on the wet basis and about 40 per cent. on the dry, assuming that there is from 20 to 21 per cent. of moisture present. Unless the muck runs well up in calcium phosphate, it hardly pays to handle it.

Table 2.—*Analyses of Brown and Muck Phosphate from Reworked Deposits, Maury Co., Tennessee*

(W. C. Wheeler, Analyst)

	P_2O_5, Per Cent.	$Ca_3(PO_4)_2$, Per Cent.
No. 2	21.16	46.24
No. 27	31.88	69.66
No. 37	20.13	43.98
No. 38	27.31	59.69

No. 2: Phosphate muck from 2-ft. layer; Hoover & Mason Phosphate Co.

No. 27: Throwbacks or material reworked 5 ft. thick; Federal Chemical Co., Century plant.

No. 37: Material reworked. Average thickness, 4 ft. where sample was collected; Ruhm Phosphate Mining Co.

No. 38: Material reworked. Average thickness, 4 ft. where sample was collected; Ruhm Phosphate Mining Co.

Working "Cutters."—The phosphate rock in the "cutters," or deep places between the limestone horses, was left unmined in the early days of mining, as has been pointed out. The development of the cutters, which probably took place along original joint planes, varies greatly within the restricted Mount Pleasant field. In some places, notably south of Mount Pleasant, on the Hoover and Mason property, they are of large size. Some were observed 30 to 35 ft. wide and as much as 20 to 25 ft. deep, averaging probably 18 to 20 ft. They vary greatly in length. In these abnormally wide and deep cutters, it is not uncommon to have small limestone horses. A short distance away from the Hoover and Mason Co.'s property, some of the cutters are so narrow that the phosphate rock in them can be removed only with difficulty.

Hand methods of mining have to be employed almost exclusively to remove the phosphate rock from these cutters, owing to the peculiar method of its occurrence (Fig. 4). Hydraulic methods are also employed, as is the case in the Hickman County field where the depth of the cutters is also great. Owing to the depth of the cutters the work has to be done in benches of convenient height for the miners. The ore is picked out and shoveled from bench to bench, and finally into wagons in which it is hauled

to the mills. Mining the deep cutters is usually carried on in fair weather, or when the roads are good. In working over virgin ground at the present

Fig. 4.—Method of Removing Phosphate Rock From a Deep "Cutter." Hoover and Mason Phosphate Co., Mt. Pleasant, Tenn.

Fig. 5.—Old Phosphate Workings Showing Phosphatic Limestone, at the Plant of the Blue Grass Phosphate Co., South of Mt. Pleasant, Tenn.

time, the rock in the cutters is readily and cheaply obtained by the cantilever method in use at the Hoover and Mason plant.

Territory formerly worked over is again being worked at the Arrow mine of the Charleston, S. C., Mining and Manufacturing Co. Most of this work is in rather shallow cutters. The material is picked out and screened either on the tines of a phosphate fork or on a small movable 1-in. mesh screen. The coarse rock is dried or burned on ricks of wood; the muck is washed at the company's mill. The old cutters containing phosphate rock are located by hand prospecting with a long, sharp, steel rod.

Fig. 6.—Outcropping Limestone Horses and Associated Phosphate Rock, Tennessee-Arkansas Phosphate Co., Scotts Mill, Maury County, Tenn.

Washing and Drying

The washing processes whereby the mined rock is freed from clay, chert, and limestone are elaborate, and the mills in which the work is done are, for the most part, large and modern. These modern washing plants, which have done so much to make the mining of low-grade rock in this field profitable, and which, therefore, are playing such an important rôle in the conservation of phosphate rock in Tennessee, have practically all been installed during the last decade. The principles of the washing processes are the same in the different plants, but the details of manipulation differ. The phosphate rock, as mined, is brought to the washer either in wagons or by tram. Where hydraulic mining is practiced, it goes to the plant through a flume. The material mixed with water is delivered into a hopper at the top of the mill and the subsequent opera-

tions, for the most part, are conducted by gravity. From the hopper the rock passes through a toothed revolving crusher and then into log washers. From the washers, it passes to a cylindrical or conical screen with circular perforations. The coarse, or lump rock, which fails to pass through the screen, passes on to a picking belt where limestone and chert fragments and clay balls are removed. The material then goes to the wet-storage sheds or piles to be dried later. The fine material may go through a settler or clarifier provided with riffles, or through several settling tanks in succession, in which the sand settles out. The clay and sand not caught in the process go to the waste ponds. The above descriptions briefly outline the fundamentals of the washing process as carried on at most of the plants, but, of course, as has been mentioned, details are widely divergent.

The clay and the phosphate sand which pass to the waste ponds are of great interest in the problem of conservation. When the material reaches the waste pond, the coarse sand settles out first, and, naturally, nearest the end of the waste pipe or flume. This material is the highest in calcium phosphate. It is planned to work material of this character at one of the plants near Mount Pleasant, and already at another the old tailing dumps are being worked. At the latter plant much attention has been paid to the process of separating the clay and phosphate sand. There is a washer at this particular plant which differs from any other in the field, and is most thorough in its action. The clay resulting from the action of this washer was observed in the waste pond. It had been in suspension for a long period of time, and material taken and rubbed between the fingers appeared almost of impalpable fineness. Some of the phosphate sand from this washing process is so fine in texture that it sifts through the meshes of the sacks in which it is shipped.

The analyses of material from ponds abandoned years ago, but which are either being worked or which it is planned to work, are given in Table 3. It has been suggested that the material in these waste ponds might be used in its present form on Tennessee farms, but this has been found impracticable, as it will not bear the cost of transportation. The high phosphate content in certain of the samples collected is noteworthy.

Drying is accomplished in two very different ways, which are representative of the old and new methods employed in the Tennessee brown-phosphate field. At nearly all the large plants, modern rotating cylindrical driers, similar to rotary cement kilns, are in use, but the rock is fed both at the hot and cold ends. It would seem that the latter method would be the more efficient. There is generally some special cause when the old-fashioned method of drying on wood ricks is employed, and where it is in use it usually saves extra handling or haulage. Drying generally reduces the moisture present from 20 to 21 per cent. to 1 or 2 per cent.

TABLE 3.—*Analyses of Waste Material from Phosphate Washers in the Mount Pleasant, Tenn., Phosphate Field**

(W. C. Wheeler, Analyst)

	P_2O_5, Per Cent.	$Ca_3(PO_4)_2$, Per Cent.
No. 13	31.79	69.47
No. 20	27.20	59.44
No. 25	29.91	65.34
No. 32	20.38	44.53
No. 36	14.36	31.37
No. 42	30.63	66.92
No. 47	20.71	45.26
No. 50	28.59	62.47
No. 54	26.50	57.90

No. 13: Sample of material from waste pond; International Agricultural Corporation, Frierson plant.

No. 20: Sample of material formerly discarded but now being reworked; Federal Chemical Co., Tennessee plant.

No. 25: Sample of material from waste pond; Tenn. Ark. Mining Co..

No. 32: Sample of material from waste pond, near outlet of waste pipe. It is planned to rework this; Federal Chemical Co., Century plant.

No. 36: Sample of material from waste pond; Ruhm Phosphate Mining Co.

No. 42: Sample of material from waste pond; International Agricultural Corporation, Jackson plant.

No. 47: Sample of material from waste pond; Blue Grass Phosphate Co.

No. 50: Sample of material from waste pond; Hoover and Mason Phosphate Co.

No. 54: Sample of material from waste pond; Charleston, S. C., Mining and Manufacturing Co.

Conservation of Fines

In drying phosphate rock, much material in finely divided form has been lost by being carried out through the flue, owing to the powerful drafts employed, especially in the modern types of driers. At many of the plants, steps have been taken to save this material. This is accomplished by means of bends in the flue or by hoods or baffles. The following is an analysis of fine material caught in the chimney at the Century plant of the Federal Chemical Co. (W. C. Wheeler, Analyst):

No. 35..........P_2O_5, 30.29 per cent.; $Ca_3(PO_4)_2$, 66.19 per cent.

Chemical Methods of Conserving Phosphate Rock[6]

Their Application in the West

There is associated with all large important phosphate-rock deposits, considerable rock which is not up to the present commercial requirements

* Some of this material is now being worked and some of it will soon be worked.

[6] J. A. Barr: Tennessee Phosphate Practice, *Trans.* (1915), **70**, 917–933.

W. H. Waggaman: *U. S. Bureau of Soils, Bulletin* No. 76 (1911), 15 and 16; also *U. S. Department of Agriculture, Bulletin* No. 144 (1914), 25 and 26.

in content of calcium phosphate. There is also being produced, in connection with the preparation of commercial phosphate rock for market, a great deal of low-grade material. To bring these classes of material up to commercial grade, that is, to a grade containing 70 per cent. or more calcium phosphate, various chemical methods have been used. The time will undoubtedly come when these chemical methods will find much more extended application than at present, and when this time arrives it will result in conserving a great deal of phosphate rock now consigned to the waste ponds and dumps. Such methods are of more than ordinary interest in connection with the Western field, owing to the long distance that phosphate rock now has to be transported before reaching a market. The immense quantities of sulphuric acid potentially available in the immediate vicinity of the Western phosphate field, and which should become available in increasing quantity as time goes on, is another important element in the situation. Indeed, the chemical method of concentrating phosphate, and thus enabling it to be transported long distances, may well be worked out in connection with the high-grade rock that this field is able to produce, and it may also be the means of conserving the enormous amount of low-grade phosphate rock in the Florida, Tennessee, South Carolina, and other Eastern fields.

Chemistry of Process

Phosphate rock is marketed now as such, and in the form of acid phosphate, including in the latter term ordinary super and double-acid phosphate, the latter containing two to three times as much soluble phosphoric acid as ordinary superphosphate.

Before the discovery of the extensive high-grade deposits of phosphate rock in this country and abroad, the manufacture of the concentrated grades of soluble phosphate was in fairly common practice. The large supplies of high-grade phosphate rock have rendered this unnecessary, though in France, Germany, and possibly other European countries, and in South Carolina, this practice is reported to be still in use.

The basic reaction involved in the preparation of soluble acid phosphate takes place when ordinary rock phosphate $Ca_3(PO_4)_2$ is treated with sulphuric acid. In simple form, the reaction that takes place may be represented thus:

(1) $Ca_3(PO_4)_2 + 3H_2SO_4 = 2H_3PO_4 + 3CaSO_4.$
(2) $4H_3PO_4 + Ca_3(PO_4)_2 = 3CaH_4(PO_4)_2$

Reduced to one equation, this is as follows:

$$Ca_3(PO_4)_2 + 2H_2SO_4 = CaH_4(PO_4)_2 + 2CaSO_4.$$

In the presence of water, which has been omitted from the above equations in order to simplify them, the calcium sulphate would be changed

into gypsum by abstracting water from the mass. The last reaction is the one desired by the manufacturers.

To utilize low-grade rock and tailings, and to make concentrated phosphatic fertilizers, the phosphoric acid produced by the first reaction is evaporated in pans until it contains about 45 per cent. phosphoric anhydride. It is then treated with a fresh supply of phosphate rock, when the following reaction ensues:

$$Ca_3(PO_4)_2 + 4H_3PO_4 = 3CaH_4(PO_4)_2.$$

It will be observed, therefore, that ordinary superphosphate is largely a mixture of soluble calcium phosphate and gypsum, while the double-acid phosphate contains little or no calcium sulphate, or dehydrater, and thus has to be artificially dried. Either the phosphoric acid itself, the double phosphate, or such compounds as ammonium phosphate, might be shipped from our Western field, since they are highly concentrated products.

SUBSTITUTES FOR PHOSPHATE ROCK

The use of substitutes for ordinary phosphate rock has been in the past of great importance, but, since the discovery of large deposits of high-grade phosphate rock, the price of phosphatic fertilizers has greatly decreased, as a result of which the more highly priced guanos and similar materials have been driven out of the market. The use of substitutes, however, still continues. There is still much low- and medium-grade material which could, if necessary, actually take the place of phosphate rock as a source of phosphorus and which may possibly be used at some future time as commercial conditions change.

In the following pages, the different substitutes for phosphate rock that have suggested themselves are named and briefly described. They may be classified in two groups: (1) The natural and (2) the artificial substitutes for phosphate rock. Under the natural substitutes come (*a*) phosphatic limestone; (*b*) other phosphate-bearing minerals, such as apatite, nelsonite, wavellite, and others; (*c*) guano; (*d*) marl; (*e*) excrement, both human and animal; (*f*) bones. Within the class of artificial substitutes may be included (*a*) the basic slags and (*b*) manufactured compounds, like ammonium phosphate; the double-acid phosphate, phosphoric acid, and ordinary superphosphate from the low-grade phosphate rock discarded at phosphate mines.

The Natural Substitutes for Phosphate Rock

Phosphatic Limestone

Directly below the phosphate-rock horizon in the Mount Pleasant and other parts of the Tennessee field, occurs the phosphatic limestone from which the brown rock itself has been derived. There must be an

enormous tonnage of this phosphatic limestone scattered throughout the phosphate-rock areas of middle Tennessee. A long period of time must elapse before any attention will be given to this comparatively low-grade material as a source of phosphate, but it would be hazardous to say that this will never be done. Of course, in the mined-over area, much of the richer limestone has been covered so deeply that it will be difficult and expensive to get at. Analyses of this limestone, some of which was in a leached and some in a partially leached condition, occurring in horses between cutters, show calcium phosphate ranging from 4 to more than 42 per cent. The carbonate and the phosphate of lime mixture in this material has considerable value as fertilizer when applied directly to the land in finely pulverized form, and, although it is difficult to predict how and when this material will be utilized, it seems fairly certain that it will prove of value at some future time.

Under present conditions of mining, the phosphate is dug from around the limestone horses and boulders, and the pits are then either abandoned or filled with material from the overburden. The time has not arrived for the utilization of such material, but, when it does, it will be removed, broken up, crushed, and spread on the land, either with or without treatment with acid, or, as suggested by Waggaman,[7] the phosphatic limestone may be burned in a kiln and then slaked with steam or hot water and the rock thus disintegrated. In experiments made by heating the phosphatic limestones, after their phosphate content had been determined, it was found that the percentage of the latter was increased in quantity, the increases ranging from 1.40 to 8.62 per cent.[8]

Apatite, Nelsonite, and Other Minerals Containing Phosphate

The mineral apatite is among the most definite in composition, if not the most definite, of the crystalline phosphate-bearing minerals. It is widely distributed and occurs in rocks of various kinds, but most commonly in those of the metamorphic and crystalline types, such as crystalline limestone, dolomite, gneiss, the mica schists, etc. The two common varieties are fluor-apatite, $3Ca_3(PO_4)_2.CaF_2$, and chlor-apatite, $3Ca_3(PO_4)_2.CaCl_2$, with intermediate compounds containing both chlorine and fluorine. The normal varieties contain in the case of fluorine 42.3 per cent. phosphoric acid (P_2O_5), and the chlor-apatite 41 per cent. of phosphoric acid (P_2O_5). The fluor-apatite is much the more common variety and here belongs the apatite found in St. Lawrence County, N. Y., Canada,[9] Spain, and the Alps. The Norway apatite is of the chlor-

[7] *U. S. Bureau of Soils, Bulletin* No. 81 (1912), 13.

[8] *Loc. cit.*

[9] R. W. Ells: *Geological Survey of Canada, Mineral Resources of Canada, Bulletin on Apatite* (1904).

For a complete list of minerals containing at least 1 per cent. of phosphoric acid, see W. B. Phillips: *Trans.* (1892–93), **21**, 188–196.

ine-bearing type. The apatite associated with the magnetite at Mineville, northern New York, on the property of Witherbee, Sherman & Co., is worthy of special mention. Attempts have been made, according to report, to separate the apatite on a commercial scale, but they have not proved successful.

Apatite in Virginia.—Rock of igneous origin, rich in titanium and phosphorus, occurs in the eastern foothills of the Blue Ridge in Virginia, near Roseland, about 7 miles northwest of Arrington on the Southern Railroad, and 24 miles northeast of Lynchburg. The titanium occurs in the form of ilmenite, and the phosphorus in the form of apatite, these being the dominant minerals. The rock occurs in dikelike forms of varying size and irregular shape, and to it the name nelsonite has been applied.[10]

Many varieties of rock have been included under this name, for example, ilmenite nelsonite, magnetite nelsonite, biotite nelsonite, hornblende nelsonite, and rutile nelsonite, from the dominant mineral that may be present, and gabbro nelsonite, from the rock-type gabbro, which some of the facies of nelsonite resemble. Ilmenite nelsonite, to which the name was first applied, is the normal and most abundantly occurring variety. The apatite present is the fluor-apatite, chlorine being present only in traces. The content of the nelsonite in apatite ranges from 0.3 to 30 per cent.

Experiments having as their object the commercial utilization of nelsonite have been carried on at the Bureau of Soils. Both the minerals ilmenite and apatite contain elements which seriously affect each other so far as commercial applicability is concerned. The problem, then, first, is to separate them. Two mechanical methods have been tried, depending (1) on differences in specific gravity, (2) on the magnetic properties of the ilmenite. Neither of these methods was found entirely satisfactory. The chemical method was then resorted to. In the experiments performed by W. H. Fry, of the Bureau of Soils, it has been found that ilmenite may be almost completely freed from apatite by means of sulphuric acid with a minimum of waste and without involving great expense. Moreover, all the products obtained can be utilized commercially.

Preliminary experiments showed that ilmenite is entirely unattacked by dilute sulphuric acid, while apatite is acted on quite energetically by this reagent. Subsequently, it was shown that the apatite remaining in ilmenite after mechanical separation can nearly all be extracted by means of dilute sulphuric acid without appreciably affecting the ilmenite. The

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[10] T. L. Watson: *Mineral Resources of Virginia* (1907), 300; Geology of the Titanium and Apatite Deposits of Virginia, *Virginia Geological Survey, Bulletin* 3*A* (1913), 100 *et seq.*; T. L. Watson: and Stephen Taber: Virginia Rutile Deposits, *U. S. Geological Survey, Bulletin* 430 (1910), 206.

details connected with the experiments are outlined in an article by Waggaman.[11]

Thus the possibility is shown of obtaining phosphate fertilizer material from this occurrence.

The great objection to apatite as a source of phosphate is the expense of mining and preparing the rock for treatment with acid. In the case of the apatite containing fluorine, hydrofluoric acid gas and possibly other poisonous gases are given off when the mineral is acted upon by sulphuric acid. Thus, the treatment of this mineral with acid may be attended with danger, unless care is exercised in the manipulation, or unless there is enough of silica or silicates present to react with the liberated hydrofluoric acid. The cheap and accessible sources of phosphate have caused a practical stoppage of the mining of apatite for fertilizer purposes, though at one time a considerable amount of the mineral was so used.[12]

Guano

Guano consists chiefly of the excrement of birds and bats, and has been found in considerable quantity in some places. It has been extensively used in the manufacture of acid phosphate. There are two types: (1) The unleached deposits found in caves and sheltered places, which contain phosphoric acid in readily available form, and also nitrogen, a fertilizer of great importance and high price. (2) The leached deposits, which contain no nitrogen and in which the phosphoric acid content, though high, is generally insoluble. Accessible and valuable deposits of guano are now rather scarce, and only those having the best transportation facilities are being worked.

Greensand (Marls)

Phosphate is found to a small extent in certain greensands of the eastern and southeastern States, notably in New Jersey, Kentucky, Tennessee, South Carolina, Florida, Alabama, and without doubt in many other coastal-plain States of the East.

New Jersey.—About 30 years ago, the greensands of certain portions of New Jersey were in great demand. On the first geologic map of that State, the location of the beds containing them was shown, and in some of the earlier reports the deposits were described and numerous analyses given, as well as instructions for their use. In recent years, however, greensand has been supplanted to a large extent by the more highly concentrated artificial fertilizers and is no longer dug extensively.

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[11] W. H. Waggaman: A Possible Commercial Utilization of Nelsonite, *Journal of Industrial and Engineering Chemistry* (1913), 5, No. 9, 730–732.

[12] Ells: *Loc. cit.*

The analyses in Tables 4 and 5 show the composition of the different grades of greensand as dug and applied to the soil. The glauconite in them is of nearly uniform composition, but mixed with it are carbonate, sulphate, and phosphate of lime, quartz sand, sulphide and phosphate of iron, shells, etc. The differences in the kind and quantity of these substances cause wide differences in the appearance of the greensand containing them, as well as in its composition and properties.

Table 4 gives the phosphoric-acid content, in percentages, of typical specimens of New Jersey greensand:

TABLE 4.—*Phosphoric Acid in Typical Greensand (Marl) of New Jersey*

	1	2	3	4	5	6	7	8	9	10
Phosphoric acid......	1.14	1.33	1.02	2.24	2.69	2.56	3.58	3.87	2.58	2.30

New Jersey greensand[13] has been of incalculable value to the region in which it is found. It has raised this region from the lowest stage of agricultural exhaustion to a high state of improvement. Found in places where no capital and but little labor were needed to get it, the poorest people have been able to avail themselves of its benefits. Lands which in the old style of cultivation had to lie fallow, by the use of marl produce heavy crops of clover and grow rich while resting. Land which had been worn out, and left in commons, is now, by the use of this fertilizer, yielding large crops of the finest quality. Everywhere in the marl district, may be seen farms which in former years would not support a family but which are now making their owners rich through their productiveness. Bare sands, by the application of marl, are made to grow clover and then crops of corn, potatoes, and wheat. "Pine barrens," by the use of marl, are made into fruitful land. The price of land in the greensand-marl belt of New Jersey was considerably below that in the northern part of the State 40 years ago; now the price is higher.

Kentucky.—The greensand (marls) of the Leitchfield, Kentucky, region have been described by N. S. Shaler[14] and have been analyzed by Robert Peter.[15] A sample from near Leitchfield, Grayson County, was sent to the United States Geological Survey by M. H. Crump, of Bowling Green, Ky. It was analyzed in the Survey laboratory by Chase Palmer and found to contain 3.73 per cent. potash (K_2O) and 0.13 per cent. phosphoric acid (P_2O_5).

Greensands containing approximately these quantities of potash and phosphoric acid are found in large quantities over a considerable area in west central Kentucky, and in the reports cited above their use was sug-

[13] *Annual Report of the State Geologist of New Jersey* (1886), 154.
[14] *Kentucky Geological Survey, Report, new ser.* (1877), **3**, 46–47.
[15] *Kentucky Geological Survey, Chemical Analyses* (1884), **A**, 250–254.

gested to rejuvenate the lands of the State, worn out as a result of the excessive cultivation of tobacco and other crops.

Tennessee.—The greensands of Tennessee were mentioned by G. Troost as early as 1835. They are found in Hardin, McNairy, and Henderson Counties. The analyses by Troost shown in Table 5 are of the greensands of McNairy County:

TABLE 5.—*Analyses of Greensand, McNairy County, Tenn.**

	1	2	3
Silica (SiO_2)	48.00	45.30	51.70
Alumina (Al_2O_3)	7.00	6.20	6.50
Ferrous oxide (FeO)	20.70	18.00	21.20
Potassa (K_2O)	10.10	10.40	11.30
Carbonate of lime ($CaCO_3$)	5.70	10.80	2.00
Water (H_2O)	8.00	8.50	7.30
Loss	0.50	0.80	0.00
Total	100.00	100.00	100.00

Phosphoric acid, which is doubtless present in the marls, does not appear to have been separated.

Alabama.—Materials of several kinds, containing phosphoric acid, have been found in Alabama, in both the Cretaceous and Tertiary formations. In several places in the lower part of Marengo County, notably near Dayton and Nixonville, there occur tolerably compact beds of shell-casts, containing from 20 to 25 per cent. phosphoric acid. According to E. A. Smith,[16] these beds are the most promising sources of phosphate in the State. Such high-grade phosphate beds should have future value as fertilizers.

Excrement

A large part of the phosphorus sent from the country in the form of cattle and grains for consumption in the cities, finds its way ultimately, via the modern sewage system, to the ocean. There it is lost so far as its fertilizer value to the land is concerned. Whitson[17] estimates that the loss in the cities due to human excreta alone is equivalent to 2 or 3 lb. of phosphoric oxide per acre for the entire cropped region of the United States. Supposing this loss to be 2 lb., $\frac{1}{1,000}$ ton, this amounts, for

* J. M. Safford: *Tennessee Geological Survey*, 515. Nashville, 1869.

[16] E. A. Smith: The Phosphates and Marls of Alabama, *Trans.* (1895), **25**, 811.

[17] A. R. Whitson and C. W. Stoddart: The Conservation of Phosphates on Wisconsin Farms, *University of Wisconsin Agricultural Experiment Station Bulletin* No. 174 (April, 1909), 1–8.

400,000,000 acres of cropped land, to 400,000 tons of phosphoric oxide—equivalent to 1,200,000 tons of phosphate rock.[18] Surely here is a problem for the municipal and chemical engineer of the future.

Bones

Before phosphate rock was found in quantity in the United States and in other parts of the world, bones were the main source of phosphoric acid for fertilizing purposes. There is still a domestic production of this form of phosphate fertilizer. The bones were steamed, charred or burned, and applied to the land after such treatment, or they were made into superphosphate in the way ordinary bone phosphate is now converted, by treatment with sulphuric acid. Ground bone, or bone meal, is a valuable fertilizer without any treatment whatever, due to its nitrogen and phosphorus. The phosphoric acid contained in bones has considerable value as a source of phosphorus for chemical purposes other than fertilizer.

Artificial Substitutes for Phosphate Rock

Basic Slag[19]

Basic slag is not soluble to any extent in water, but it has been shown that the phosphate contained in it is readily assimilated by growing crops and splendid results have followed from its use as a fertilizer.

It is produced in the manufacture of steel by the basic Bessemer and basic open-hearth processes. In these the phosphorus is separated from the iron by the addition of lime or limestone, which, having a strong chemical affinity for the phosphorus, forms with it basic phosphoric compounds which enter the slag. In slags produced by these processes, the phosphoric acid may run from 10 to more than 25 per cent.

The iron ores used in this country, with the exception of some mined in Alabama, are low in phosphorus and consequently the basic Bessemer process is not applicable to them. The slags produced from such low-phosphorus ores are consequently low in this fertilizing element, but abroad, and especially in Germany, where the iron ores are high in phosphorus, the basic Bessemer process is widely used and as a result the slag, rich in phosphate, is widely applied as a fertilizer. There are indications that the importance of these high-phosphatic slags is coming to be realized in this country.

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[18] C. R. Van Hise: *Conservation of Natural Resources in the United States*, 325. The MacMillan Co., New York (1910).

[19] E. C. Eckel: Utilization of Iron and Steel Slags, *U. S. Geological Survey, Bulletin No.* 213 (1903), 225–227.

W. H. Waggaman: Utilization of Acid and Basic Slags in the Manufacture of Fertilizers, *U. S. Bureau of Soils, Bulletin No.* 95 (1913).

In rock phosphate the phosphoric acid is combined with lime as tricalcium phosphate with the symbol $Ca_3(PO_4)_2$. In slags, however, the combination is that of tetra-calcium phosphate ($4CaO.P_2O_5$ or $Ca_4P_2O_9$), according to certain investigators. Other experimenters have found that different compounds have formed with variations in the composition of the molten bath and that toward the end of the operation, when much silica was present, a compound having the formula $P_2O_5.SiO_2.5CaO$ was formed. It has also been shown that in slags having the same phosphate content, those containing silica were the most soluble and hence the most valuable for fertilizer purposes.

PHOSPHATE ROCK RESERVES

FLORIDA[20]

Hard Rock

Several maps of the Florida phosphate fields have been published, but they are of a general nature and attempt to show only the approximate location of boundaries. Even this is difficult to do with any degree of accuracy without careful prospecting. In estimating the available hard rock, some idea of the total area in the State within which such deposits may occur is essential. The estimates made agree fairly closely, and indicate that the workable beds of hard phosphate rock in Florida occur throughout an area of several hundred square miles. The area actually underlain by workable deposits of hard rock is, however, but a small fraction of that within which the deposits have been mapped. There are, moreover, no existing data from which one may calculate the area underlain by workable deposits with a degree of accuracy that would have any practical value whatever.

Sections of square miles could be taken within which the deposits have been most completely mined out and tonnage estimates made from them, but figures thus obtained could not be used in other areas as a standard, since over many square miles there are no deposits at all. The deposits in the sections which have been most completely mined out, moreover, are usually the most accessible. With due allowances, it is conservatively estimated that there is as much hard rock available in Florida as has already been removed, that is, approximately 10,000,000 tons, and with an annual output of 500,000 tons the hard-rock phosphate deposits may be expected to last at least 20 years longer.

Land Pebble

The land-pebble beds are more regular in their occurrence than the hard-rock deposits, but close estimates cannot be made except by actual

[20] The writer is greatly indebted to Dr. E. H. Sellards, State Geologist of Florida, for valuable suggestions in preparing this note on the Florida phosphate-rock reserves.

prospecting; and this will be done only gradually by those who are interested in or are engaged in mining. The land-pebble phosphate belt is approximately 30 miles long by 5 to 10 miles wide. On the basis of a conservative estimate of acreage and of tonnage per acre, the writer has calculated a total of 190,000,000 tons of land pebble.

The output of land pebble per year in Florida is, in round numbers, 2,000,000 tons. The estimate of available land pebble, which is considered extremely conservative, leads to the conclusion that this type of phosphate rock in Florida will last several generations, and for present purposes it may be considered practically inexhaustible. The refinements in methods of mining land pebble are gradually reducing the quantity of small pebbles that go to the waste dump, and this factor will tend to prolong the life of these deposits beyond that calculated from the figures given above.

In making up the estimates for Florida, river pebble has not been included, owing to the difficulties connected with estimating its quantity. This factor also adds to the conservatism of the figures given for this State.

Waste Material

The phosphoric acid in the Florida deposits in the form of soft phosphate, so-called, together with large quantities of aluminum and iron phosphate, go to the dumps in the preparation of the hard and pebble rock for market. The loss calculated in terms of phosphate of lime is considerable and it may possibly equal the actual quantity saved and marketed. It has been calculated by W. H. Waggaman[21] that the marketed material is probably not more than 15 per cent. of the total material mined, and that in the discarded material is an average of at least 10 per cent. phosphoric acid. The total quantity of Florida phosphate rock marketed up to and including 1914 is approximately 27,500,000 long tons. On these bases, the low-grade material in the waste heaps is approximately equivalent to 27,000,000 to 30,000,000 tons of high-grade material.

Tennessee

Introductory Note

There are three varieties of phosphate rock in Tennessee, the white, the blue, and the brown. The very irregular character of the white phosphate-rock deposits, and the fact that they have been but meagerly prospected, make it impossible to estimate the available supplies even approximately. The omission of estimates of white rock may be considered to contribute to the conservatism of the total figure for the State.

[21] W. H. Waggaman: *U. S. Bureau of Soils, Bulletin No.* 76 (1911), 14.

Blue Rock

The blue bedded phosphate-rock deposits are found chiefly in Perry, Hickman, Lewis, and Maury Counties. In estimating them, only rock 24 in. and over in thickness will be considered. The area underlain by rock of this thickness is large, but the area in which the rock is thinner is much greater. This estimate includes the rock under heavy cover, but this is not a bar to its availability, since blue rock is worked wholly underground.

As stated in another part of this paper, blue phosphate rock, even in those localities where thickest, is extremely variable and may locally be absent altogether. As an offset to this factor, however, a very large area has been omitted in the calculations in which the rock has been considered to be less than 24 in. in thickness, but which future prospecting may show to be thicker. It may reasonably be expected that outlying areas in which the rock is locally workable will be found in the future. It must also be considered that rock only a foot thick, but of high-grade, may be considered of future workability. It is also possible that all of the blue rock estimated may not be up to present commercial requirements. As time goes on, however, the standard in this respect will tend to fall, and ore may then be treated chemically to raise the grade for shipment to distant points.

Taking all the above factors into consideration, it is estimated that there are approximately 84,000,000 tons of available blue rock phosphate in Tennessee.

Brown Rock

The brown phosphate rock included in the estimate given below occurs chiefly at the Bigby horizon. It is found chiefly in Maury, Giles, Hickman, Lewis, and Sumner Counties, Tennessee. The Mount Pleasant, Maury County, district contains the most extensive deposits, and the major operations on brown rock phosphate in the State are located there. The quantity of brown phosphate rock in the Mount Pleasant district, which may be taken as a type, ranges from 600 to 1,000 tons per acre-foot; and 850 tons is considered a fair average. A considerable portion of the areas where are located important deposits of brown rock has been worked over in Tennessee, especially in the Mount Pleasant field. This fact, and the additional fact that the entire acreage can not be underlain by phosphate rock, would tend to reduce the estimate; but, on the other hand, it is very reasonable to suppose that as detailed prospecting takes place new and important areas, though small, will be found. Two such areas have recently been brought to the writer's attention. Taking these factors into consideration, it is estimated that an available tonnage of approximately 4,000,000 tons of brown rock

remains in middle Tennessee, or approximately about as much as has already been marketed.[22]

In addition to the virgin rock remaining unmined in Tennessee, a great deal of phosphate rock will be recovered from waste ponds and from the dumps from old workings. Such work is in progress, as described earlier in this paper. The recovery of this material will tend to greatly prolong the life of the brown phosphate-rock field beyond the period indicated by a theoretical calculation based on the estimate of reserves made above and the yearly marketed output of brown rock in Tennessee.

South Carolina

Several estimates have been made of the quantity of phosphate rock available in South Carolina for future use. In the very early days of the industry, the data for such estimates were lacking, but increased knowledge gained from working the deposits has lessened this difficulty to some extent. Shepard, in 1880, estimated the total available supply to be less than 5,000,000 tons, but from 1881 to 1914, inclusive, the quantity of rock removed was 11,600,000 tons out of a total of 13,000,000 tons for the State since the industry began in 1867.

P. E. Chazal[23] in 1904 estimated the quantity of rock remaining in the land deposits at between 9,000,000 and 11,000,000 tons. Since 1904, there have been removed from the South Carolina deposits more than 2,000,000 tons, leaving still available between 7,000,000 and 9,000,000 tons of phosphate rock. These figures do not include the river rock. The higher figure, namely, 9,000,000 tons, may therefore be taken as a conservative estimate of the available supply in South Carolina.

The annual production of South Carolina phosphate rock is approximately 100,000 tons. At the present rate of production, the phosphate rock supply of South Carolina would last approximately 90 years. In view of the fact that the production of this State has been steadily decreasing for many years, and the possibility that other deposits may be found, it seems safe to conclude that the South Carolina deposits can produce rock of somewhat low grade for many years to come and should become an important source of supply when the chemical concentrating methods come into general practice.

It is interesting in this connection to note that W. H. Waggaman[24]

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[22] James A. Barr: *Trans.* (1916), **54,** 475 states: "In the event of low grades of phosphate becoming commercial products, 20,000,000 tons would become available within a 50-mile radius of Mount Pleasant alone, and perhaps 100,000,000 tons of rock, especially if phosphatic limestone is taken into account."

[23] P. E. Chazal: *A Sketch of the South Carolina Phosphate Industry* (1904), 17 and 18.

[24] W. H. Waggaman: *Journal of Industrial and Engineering Chemistry* (June, 1914), **6,** No. 6, 464.

has estimated a somewhat higher available supply of phosphate rock of high grade in South Carolina, namely, 10,000,000 tons. F. B. Van Horn's[25] estimate made in 1909 places the available South Carolina phosphate rock at 3,000,000 tons, but that writer qualifies the estimate by observing that careful and deep prospecting may increase these figures.

Kentucky

The rock occurring in this field is of the brown-rock type, similar to the brown phosphate-rock deposits occurring in middle Tennessee. The areas underlain by high-grade deposits are scattered, but the most important areas occur to the south and northwest of Midway, Woodford County. The writer has explored this region with the drill, though not closely enough to give accurate figures. A very conservative estimate of the phosphate rock available in this region is 1,000,000 tons.

Arkansas

Though there is only a small production of phosphate rock in Arkansas at the present time, it must not be inferred that the deposits in this State will not prove of future value. In many of the analyses published by Branner and Newsom,[26] the content of the iron oxide and alumina runs high, and although this factor tends to unfit the Arkansas rock for the manufacture of superphosphates under present conditions, the time without doubt will come when these deposits will be extensively exploited for direct application to the soil. Chemical treatment should also render a very large part of the Arkansas phosphate available for future use. The tonnages per acre calculated for small tracts in which conditions are well known, show a large quantity of phosphate rock.[27] Calculations of tonnages indicate as much as 11,600 tons in one locality where the thickness of the rock was approximately 4½ ft. In another place, the acre tonnage was 23,200 where the average thickness was considered to be 8 ft. The latter figure is reckoned on the basis of 2,900 tons per acre, which, in turn, is based on a weight of 150 lb. per cubic foot of rock. Waggaman[28] estimates 20,000,000 tons of high-grade rock in Arkansas.

The Western States

The available tonnage estimates for the Western field, which are only partial, are chiefly for Idaho, but include figures for small parts of Utah,

[25] F. B. Van Horn: Phosphate Deposits of the United States, *U. S. Geological Survey, Bulletin No.* 394 (1909), 164.

[26] *Arkansas Agricultural Experiment Station, Bulletin No.* 74 (1902), 116–119.

[27] *Loc. cit.*, 71–80.

[28] W. H. Waggaman: *Journal of Industrial and Engineering Chemistry* (June, 1914), **6**, No. 6, 464.

Wyoming, and Montana. The estimates prepared, which are considered conservative, are all for high-grade rock, 65 per cent. or more tricalcium phosphate, and refer chiefly to the main bed which ordinarily lies near the base of the phosphate shales and in the Idaho field is usually 5 or 6 ft. thick. Thus they do not include the thinner beds of high-grade rock nor the great body of low-grade material. The main bed itself is included only for those parts of the field in which, according to the present practice of the Geological Survey, the rock lies at depths considered workable. The estimates are based on the assumption that the phosphate rock at depth, and remote from the outcrop, maintains the generally uniform qualities displayed at the surface in so many parts of the field. Some tendency toward enrichment by weathering has indeed been noted, but observations thus far obtainable suggest no marked decrease in richness within the body of the material.

Under the above conditions the areas examined in the field work of 1909 to 1913, inclusive, in Utah, Wyoming, and Idaho, are estimated to contain at least 5,290,296,900 long tons of high-grade phosphate rock. In addition, the Elliston field in Montana is estimated to contain 86,000,000 short tons,[29] or approximately 76,785,700 long tons, a total available estimate of 5,367,082,600 long tons. Since 1913, additional areas in some of these regions have been examined in detail and the presence of still more of the high-grade rock has been determined. There yet remains a considerable area of withdrawn land that has not been examined, in which it is probable that high-grade phosphate will be found. When the results of these examinations and the results of more detailed work in regions now known only by reconnaissance are added to the figures given, it is probable that the estimates of high-grade rock in the Western field will be considerably increased.

Total Tonnage Available

In Table 6 is given the estimated tonnage of phosphate rock available in the United States at the present time.

Table 6.—*Phosphate Rock Available in the United States*

Eastern States	Long Tons	Western States	Long Tons
Florida	227,000,000	Montana, Idaho, Utah, and Wyoming	5,367,082,600
Tennessee	88,000,000		
South Carolina	9,000,000		
Kentucky	1,000,000		
Arkansas	20,000,000		
	345,000,000	Total	5,712,082,600

[29] R. W. Stone and C. A. Bonine: The Elliston Phosphate Field, Montana: *U. S. Geological Survey, Bulletin No.* 580 (1915), 382–383.

The United States is now producing for domestic use and export about 3,000,000 tons annually.[30] More than 99 per cent. of this comes from the Eastern States, and in 1914 nearly 80 per cent. came from Florida. On this basis, Eastern phosphates should last fully 100 years, taking into account material of good grade.

FOREIGN PHOSPHATE DEPOSITS

Important deposits of phosphate rock are located outside the United States. Perhaps the best known of these foreign deposits are those in the South Sea Islands, Christmas Island in the Indian Ocean, the African deposits, and those on Curaçao in the Dutch West Indies. In Africa, the principal deposits are located in Egypt, Tunis, and Algeria. In the South Pacific, Angaur of the Pellew group of islands, Nawoda (Pleasant), Panapa (Ocean), and Makatea (Aurora) islands, all contain important deposits. A deposit has recently been reported in northern Chile.

African Deposits

Egypt[31]

The development of the extensive deposits of phosphate near the Red Sea has, during the past 2 years, assumed important proportions. The mines are worked by a British concern and are connected by rail with the Red Sea, where the rock is loaded on steamers for export. The rock contains 65 per cent. or more of tricalcic phosphate.

A company managed by Italians and founded in 1912, has obtained extensive concessions about 12 miles inland from Kosseir and also at Sebaia, on the eastern bank of the Nile, between Keneh and Assouan. The former concession is being connected with the port of Kosseir by a light railway, which should shortly be completed. The rock from the latter mines will be transported down to the Nile by a ropeway and thence to Alexandria by boat for shipment.

The total output of phosphate in Egypt for the years 1908 and 1912 was as follows: 1908, 700 tons; 1909, 1,000 tons; 1910, 2,397 tons; 1911, 11,925 tons; and 1912, 69,985 tons. According to the *Financial Adviser's* report, the output during 1913 exceeded that for 1912 by about 33,000 tons.

Although other beds of phosphate are found in various districts in Egypt on both sides of the Nile Valley, the Red Sea area is responsible for almost the whole output. The rapid development of the business of the Egyptian Phosphate Co., the British concern, and the impending com-

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[30] Normal conditions are referred to.

[31] U. S. Consul-General Olney Arnold: *U. S. Daily Consular and Trade Reports* (Aug. 20, 1914), 991.

mencement of active operations by the other company, will lead to a considerable increase in production.

Practically all the raw phosphate produced is shipped from Egypt, principally to Japan. That country in 1912 took about 49,000 tons out of a total export of 52,000 tons, and in 1913, 59,000 tons out of a total of 64,000. According to the customs' returns, the average value of the raw phosphate shipped during 1913 at Port Safaga was about £1 ($4.87) per ton.

A foreign correspondent of the Survey gives the following interesting information with reference to the extent and continuity of the North African deposits:

They correspond geologically to the deposits of phosphate rock in Tunis and Algeria, and there can be little doubt that the phosphate-bearing formation extends from Algeria to the Red Sea, a distance of about 2,000 miles. The intervening territory, comprised in Tripoli and the Lybian Desert, has not been explored. There is also evidence that the same phosphate-bearing formation extends beyond the Red Sea and through the Arabian Desert into Persia, a farther distance of at least 1,000 miles, and there is a possibility that it extends much farther.

Algeria and Tunis[32]

The deposits of phosphate rock in Algeria are continuations of the deposits in Tunis. The two important mining districts in Algeria are located near the towns of Setif and Tebessa, in the eastern part of the State. They have been developed during the last 15 years, the production having increased from 1,057 short tons in 1899 to 550,000 in 1912. The percentage of lime phosphate in the rock exported from the Setif district ranges from 58 to 63 per cent., and that in the rock from the Tebessa district, from 58 to 68 per cent.

Tebessa District.—The deposits of Kouif, the most important exploited in Algeria, are located near Tebessa, close to the Tunisian frontier. Of the five beds, three are workable, the thickness ranging from 3 to 9 ft., 3 to 4½ ft., and 1⅘ to 3 ft., in the different beds. The average percentage of phosphate in the rock varies, being, respectively, 58.64 per cent., 68.50 cent., and 48 per cent. in the three beds. Where the overburden does not exceed 24 ft. in depth, open-cut mining is practised; where the overburden is deeper, the rock is mined by tunneling. Because of the basin, or saucer shape of the deposits, the tunnels are inclined.

Important deposits of phosphate rock have been found at Dy Nord and Djebel-Onck, about 62 miles south of Tebessa, which are believed to contain 300,000,000 to 400,000,000 tons of rock. These deposits have not been thoroughly explored, and estimates are therefore not entirely dependable.

[32] *U. S. Daily Consular and Trade Reports* (Aug. 30, 1913), 1240–1242.

Setif District.—In 1906, La Compagnie des Phosphates de Paris leased for 20 years the deposits in the commune of Bordj-Rhir. Its shipments are made by the way of a 12-mile cable to El Anasser, on the railway from Setif to Algeria; thence they go to the part of Bougie, from which the exports amounted to 64,986 short tons in 1910, 64,824 tons in 1911, and 62,702 tons in 1912.

La Compagnie Algérienne des Phosphates is exploiting two deposits in the commune of Tocqueville. The beds are from 1 to 6 ft. thick. The phosphate is transported by a 90-mile narrow-gage branch line to the railway station of Texter-Tocqueville. In 1900, 17,807 short tons, and in 1912, 230,864 short tons of rock were mined. At present 350 workmen are employed.

Another French company, with a capital of $1,000,000, is exploiting the Mzaita mine in the communes of Maadia. A broad-gage railroad has been built from the mine to the station of Ain-Tassers, and an electric plant of 150 hp. has been installed for the stamp mill. Only 7,560 tons of rock were extracted in 1912, but it is expected that eventually 300,000 tons of rock will be mined annually. The estimated supply of rock is 16,500,000 tons.

Southern Pacific Ocean (South Sea) Islands

The South Sea islands contain the richest, if not the most extensive, phosphate deposits in the world. Japan, Australia, Hawaii, and our own Pacific coast under normal conditions, take very nearly half the production of the British, French, and German South Sea companies engaged in mining the phosphate rock. The deposits in the South Seas are notably high in calcium phosphate, and contain from 85 to 90 per cent. of this ingredient. They are also notable in their very low content of iron oxide and alumina, which ranges usually below 1 per cent.

Angaur (Pellew Group)

It is understood that the Japanese Government is now in control of the Pellew group of islands in the South Pacific, the islands having passed from German to Japanese control since the outbreak of hostilities in Europe. The principal deposits of phosphate rock are located on Angaur Island, and these are reported to be now worked to supply the Japanese fertilizer manufacturers. The deposits on the island are reported to contain in round numbers from 2,000,000 to 3,000,000 tons of high-grade phosphate rock.

Ocean Island

Ocean Island is estimated to contain not less than 50,000,000 tons of the highest grade of phosphate rock. Some shipments from this island

are reported to run from 87 to 89 per cent. calcium phosphate and less than 1 per cent. of iron oxide and alumina.

Makatea (near Tahiti)

On Makatea, near Tahiti, it is estimated that about 10,000,000 tons of phosphate rock of high grade are available.

Other Foreign Deposits

Christmas Island

Christmas Island, near the west end of Java in the Indian Ocean, produces as high-grade phosphate rock as is found in the South Sea islands. Great secrecy is maintained as to the quantity of phosphate existing on the island. It has been assumed that there is an available tonnage of high-grade rock amounting to 8,000,000 tons, but this estimate is not considered authoritative.

Curaçao (Dutch West Indies)

After a suspension lasting approximately 20 years, the reorganized Curaçao Phosphate Mining Co. began operations in June, 1913, at its Santa Barbara mines, shipped its first cargo in October, and is pushing business now. The phosphate goes to Germany and England. About 200 men are employed, and in one way or another a considerable amount of money is left in the island because of the industry.[33]

Curaçao produces the same high grade of phosphate rock as do the South Sea Islands. It is reported that there came from this island in 1914 about 100,000 tons of phosphate rock averaging 85 to 90 per cent. calcium phosphate. The mining of such high-grade deposits, which will tend to increase as time goes on, owing to the opening of the Panama Canal, may to a certain extent influence the export trade of Florida, but should at the same time tend to conserve these deposits for future use.

Chile[34]

A large, rich deposit of phosphate has been discovered in the valley of the Huasco River, about 300 miles north of Valparaiso. Government engineers are preparing a report thereon, and it is considered of much importance, since the use of phosphate on the farms of Chile is increasing rapidly, with good results. In 1905, only 3,726 metric tons were con-

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[33] Consul Elias H. Cheney: *U. S. Daily Consular and Trade Reports* (April 20, 1914), 359.

[34] *U. S. Daily Consular and Trade Reports* (June 15, 1914), 1560.

sumed in Chile, against 20,000 metric tons for 1912. The Government railways give a reduction of 30 per cent. on transportation charges for fertilizers.

Available Foreign Reserves

From the recent available data, it is evident that the foreign reserves of phosphate are very large, but apparently they are not so large as those within the United States. It must be remembered, however, that the north African deposits, which are thought to extend eastward across Arabia, into Persia, have not been explored sufficiently to know even approximately what their real magnitude is. The Algerian deposits, those considered in some detail in this report and in a former report by the writer,[35] are apparently low-grade, but the apparent tonnage runs up into the hundreds of millions. The high-grade rock of the South Sea Islands is estimated approximately at 70,000,000 tons.

Discussion

E. G. Spilsbury, New York, N. Y.—I would like to emphasize what Mr. Phalen says in his paper regarding the possible utilization of the waste acids in the West for the acid concentration of phosphate rock, and to call attention to the fact that this is already being done by Mr. Laist at Anaconda, who is producing concentrated phosphoric acid from these low-grade rocks. During the last year, he has not been able to carry this along to the extent he had expected, owing to the tremendous demand which they have had for all the acid they can produce, so instead of having a surplus of acid, as was expected, they are really short of this material for their electrolytic copper and zinc work.

[35] W. C. Phalen: Production of Phosphate Rock in 1913, *U. S. Geological Survey, Mineral Resources of the United States*, 1913, Pt. II (1914), 273–289.

An Investigation on Rock Crushing made at McGill University *

BY JOHN W. BELL, † M. SC., MONTREAL, QUE.

(New York Meeting, February, 1917)

Aim of Rock-crushing Experiments

THE aim of the laboratory experiments described in this paper was twofold:

1. To measure as accurately as possible the maximum amount of crushing that can be effected by 1 hp. acting for 24 hr., by the two proposed methods for measuring the amount of crushing, namely: (*a*) by Rittinger's method; (*b*) by Stadler's method, based on Kick's law.

2. To show, if possible, that by one of these methods, the amount of crushing produced by 1 hp. in 24 hr. is a constant amount throughout a wide range in the diameter of the piece crushed, and that by the other method, the amount of crushing is a variable quantity over the same range of diameter; or else show that neither of the proposed methods for measuring the amount of crushing indicates a fixed relation between power and crushing and consequently a new method for computing this amount will have to be found to permit the establishment of a law of crushing.

Experimental Methods

Testing machines, calorimetric methods and commercial types of rock-crushing machines were considered with reference to their value for measuring power used in crushing. The following electrically driven crushing machines, installed and available in the McGill Ore-dressing Laboratory and which had been used in previous rock-crushing tests, on the whole seemed especially suited to the investigation: (1) Comet "A" crusher; (2) 7 by 9-in. Dodge crusher; (3) 10 by 16-in. rolls. In later tests a 3½-ft. Huntington mill was used.

* Abstract of a paper prepared for publication in the *Transactions of the Canadian Mining Institute* giving an account of an experimental investigation of the theory of rock crushing made by John W. Bell, M. Sc., W. G. Mitchell, M. Sc., P. P. Baily, M. Sc., and W. E. Cockfield, M. Sc., in the Ore-dressing Laboratory of McGill University, Montreal, Canada. This paper is a continuation of the discussion of the relative value, for measuring the efficiency of crushing machines, of Kick's law (advocated by Stadler) and Rittinger's theory (advocated by Gates). The abstract is published in this *Bulletin* to afford opportunity for a discussion of this important topic at the New York meeting in February. In this brief abstract only the most important data and conclusions are given.

† Assistant Professor, Department of Mining, McGill University.

Rock Used

In all the series of tests, except one, the rock crushed was a tinguaite—an intruded eruptive obtained from a local quarry. It is a hard and brittle rock, with a tendency to break rather more easily in one plane than in another at right angles to it. It, therefore, is inclined to break in slabs, although the fractured pieces are always irregular in shape.

1914 *Crushing Tests*

In 1914, three series of tests were made to measure the work done per "apparent effective horsepower in 24 hr." In the first series the laboratory Comet "A" crusher was used; in the second series, the Dodge crusher; and in the third series, the rolls.

In certain respects the results of these tests were unsatisfactory, but the data obtained indicated strikingly the probability of a fatal error in Stadler's method and a degree of reliability in Rittinger's. The figures showed an enormous variation in the amount of crushing done by a measured horsepower through a wide range in diameter when calculated by Stadler's method, based on Kick's law, and a remarkable constancy when calculated by the law proposed by Rittinger in 1867. The comparative figures in Table 1 justify fully the indicated, but not necessarily proved, outcome of the whole investigation.

TABLE 1.—*Summary of the Most Efficient of the* 1914 *Tests*

Test No.	Crusher	Diameter of Piece Crushed, Inches	Work Done per A.E. Hp.* in 24 Hr.	
			Measured in Staddler E. U.†	Measured in Rittinger S. U.‡
6	Gyratory	3.50	710	947
27	Dodge	1.20	520	1,030
34	Rolls	0.50	286	1,128
41	Rolls	0.29	138	1,000
47	Rolls	0.18	68	1,022
48	Rolls	0.11	89	1,187
49	Rolls	0.07	52	823

Total range..50 diameters.
Range in roll tests.................................. 7 diameters.

* Apparent effective horsepower. † E. U. = energy units. ‡ S. U. =surface units.

1914–1915 *Rolls Tests*

In the fall of 1914, the fourth series was commenced to investigate in more detail the effect of "tonnage" and "work done per ton" before adopting a set procedure for the final series with rolls.

The decrease in efficiency which results by increasing the feed tonnage, or by increasing the crushing work per ton, for sizes 0.46 in. in diameter and upward was clearly indicated by the results obtained.

The sudden increase in efficiency, when the feed diameter is reduced to 0.30 in., is explained by the marked change in the sound and vibrations emanating from the machine. When crushing 0.46-in. feed at the 30-ton rate, the vibrations in the laboratory concrete floor were perceptible 20 ft. distant from the rolls. When crushing 0.30-in. feed, the floor vibrations practically disappeared, and there was a marked diminution in the smashing shocks, resembling a series of explosions, which were so pronounced during crushing of the larger feeds. In the next and succeeding smaller sizes the characteristic sound effect may be described as being in the nature of a steady grinding noise, indicating that a more uniform pressure replaces the fluctuating and violent forces transmitted to the journals during the crushing of the larger pieces. That there should be an enormous decrease in machine efficiency under the stated circumstances, is in the writer's opinion opposed to reason and common sense, and yet that is the one and only conclusion that can be reached by acceptance of Stadler's theory based on Kick's law, as will be apparent by examination of the data in Table 2 and Fig. 1.

Table 2.—*Summary of Results of* 1914–1915 *Rolls Tests*

Test No.	Remarks	Diameter Feed, Inches	Tons Crushed in 24 Hr.	Work Done per Ton, Rittinger S. U.	Work Done per A. E. Hp.	
					Stadler E. U.	Rittinger S. U.
50	....	1.00	12.2	6.9	623	1,198
53	....	0.70	10.2	9.8	472	1,192
60	....	0.46	5.8	20.6	272	1,097
79–88	Average 9 tests*	0.30	28.7	21.3	190	1,002
89–93	Average 5 tests	0.19	31.3	30.9	150	1,115
94–97	Average 4 tests	0.12	33.8	22.6	109	1,054
98–101	Average 4 tests	0.08	34.5	31.8	82	1,028
102–105	Average 4 tests	0.05	31.6	21.5	77	1,137
106–109	Average 4 tests	0.03	29.8	28.0	77	1,250

Range..............33 diameters. Average R. S. U..............1,120.

* Test 86 excluded.

The Stadler energy unit varies depending not only on the diameter of the feed, but on the work done per ton as well. In tests 50–53 the "work done per ton" is less than in the remaining tests (see Table 2 and Fig. 1) and the Stadler efficiency increases correspondingly.

It follows that there are an infinite number of values of this apparently misnamed "energy unit of crushing," consequently it is not a unit indicat-

ing a fixed relation between power and crushing when the material crushed is rock, however applicable it may or may not be in estimating the power required to crush an ideal and more or less imaginary substance. The unavoidable conclusion to be drawn from the experiments is that the Stadler energy unit is of no value in determining the relative efficiencies of various types of rock-crushing machines. For a single machine, crushing feed of limited range in diameter with only a small variation in the

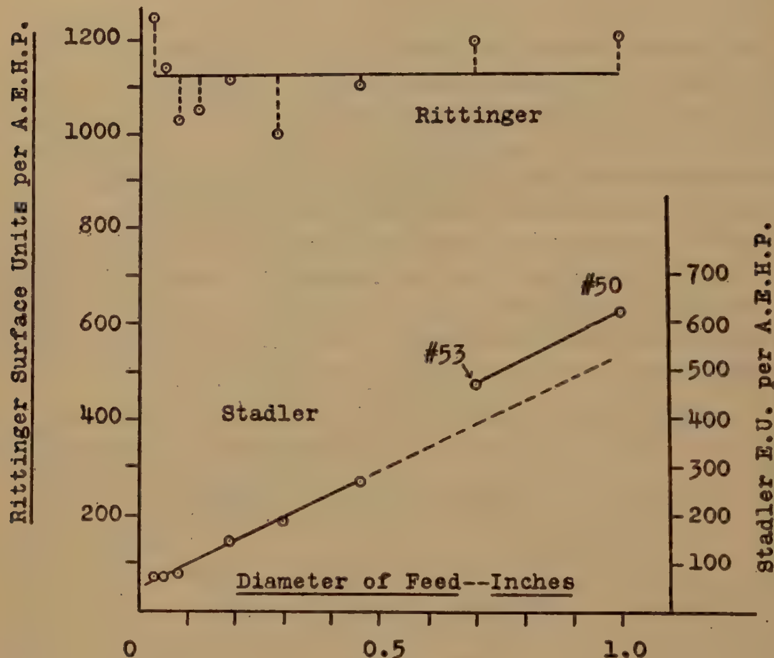

FIG. 1.

work done per ton, Stadler's theory, based as it is on one of the manifestations of crushing, shows the character of the changes in efficiency produced by changes in adjustment, but the Rittinger system, even if it fails to conform with a law, is preferable by reason of its greater sensitiveness to small variations in the amounts of crushing.

Power in Fine Crushing

The sudden increase in the work per apparent effective horsepower in the last four tests (see Table 2), indicated the importance of investigating the relation of power and the crushing of rock particles of very small diameter. Some preliminary work convinced the author that a Huntington mill was admirably suited for the investigation and this machine, with some modifications, was accordingly used.

TABLE 3.—*Summary of Average Results, Huntington Mill Tests*

Test No.	Remarks	Grade	Diameter Feed, Inches	Work Done per A. E. Hp.	
				Stadler E. U.	Rittinger S. U.
114–115	....	+ 20	0.031	55.0	1,590
116–119	....	+ 30	0.020	47.5	1,860
120–122	....	+ 40	0.014	45.5	2,030
125–127	....	+ 60	0.009	34.2	1,970
128–129	Test 130 excluded...	+ 80	0.007	28.1	1,910
131–132	....	+100	0.005	20.2	1,800

Range of diameter tested..........6.

The results in Table 3 show the characteristic drop in Stadler units per apparent effective horsepower, although the range of diameters tested is not large. However, many more experiments would be required to determine the effect of alterations in tonnage, percentage of water in pulp, speed of machine, etc.

Test 116 was made to obtain an indication of the effect of a reduced tonnage. The number of Rittinger units appears to increase, but several check tests would be required to form a definite conclusion.

Considered as a whole, the results are in closer agreement with Rittinger's theory than could be expected, when two obstacles which conspire to defeat the purpose of the investigation are taken into account. One, is the difficulty of measuring the power used only in crushing. The second has to do with the measurement of the surface in the −200 grade. Nobody has yet made a reliable measurement of this, and it is probable that it is a variable factor. For the −200 grade, Stadler's ordinal number 28 was adopted, for although it was realized that it was not necessarily an accurate estimate of the average volume of the particles it was supposed to represent, it would at least serve to make the comparison as fair to one as to the other theory. The corresponding Rittinger number is 780 which was adopted. It is probable that the average factor is appreciably greater than 780.

If in going from coarse to fine crushing there is a considerable decrease in surface in the −200 grades, the results tabulated in this paper would be appreciably affected, although the effect of such a variation would be more disastrous to Stadler's theory based on Kick's law than to Rittinger's, and the Stadler units per apparent effective horsepower might in consequence approach perilously near to zero.

Crushing Tests Using Quartz

In this investigation, the desirability of crushing a quartz gangue was recognized as being of more interest to those engaged in practical work;

unfortunately, however, none was obtainable locally, and tinguaite was employed for the experiments. It was feared that to some extent the habit of fracture of tinguaite might have prejudicially influenced the results obtained, hence upon the conclusion of the Huntington mill tinguaite tests, a series of grades of quartz sand was prepared and used to determine this point.

Convincing evidence was furnished by the results obtained that the relation between power and crushing indicated by the tinguaite tests remains unchanged when the rock crushed is quartz, excepting that the crushing force is less.

Conclusions

When it is considered that Gates, using a testing machine, obtained similar results for three entirely different rocks, a sufficient amount of experimental data now seems available to justify the following conclusions:

1. That in the reduction of any given rock there is a constant relationship between the power applied and the crushing effected.
2. That Rittinger's theory appears to conform agreeably with this relationship.
3. That Stadler's theory based on Kick's law does not so conform.

Discussion

R. B. T. Kiliani, New York, N. Y.—I wish to present some figures based upon actual observation extending over a few months time which seem to prove the author's conclusions. It is a comparison of two machines operating on the same ore, in the same plant, one taking the undersize of ½-in. screen and the other taking the undersize of 8-mesh. Each of the machines was first operated as a pebble mill, with flint pebbles, in open circuit, then with metal pebbles in open circuit and, finally with metal pebbles in closed circuit. For the first case, the percentage of oversize on 48-mesh, Tyler screen, in the discharge, varied, in the mill taking ½-in. feed, from 37.2 to 19.9 per cent. The second mill, taking the 8-mesh feed, had a variation in oversize on 48-mesh, from 36.8 per cent. to 0.5 per cent.; so it is seen there is considerable variation in these figures. I have computed the tons per kilowatt-day reduced through 48-mesh for each machine under each one of these conditions, giving 6 sets of figures. The first of these I called 1.0 and determined the ratio of the others to this. In the same way, I figured the relative mechanical efficiency according to Kick's law, using the figures given by Mr. Taggart in his paper, for ordinal numbers of the Tyler screens. I have also figured the efficiency in mesh-tons per horsepower according to Gates' method by plotting the screen analyses of feed and product and multiplying the area by a constant dependent on the scale used. The commercial efficiency of tons per kilowatt-day reduced

through 48-mesh checks very closely with the efficiency by Rittinger's law, within 1 per cent. except in one case. By Kick's law, the efficiency checks fairly closely, within about 5 per cent., with the commercial efficiency of the mill grinding ½-in. feed, but when we start to grind a finer, 8-mesh material, and especially grinding very fine, with a greater percentage of −200 mesh material produced, there is a big discrepancy. The ratios obtained may be tabulated as follows:

Relative Efficiencies

	Open Circuit		Open Circuit		Closed Circuit	
	Flint Pebbles		Metal Pebbles		Metal Pebbles	
Feed size	− ½ in.	−8-mesh	− ½ in.	−8-mesh	−½ in.	−8-mesh
Commercial	1.00	0.81	1.07	0.76	1.26	0.79
Rittinger	1.00	0.67	1.06	0.77	1.22	0.79
Kick	1.00	0.42	1.03	0.44	1.13	0.45

J. W. Bell (communication to the Secretary*).—Mr. Kiliani's results indicate very interestingly the inherent defect in Kick's law demonstrated by our own experiments that the Kick or Stadler efficiency figure is influenced far more by the size of the feed to a crushing machine than by its actual performance. Mr. Kiliani, of course, appreciates that the number of tons reduced to −48-mesh is a very rough measure of the amount of crushing performed since obviously a large variation in the percentage of the finer sizes would be possible without conflicting with this specification. He has used this method merely for the sake of the comparison of the several results.

I mention this only because if the conclusions based on experiments performed at Purdue University and McGill University are correct, as I am confident they are, a method for measuring the efficiency of rock-crushing machines is at hand, which is at once simple, dependable and practical.

C. W. Merrill, San Francisco, Cal. (communication to the Secretary†).—I have read with the greatest interest Mr. Bell's paper and cannot compliment it too highly for the scientific method by which he has arrived at his deductions. It is my opinion that the paper will be a classic for some time to come, and I can only hope that it will stimulate further pursuit of the subject in detail in the field of fine crushing. Particularly would it be of value to the profession if, for instance, the

* Received Mar. 6, 1917.
† Received Feb. 20, 1917.

ordinary tube mill, the Marathon mill and the Hardinge mill could be compared by the methods used by Mr. Bell.

A. O. Gates, Salt Lake City, Utah (communication to the Secretary*).—The writer is delighted by the results shown in Mr. Bell's paper, which prove in an experimental way different from that followed by the writer[1] that "the work done in crushing is proportional to the surface exposed by the operation," in accordance with Rittinger's Law.

Mr. Bell's work was all done in commercial types of crushing machines, while my work was done in a testing machine; his work represents fully one hundred times the expenditure and time that my work represented; a great number of tests were made by his organization to determine single points, while I was obliged to depend on the general trend of a curve for results; we agree absolutely that "Kick's law," for which Mr. Stadler was sponsor, does not apply to rock crushing. While my work indicated the reliability of Rittinger's theory, Mr. Bell's work clearly establishes the reliability and makes it Rittinger's law.

While Mr. Bell clearly recognizes the variation in the value of the minus 200-mesh product, I hope his paper will not set a precedent in the adoption of 780 as the average reciprocal of diameter of the minus 200-mesh. I indicate, on page 900 of my paper, a method of approximating this value, but I hesitate to set any value to it. If the curve of "per cent.-through" plotted against "reciprocals of diameters" follows the law of the hyperbola there will be as much, or more, surface in the finest 1 per cent. as there is in the next finest 9 per cent., and as much in the finest 0.1 per cent. as in the rest of the finest 1 per cent., and so on.

Probably the safest way to make comparisons is to neglect something like the finest 1 per cent. from all calculation, and determine the surface in the minus 200-mesh and plus the finest 1 per cent. by assuming that the 200-mesh points lie on the straight line of the logarithmic plot shown on page 900 of my paper.

With regard to choke crushing: Mr. Bell in his 1913 tests, as shown in Table 2, obtains much higher results by small reductions and by feeding singly to avoid choke crushing, while in section 31 of my paper I indicate the same thing. But I wonder if these are not *apparent* higher results, for the reason that in "choke crushing" there is probably an excess of unmeasured surface in the minus 200-mesh and minus 2,000-mesh to account for the power consumed. Probably there is no economic value in the minus 2,000-mesh except in Portland cement; the practical man may neglect it, but theorists must take it into consideration.

Mr. Bell is not convinced that the work could be carried on to better advantage in a testing machine such as I used, than in the commercial

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* Received April 4, 1917.

[1] *Trans.* (1915), **52**, 875.

machines that he used. Of course, the best way for him to learn that it could, would be to try it. My first "testing machine" was a 22-lb. stamp. The idea of the 22 lb. was that when dropped 1 ft. 1,500 times, 1 hp. was expended. I used this in 1909 long enough to prove that the "efficiency" of certain rolls was about 25 per cent. How much of the horsepower was expended in vibrating the kitchen floor was not recorded.

I did consider a small set of rolls for the work I wished to do in determining the crushing constant of rocks, but when I went to Purdue shortly thereafter, I cast about for a suitable machine in the University laboratories. The ordinary power-driven Riehle and Olsen machines I rejected because of the trouble in getting power and in controlling it when I did get it. We had a drop testing machine with a very heavy weight, with a drum on which the rebound could be measured so that the actual energy spent could be readily calculated, but before adopting this device I came upon the Amsler-Laffon machine which so admirably served my purposes.

Such testing machines must be in the laboratories of many of our universities and cement plants. Washington University has one at St. Louis and the Portland Cement Co. of Utah has one, both being of American make. There was no difficulty in measuring the energy absorbed; in fact, my machine had an automatic recorder. My machine was just about 100 per cent. efficient, for I had actual measurement of the pressures exerted between the dies and the actual distance the dies were moving measured right at the point of crushing. I believe my method eliminated all friction losses, and therein lay its scientific advantage.

I would like to see some of our Schools of Mines put into their mining laboratory courses such tests as I made. These tests could be made and worked up in two afternoons. They would perhaps make the theory of crushing plainer to the student and would stimulate more research along this rather important line of milling. Outside of the machine, the expense would be small, and with the right supervision, valuable data could be accumulated.

Now that we have what I believe is satisfactory proof to most engineers of the manner in which energy is distributed in crushing, the next step is to make further practical use of it. Constants should be determined for the various rocks that are crushed. This should be done by the U. S. Bureau of Mines. Institute Committees should consider the standardization of the measurement of crushing—whether we shall speak of it in terms of "surface units" or "mesh-tons," and if the latter, whether we shall use the Tyler mesh number, or the corresponding reciprocal-of-diameter.

I congratulate Professors Porter and Bell and their associates on the success they have achieved in this investigation, and the milling fraternity upon having one important point absolutely settled.

John W. Bell (communication to the Secretary*).—I have to thank Mr. Merrill not only for his very kind remarks, but for the weight of his implied opinion that the work accomplished probably has an important practical aspect. I think there is a distinct although unuttered impression among engineers, that it will be time enough for them to begin to take an interest in the law of crushing when it emerges from the cloud of violent and even vituperative controversy that has hovered over it during the last 6 or 7 years. I have a very confident feeling that as the result of the work done by Mr. Gates, by me and by Messrs. Mitchell, Baily and Cockfield, who so loyally helped me to obtain the necessary data, at least the practical law of crushing has emerged; that it is ready and available for use by our fellow engineers, and that it will help them to unravel a number of the many practical problems they are constantly confronted with.

In regard to Mr. Gates' discussion, I would indeed like to believe that McGill could claim, as a minor achievement, the credit which Mr. Gates so generously bestows of having established the "law" of crushing.

I think that more work would be required to establish this claim in the strict sense of the word, as there is a fairly large gap between the coarse and fine crushing results. I am not worried about that gap, because the reason for it is quite clear to me; at the same time I think it will be well to bridge it by a series of tests, and I am bound to admit that the testing machine is the most promising machine right at hand to work with, although whether the minute amounts of power required to crush small quantities of rock can be so accurately integrated as to insure high accuracy in the final result is still a question in my mind. If we had an Amsler-Laffon machine I would certainly try it.

Mr. Gates' remarks about choke crushing are extremely interesting. He compels me to wonder, too. It is a new and probably a very important factor to be considered in this connection.

I am in perfect accord with Mr. Gates about the desirability of the determination of crushing constants, because when this is accomplished the efficiencies (not relative mechanical, but real efficiencies) of different types of rock-crushing plants in any part of the world will then be comparable. For this reason it would seem desirable that the American Institute Committee should not only consider ways and means for accomplishing this (and the method adopted is immaterial provided it is the best method), but that they should take the matter up with some of the other large societies in and outside of America, to the end of adopting a standard method for determining rock-crushing constants, the efficiency of a crusher, and a standard series of screens.

* Received Apr. 30, 1917.

Countercurrent Decantation*

BY LUTHER B. EAMES,† E. M., TIMMINS, ONT., CANADA

(New York Meeting, February, 1917)

The recovery of dissolved gold from slime pulp in the cyanide process was first accomplished by intermittent decantation. This simple process consists in mixing with the pulp containing the values in solution, a solution of lower gold content, settling the mixture in a tank and decanting the clear supernatant fluid. The thick pulp remaining in the tank is pumped to a second tank together with more barren solution and again settled and decanted. After several repetitions of this operation, values are so far reduced that further washing is not profitable. The gold recovery of this process is high, but the plant required is bulky, labor cost is high and the amount of solution to be precipitated is excessive.

Historical Data

As early as 1901, a plant was built in the Black Hills of South Dakota by John Randall, employing the same principles but attempting to make the process continuous by substituting for flat-bottomed tanks, cones which operated continuously, receiving a constant feed and discharging a steady stream of thickened pulp. These cones were operated in series, the thick underflow of the first one forming, with a stream of diluting solution, the feed to the second cone of the series. Barren solution was added to the tank immediately preceding the discharge tank and, after being slightly enriched by the low-grade pulp in this tank, overflowed to form a diluting solution again for the richer feed entering the third tank from the end of the series, and so on back to the richest tank of the series. Clear water was used for the wash in the final tank. This is the principle on which all successful countercurrent decantation plants operate at the present time, but Randall's plant was not successful because of mechanical difficulties in getting a continuous thick discharge from his cone tanks. A similar plant was built in South Africa although there the washes were not repeatedly used, as in Randall's case, but were precipitated after each contact with the ore. This also was abandoned

* Also published at this time in the *Bulletin* of the Canadian Mining Institute.

† Mill Superintendent, Hollinger Consolidated Gold Mines, Ltd.

because of mechanical difficulties and the cost of precipitating the large quantities of solution that had to be used. For a number of years the process was not used, and it was not until the introduction of the Dorr thickener that the minds of metallurgists began to turn again to the continuous decantation principle.

In 1910, two decantation plants were built making use of flow sheets similar to that used by Randall nine years before, but substituting Dorr thickeners for the cones. One of these was at Mocorito in Sinaloa, Mexico, and was installed under the direction of C. Dupre Smith, while the other was designed by J. V. N. Dorr, assisted by the writer, for the Vulture Mines Co. of Wickenburg, Ariz. While perhaps not perfect at first, both of these pioneer plants were so successful as to encourage further installations, few and scattering at first but in considerable numbers during the past three years.

Theoretical Considerations

In view of this increasing importance, the following discussion of the principles and characteristics of the process is offered. For the purpose

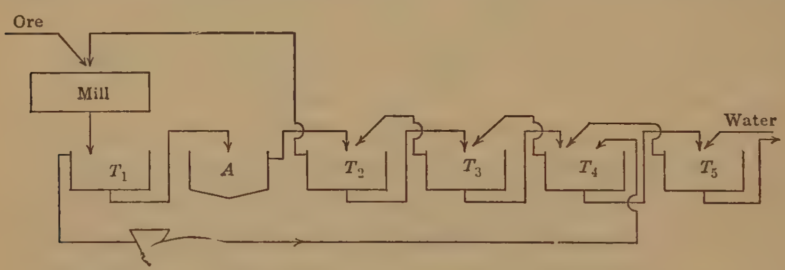

Fig. 1.—Typical Flow Sheet of Countercurrent System.

of investigation a simple yet typical flow sheet has been selected. This is shown in Fig. 1.

This flow sheet assumes that crushing is done in solution, the overflow from the tank T_2 being used for the crushing solution. This crushing solution leaves the grinding circuit with the ground pulp and enters T_1, and that part which does not pass to the agitators with the pulp overflows T_1 and goes to precipitation. After depositing its gold contents, it is used to dilute the underflow of T_3 as it enters T_4. The overflow of T_5 is also mixed into the feed to T_4. The overflow of T_4 mixes with the underflow of T_2 to form the feed to T_3, and so forth, as indicated in the flow sheet. At each succeeding mixture the solution meets a pulp of higher dissolved content than itself and is enriched while the pulp is correspondingly impoverished. The pulp at each step approaches the discharge end of the mill while the solution goes to the feed end—hence countercurrent decantation.

Variables Affecting Decantation Process

The principal variables that may affect the efficiency of the process are:

1. Grade of ore.
2. Ratio of solution precipitated to ore treated.
3. Thickness at which pulp can be discharged.
4. Cost of chemicals.
5. Rapidity of dissolving, and the place in the circuit where it takes place.
6. Efficiency of precipitation.

Since the decantation process is one involving volumes and dilutions, it is possible to calculate accurately what distribution of values should take place under any given set of conditions. As far as possible, each one of the above variables has been mathematically considered independently of the rest and the results have been plotted.

The effect of variations in the grade of ore is shown in Fig. 2 and scarcely needs comment. In practice, of course, no such increase as is shown in the gold-solution curve would ever be allowed to take place on account of the difficulty of precipitating such high-grade solution and the danger of leakage, but the graph shows conclusively that all solutions increase in value in direct proportion to the increase in the grade of the ore and that the higher-grade solutions increase at a much more rapid rate than the final washes.

The important part played by the ratio of solution precipitated to ore milled is shown in Fig. 3. For the particular grade of ore considered, which in this case was $10 recoverable per ton, with precipitation to 3 c., the economic ratio may be roughly determined by inspection of the lowest curve, which represents the value of the solution leaving the last tank of the series with the tailing. It will be noted that the loss in gold increases very fast as the amount of solution precipitated is decreased, while after a certain point the increased recovery due to increasing the volume precipitated is very slight.

The final choice of the exact ratio to be used must be influenced by the cost of precipitation, which is mainly the cost of zinc. To show clearly the effect of precipitation cost on the economic precipitation ratio, Fig. 4 was plotted. I have considered it safe to assume that the cost of increasing the amount precipitated is due to the additional zinc used. The loss in dissolved gold and the cost of the zinc used in precipitation are both plotted to the same scale in cents per ton of ore. By adding the ordinates of these two curves a third is formed which represents the total loss in gold and zinc. The lowest part of this curve represents the economical range and indicates that for a $10 ore, under the conditions assumed with zinc at 26 c., from 200 to 250 tons should be precipitated

per 100 tons of ore treated. In dotted lines are shown the corresponding curves for a $5 ore under similar conditions. In this case the range is from 150 to 200 tons. While there is a considerable range within which practically identical results may be expected, it is apparent that each operator should figure out for his own conditions just what his range is.

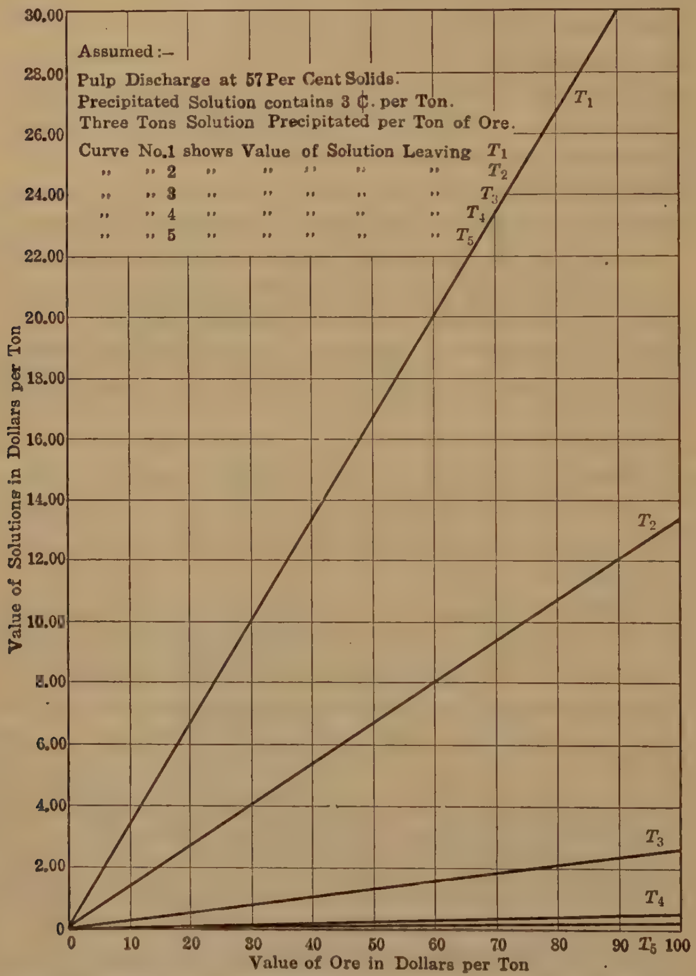

FIG. 2.—EFFECT OF VARIATIONS IN GRADE OF ORE TREATED.

The next, and one of the most important variables to be considered, is the thickness, or percentage of solids, to which pulp can be settled. In Fig. 5 the values of the various overflows have been plotted so as to show the effect of variations in the moisture in the underflows. The full-line curves represent the values of the solutions—that is, they are shown as per ton of solution—but in calculating losses per ton of ore, the ratio

of the solution to the ore present must be considered. The loss in dissolved gold per ton of dry ore for the last tank has, therefore, been plotted in a separate curve, and it is this curve which should be given the most serious attention in determining the suitability of an ore for decantation. The moisture in the pulp also has a direct bearing on the cyanide loss,

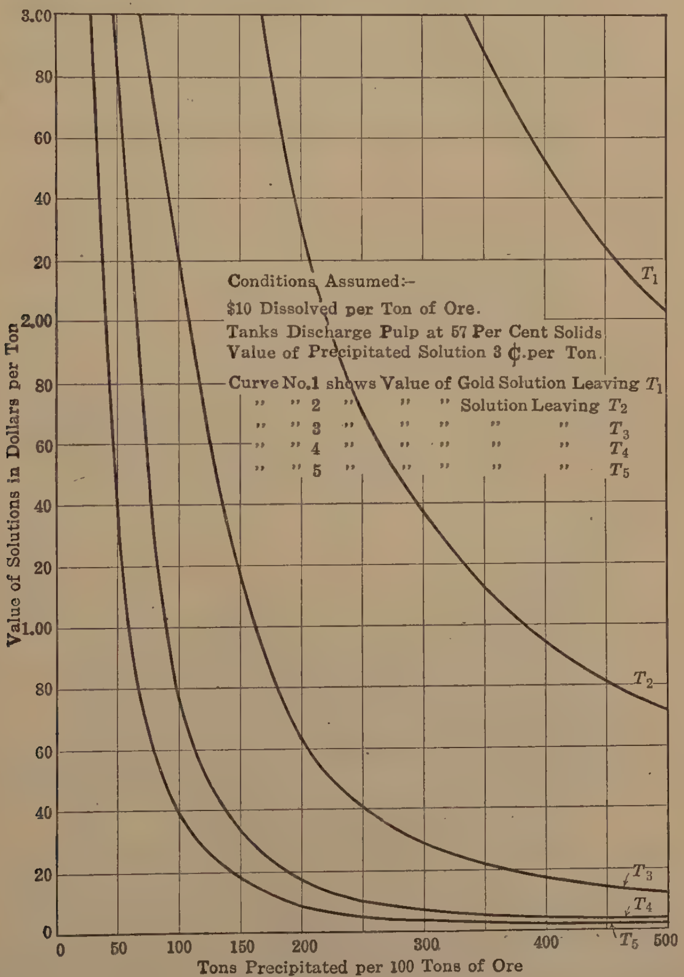

FIG. 3.—EFFECT OF VARIATION IN THE RATIO OF SOLUTION PRECIPITATED TO ORE TREATED.

which is also shown in Fig. 5. This has been shown in pounds rather than in cents because solution strength and the price of cyanide both affect its value. Enough is shown, however, to make it plain that there is a very decided limit to the density of the pulp that can be handled economically, and one is forcibly reminded that the cyanide strength

should be kept as low as possible. Operators as a rule seem to be inclined to "play the game safe" as regards solution strength and it is probable that in many cases cyanide could be saved without any considerable loss in gold by using solutions of a lower strength.

In making the calculations upon which the foregoing curves are based, it was assumed that 75 per cent. of the gold was dissolved in the grinding department and the remaining 25 per cent. in the agitators. This is, of course, an approximation which cannot be accurate, and even with the cleanest ores some gold is dissolved in passing through the tanks following the agitators. This is a condition that has always been met, no matter

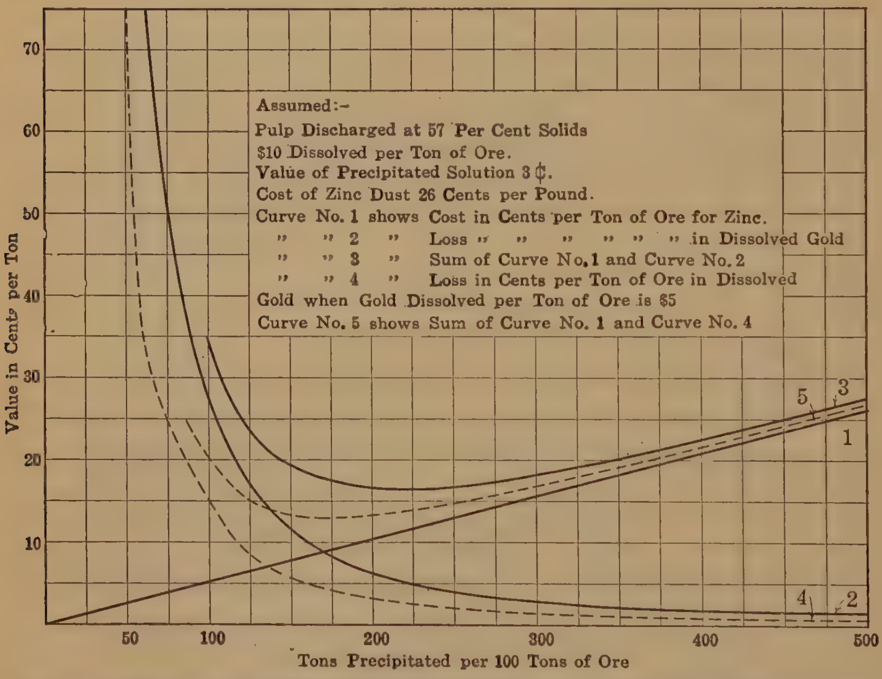

FIG. 4.—METHOD OF DETERMINING THE PROPER RATIO OF SOLUTION PRECIPITATED TO ORE TREATED.

what the method of recovering the dissolved value. Changing solution during agitation has been practiced for years in the treatment of silver ores, and in the treatment of concentrates at Goldfield with a view to reducing this lag of dissolving. I have also found that on Goldfield ore, pulp from the final agitators, reagitated without change or addition of solution, would give up no more gold, while reagitated filter tails from the same ore would show a distinct reduction. The same condition exists in the pulp fed to decantation plants and there is little doubt that some dissolving takes place even in the last tanks of the series. Some of this value is lost, particularly that freed near the final discharge

but much is recovered that would probably not be won by any other method. The superintendent of one of the decantation plants treating Tonopah ore informed me that his sampling indicated that the additional dissolving taking place in his plant was sufficient to offset his entire dissolved loss and 3 c. per ton additional.

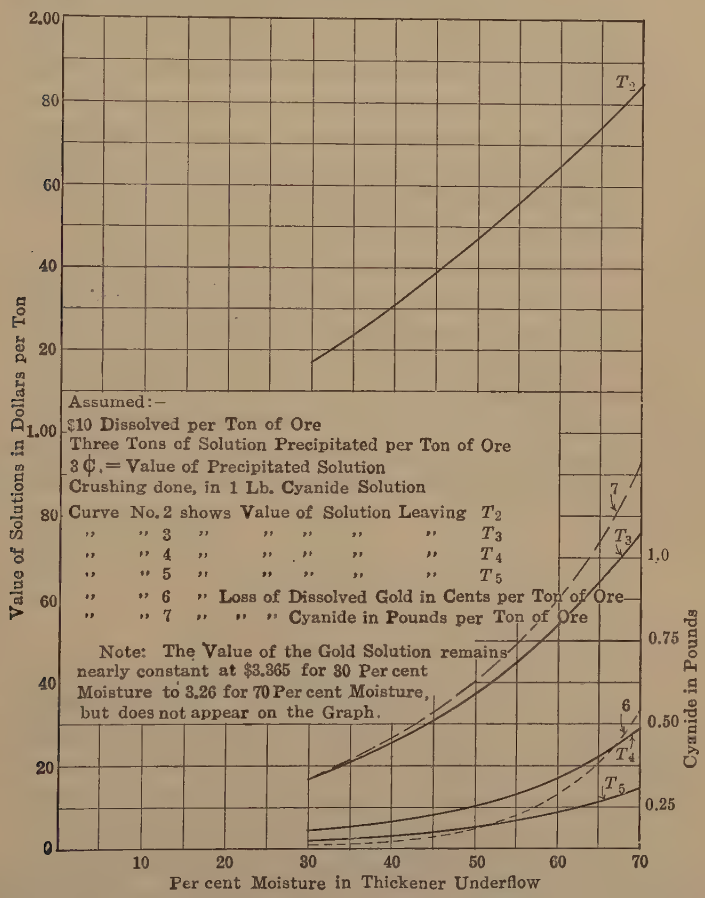

FIG. 5.—EFFECT OF VARIATIONS IN THE PERCENTAGE OF MOISTURE IN THE THICKENER UNDERFLOW.

Any considerable dissolving during decantation will be indicated by a difference in the assay value of the solution in the underflow of the tanks as compared with the overflowing solution. In practice there is always more gold per ton in the underflow solution than in the overflow of any given tank, but in the ores of Porcupine district this difference is very small. Other causes may, and no doubt do, tend to produce this difference between the overflow and the underflowing solution. Adsorption is probably the most important and perhaps the least understood of these. In the case of the ores of the Porcupine district this phenomenon

is of small importance, as the ore is composed of crystalline schists and quartz and there is little tendency for the ore to flocculate under the influence of the solutions used. The gold and silver ores of the Western States are in many cases in eruptive rocks; these ores usually flocculate in solution and in doing so seem to entrap a portion of the value in the solution. At any rate there is a much more noticeable difference in the assays of tank effluents in the treatment of these ores. Faulty mixing of the products fed to the tank has in some cases been blamed for this. In most plants riffles and other devices are put in launders to insure thoroughness in mixing, but our experience in Porcupine would indicate that the ordinary launder makes a perfect mixture. Adsorption, then, would seem to be the more important cause of these differences, and should be a profitable field for further investigation.

Proper precipitation is essential in decantation, as it has always been in every other process, and, as shown by Fig. 6, dissolved gold is lost in proportion to the amount of value in the barren solution used. It will be observed, however, that the loss does not increase as fast as the value of the barren solution does.

It will be observed that the value of the solution leaving T_4 increases at nearly the same rate as the barren solution and that all the preceding overflows increase by exactly the same amount as T_4. The water dilution cuts the final loss down to a slower increase than the barren solution itself.

Mechanical Features

Dorr Thickener

The mechanics of the decantation system is of the simplest, but is worthy of study none the less. The Dorr thickener is universally used for the separation and is so well known and so standardized that it need only be mentioned in passing. It has been fully described by Mr. Dorr himself in a paper read before the Institute.[1]

The arrangement of the tanks in plan may be largely governed by space available and other local conditions. In elevation it is very convenient to have the last tank the highest and the preceding ones successively lower, so that the overflowing solutions may be transferred by gravity with a minimum of attention. Where such an arrangement is not possible, the tanks may all be placed on the same level using automatically regulated air lifts to transfer the solutions from tank to tank.

Diaphragm Pumps

As a means of transferring pulp, the diaphragm pump has distanced all competitors. The main reasons for this are that it not only pumps

[1] The Dorr Hydrometallurgical Apparatus, *Trans.* (1914), **49**, 211–237.

but at the same time measures the volume of pulp transferred. The flow of pulp is not easily obstructed by foreign matter, such as chips, waste and

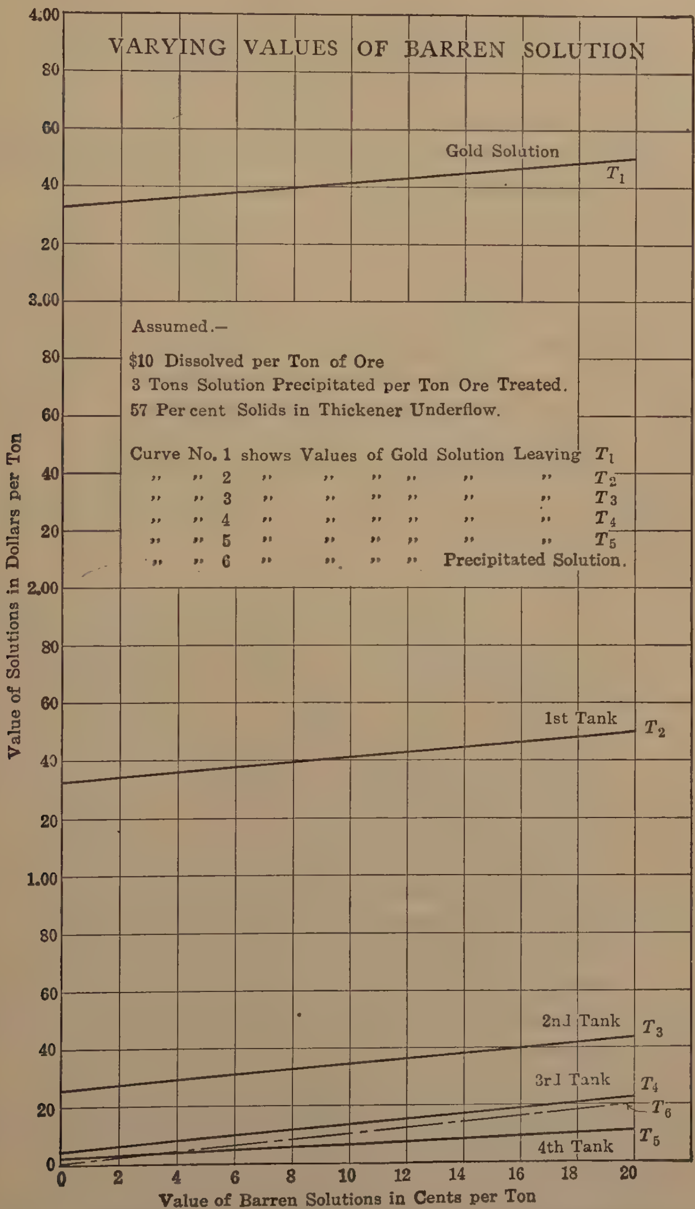

Fig. 6.—Effect of Variations in the Value of Barren Solution.

the like, since the openings are full pipe size straight through the pump. The capacity of the pump can be controlled with certainty by means of

cone pulleys. The attendant's duties are all on one floor, as it is operated from the overflow level instead of at the bottom of the tank. Operating cost is very low on a properly designed pump.

Air lifts were used for a time, owing to their cheapness and simplicity, but they are hard to regulate and are wasteful of power. The most careful watching will not prevent them from "running away," that is, transferring too fast, consequently thinning the pulp and sending large quantities of rich underflow solution toward the discharge. On the other hand, if they are set too slow the gradual thickening of the discharge slows up the flow and finally stops it.

Spigot discharge to a bucket elevator or centrifugal-pump sump has also been tried, pulp and solution being mixed and elevated together. There are several objections to this arrangement, the principal one being high power and maintenance cost for the pumping or elevating machinery. Both the solution and the pulp must be lifted more than the full height of the tank for each decantation. The small high-velocity stream of pulp passing through a spigot at relatively high pressure is subject to frequent stoppages due to foreign matter, and even when this does not cause a complete plugging it interferes with uniformity in operation. There is also the objection that work is done on two floors, one below and one above or near the top of the tanks.

There has been a certain amount of prejudice against diaphragm pumps due to the fact that some of the earlier pumps were poorly designed for the work they had to do. Faulty valves were responsible for much of this. Poor methods of regulation also had their effect.

The practice in most plants in the Porcupine district now is to use cone pulleys for the regulation of capacity, although some of the operators favor regulation by varying the length of stroke.

The valves should be of the floating type, as any hinge device will catch the wood chips that are present in the best-screened pulp. The chips lodged in the hinge of the valve cause leaks which, though small in amount at first, cause cutting of the seat and consequently permanent leakage. With the floating valve there is no place for chips to lodge and the whole circumference of the seat is washed by pulp at every stroke. This type of valve also has the advantage that the lower valve may be placed directly below the upper one and made small enough to be lifted out through the upper-valve seat when the upper valve has been removed. No tools are required for the removal of these valves and it is a simple matter to inspect them.

The best results have been obtained when the working surface of both the valve and seat were of high-grade rubber. Belting was used at first, but it was found that minute leaks were almost sure to start, due to the fact that belting is not yielding enough to close over any chip that may lodge on the valve seat, and that a leak once started would ruin both

valve and seat in the course of a few days. On the other hand, valves of rubber seating on rubber have operated 6 months without the slightest decrease in efficiency and with scarcely perceptible wear.

Diaphragm pumps have been operated at speeds varying from 15 to 100, the higher speeds usually in conjunction with a short stroke.

The practice at the Hollinger mill has been to use a low speed and a stroke as long as the diaphragm could safely stand. Measurements taken on the Hollinger pumps equipped with standard No. 4 Gould diaphragm at 3-in. stroke gave results as shown in the following table:

Number of Strokes	Volume per Stroke, Cubic Feet	Specific Gravity	Per Cent. Solids
14.5	0.139	1.54	54.5
23.0	0.148	1.48	50.5

Tons Solids Pumped per day	Per Cent. Increase in Speed	Per Cent. Increase in Volume per Stroke	Per Cent. Increase in Tonnage
76.3			
114.5	58.5	6.5	50

From the above figures, which are typical of numbers of tests made, it may be inferred that the volume pumped is roughly proportional to the speed of the pump but that leakage is slightly greater on the lower speeds.

The low speeds and the placing of the discharge lips high enough above the discharge valve to leave 3 or 4 in. of pulp over the valve at the end of the upstroke have rendered the pumps of the Porcupine district practically free from the splash and dirt that have been one of the chief objections to diaphragm pumps in the past.

Where tanks have a settling capacity of over 125 tons of solids per day, it may be found advisable to use two diaphragms in parallel, making a duplex or even a triplex pump. This arrangement has several advantages; with the lowered speed, strains of the pump are lessened and distributed, and repairs can be made on one unit without complete stoppage of the tank discharge. Diaphragm life appears to be roughly proportioned to the number of strokes, so that an increase in the number of diaphragms employed, with a corresponding decrease in the strokes per diaphragm does not result in an increased cost for diaphragms.

As a safeguard against waste and other foreign matter it is advisable to screen the pulp before it goes to the decantation tanks, and where small wood chips are to be expected a fairly fine screen, usually of punched plate, will be of great service in protecting thickeners and pumps.

Measurement of Solution Precipitated

Every cyanide plant has some method of determining the amount of solution precipitated, and in decantation plants having only one series

of tanks the entire tonnage of precipitated solution is used at one place. If, however, more than one series of tanks is used it becomes important to split this precipitated solution into parts proportional to the tonnage of ore being washed in each series. This can not be done by regulating the valve at the outlet of the barren line, as trial has shown that under some conditions an increase of as much as 50 per cent. can be made in the amount flowing from a pipe line without any visible change in the stream.

This difficulty has been overcome by the use of V-notch weir boxes, one for each series of tanks. In each weir box is a float compartment and a float operating an indicator on a scale which reads in tons per 24 hr. While the weir box, which can be readily made at any mine, is not a recording instrument but gives only an instantaneous rate reading, its use greatly simplifies the problem of proper distribution of barren solution. Water also should be added in proper amounts and uniformly distributed. A smaller weir box has been found convenient for this purpose.

It is usual to determine the specific gravities of various pump discharges about a decantation plant at least once a shift. In a small plant this consumes little time but if there is a large number of measurements to be made the distance to be covered by the operator is considerable, especially if he has to return to a central point for each weighing. In such plants, a spring balance with a dial and revolving hand, that can be carried about the plant, is a good type. I have used a milk scale having an adjustable tare-indicating hand, which makes one complete revolution for 10 lb. I use a narrow-necked can which holds just 10 lb. of water when level full. This makes it possible to add a paper dial which can easily be divided so as to read directly specific gravities. The large sample makes possible accurate readings and enables the operator to determine gravity at the place where the sample is taken.

Tray Thickeners

In his paper on the Dorr metallurgical apparatus, Mr. Dorr touched on the future of tray thickeners, and spoke of the possibility of a complete decantation plant being installed in one tank. While this has not been done as yet, a four-step decantation followed by a continuous filter is in operation, the four steps of the decantation being completed in two tanks equipped with single trays. While no figures have been made public the results are said to be creditable.

Another use of the tray in decantation, which is shortly to be tried at the Hollinger mill, is to increase the capacity of existing plant without increasing the number of tanks installed. Since the same grade of solution is handled in both tank and tray there is no danger of any

mixing of solutions of different grades. This plan should decrease the cost of buildings, tank foundations and building site in almost direct proportion to the increase in capacity gained, while tank cost, power and labor should all be decreased to a marked degree. The outcome of this experiment should have a direct bearing on the future of the decantation process. It must, however, be borne in mind in this connection that some slimes require time for their final thickening and consequently necessitate a tank of some depth, while other slimes find their capacity limit in the thin-pulp settling rate. For this latter class, as explained by Messrs. Coe and Clevenger in their paper on slime settling,[2] a shallow tank or tray is sufficient while the former class requires a tank of carefully calculated depth to give the required time for thickening.

The Decantation Plant of The Hollinger Co.

The Hollinger decantation plant consists at present of five rows of 40-ft. tanks, four tanks to a row, forming a plant of five units. The tanks are arranged with a difference in elevation of 2 ft. 6 in. between steps with the final tanks of the series the highest, so that all solutions gravitate through and out of the plant to precipitation. The Barrett specification roof is supported on flat trusses, the lower chords of which pass just above the tank rims. These trusses also serve to support the thickener mechanisms and the walks between the tanks.

The diaphragm pumps used were designed by the company's staff, and have been very reliable and economical. They are all three-throw or triplex pumps so that in spite of the large tonnage handled the duty on each diaphragm is light. It is not uncommon for diaphragms to last 300 days while the life of the present type of valves and seats has yet to be determined.

The pumps are used not only for pulp transferral, but also for the final discharge. This makes regulation of the final discharge for moisture much easier, more reliable, keeps the work of the operator all on the upper floor and allows the tailing to be discharged at a considerably greater elevation than would otherwise be the case.

The barren solution and water wash added to each row are measured by separate float-reading weir boxes assuring uniform results from the various units.

Labor

The plant is operated by one man per shift who oils all machinery, watches and adjusts the pumps and records their performance. The

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[2] Methods for Determining the Capacities of Slime-Settling Tanks, *Trans.* (1916), **55**, 356.

solution man makes titrations and regulates the addition of water solution but has no other duties in the decantation plant. A repair man on day shift makes all repairs and has time for other work.

Power

The power for each tank including motor and line-shaft losses is under 1 hp., while each three-throw pump consumes about the same amount.[3]

Cost

The costs for the 12 weeks from Jan. 28 to Apr. 21, 1916, have been taken as typical of what is done by this plant at its present capacity. During this time 85,854 tons were decanted at a cost of $599 for supplies, including power, and $1,194 for labor, or $0.007 per ton for power and supplies, and $0.0139 for labor, making a total of $0.0209 per ton for decantation. Labor is no doubt higher here than it will be in the future, as a greatly increased tonnage is to be treated while supplies and power should remain nearly the same. The cost as it stands is about 40 per cent. of the cost of filtering on leaf filters at about the same daily tonnage.

Extraction and Recovery

In the ores of the Porcupine district the recovery by dilution seems to be almost the theoretical maximum. Adsorption does not seem to have any appreciable effect. There is a slight dissolving during decantation which, while it adds to the recovery, makes the soluble loss somewhat greater than it would otherwise be.

The figures quoted below on chemical consumption and recovery refer to only two units of the Hollinger plant. The figures on these units are given because the other units of the mill share their feed with the original Moore filter plant, and likewise their barren solution, while for commercial reasons the two units in question have been given a separate solution system and separate precipitation presses. These two units are therefore the only ones upon which all the figures are available.

In comparing the results quoted, however, it should be borne in mind that the flow sheet has been modified in this plant somewhat, because of limitations of space, so that the overflow of T_2 instead of that of T_1 goes to precipitation. The effect of this is to raise the theoretical value of the overflow of the last tank 3 c. at 3 to 1 precipitation.

A statement of results follows:

Period covered, same as that for which costs were given—from Jan. 28, 1916, to Apr. 21, 1916.

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[3] More recent measurements show that 20 tanks, of which four contain trays, consume 9.2 hp. motor input; while 24 three-throw pumps take 11.1 hp., or a total of 20.3 for the plant.

Tons of ore treated	38,885
Value per ton of ore treated	$8.92
Ratio of ore to solution precipitated	100 to 285
Tons solution precipitated	110,604
Strength of cyanide used	0.9 lb. per ton, or 0.0045 per cent.
Cyanide added per ton of ore	0.46 lb.
Difference between pulp feed and pulp discharge for first tank after agitators	25 c.
Average moisture in tails	45 per cent.
Average value of barren solution	3.2 c.
Dissolved gold per ton of solution discharged	11.71 c.
Dissolved gold per ton of ore discharged	9.57 c.

It is theoretically possible, taking into consideration the flow sheet, the grade of ore treated, the barren solution used and the thickness of pulp attained, to have reduced the overflow of the last tank to 7.6 c. leaving a difference of 4.1 c. to be accounted for by continued dissolving, adsorption, etc.

Viewed in one way it may be said that actual losses are 54 per cent. higher than theoretical, but where one is dealing with samples so easily affected by faulty manipulation and where any error except losses in assaying tends to raise the results, a check to 4 c. does not seem bad. The average loss would have been somewhat less if the occasional high results had been omitted, but this was not done.

From the foregoing, I believe one is warranted in concluding that a reasonably accurate forecast can be made of the results to be expected from a decantation plant and that these results may compare very favorably with the results obtained from filter plants.

In conclusion 1 would say that I am indebted to P. A. Robbins, Managing Director of the Hollinger company, for permission to quote results from the Hollinger plant.

Discussion

J. V. N. Dorr, New York, N. Y.—I have read Mr. Eames' excellent paper on this subject with great interest, for besides being connected with the design and installation of the first modern countercurrent decantation plant, he was associated with me in the development of the thickener in the Black Hills some years earlier, and has now been operating the largest plant that has used this method of treatment. His discussion of the influence of different factors on the results obtained is quite thorough, but several features I think may not be apparent to the reader who has not studied the subject carefully.

1. *Grade of Ore That Can Be Treated.*—Mr. Eames has shown graphically that the assay of all solutions increases directly as the grade of the ore treated. He has not emphasized, however, the fact that, as most operators prefer to keep the assay of pregnant solutions down below a certain

figure in order to insure complete precipitation, an increase in ore value beyond a certain point means a proportional increase in solution precipitated and therefore not only an increase in the percentage of dissolved gold recovered, but an actual descrease in tailing loss.

The importance of the influence of the percentage of moisture in the underflow on loss is made clear, and I may add that unless it is quite evident that pulp can be safely settled to 50 per cent. solids, or more, it is usually advisable to use a continuous filter at the end of a decantation plant to reduce losses in cyanide and gold.

Dissolution in Thickeners.—The influence of a change of solution on silver dissolution has long been emphasized and most gold operators have recognized the annoying way in which a gold ore after being agitated until nothing more can be dissolved will loosen up again when almost ready for the dump after the dissolved metal has been removed by decantation or filtration. Continuous decantation gives favorable conditions for this additional dissolution and a chance to recover most of what is thus dissolved. This is especially true if an extra tank is used in the series and barren solution added at the second tank from the end instead of the first, as shown in Mr. Eames' Fig. 1.

It can be calculated that the use of this tank will reduce the mechanical loss of cyanide by one-third and cut the gold loss as well; but the saving in gold or silver that may dissolve when the pulp is first diluted with weak solution may be as important.

At one plant in the Southwest, where one of the agitators was not in use as additional dissolution would not pay the cost of its operation, the company found that 30 c. more was being dissolved in the decantation thickeners.

It has been pointed out that decantation alone should only be applied to suitable ores, but it is becoming well recognized that when filters are used one or more steps of countercurrent decantation should precede them, as the additional dissolution will more than pay operating expenses; also, the lower-grade solution going to the filter means lower filter costs and tailing losses.

The added cost of operating an additional thickener on the average ore is so low as to be difficult to estimate and may safely be placed at less than 1 c. per ton, including reasonable amortization.

Trays.—Mr. Eames has referred to the use of trays in decantation plants. I may say that one self-supporting tray giving double capacity has been in operation for some time in a 50-ft. tank and one for a 75-ft. tank has been designed.

Tanks with several trays are also in operation, so that the problem, especially apparent in cold climates, of large mill buildings for a decantation plant, is being rapidly solved.

Anthracite Stripping

BY J. B. WARRINER,* LANSFORD, PA.

(New York Meeting, February, 1917)

Introduction

STRIPPING is the name given to the process of removing clay, rock, or other cover from deposits of coal or ore. In this paper it is intended to cover the methods used in carrying on this operation in the anthracite coal regions of Pennsylvania.

In doing this, attention will be chiefly devoted to those methods wherein practice in different sections of the region differs, or is susceptible to change or improvement, though it is intended to give as complete data as possible concerning all stages of the operation.

The data from which this paper has been prepared have been collected and submitted by a committee† representing the larger users of stripping methods, of which committee the writer has acted as chairman. Data and assistance have also been obtained from various mining men, engineers, contractors and manufacturers of stripping equipment, to all of whom our appreciation is due.

Early History and Statistics

The earliest mining operation on a commercial scale in the anthracite region was a stripping operation. This was the famous Quarry mine at Summit Hill, where this method was used only because no other method of mining a deposit of the nature found there seemed feasible to the early operators. A description of this stripping from a report published in 1821 is as follows:

> "The coal mine at present worked by the company lies on the top of a mountain, and appears to extend over some hundred acres of land covered by about 12 ft. of loose black dirt resembling moist gunpowder, which can be *removed by cattle with scrapers*, and thrown into the valley below so as never to impede the work. The thickness of the coal is not known, but a shaft has been sunk in it 35 ft. without penetrating through. More than an acre of mine has been uncovered and presents a huge rock of coal, which is easily quarried without blasting."

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* Chief Engineer, Lehigh Coal and Navigation Co.

† The members of the committee that furnishes the data used in preparation of this paper were Messrs. Walter Fahringer, Arthur Lewis; H. J. Heffner, J. G. Troutman, C. F. Lewis, B. H. Stockett, and A. J. Garrett, representing the Lehigh and Wilkes-Barre Coal Co., the Lehigh Valley Coal Co., G. B. Markle Co., the Phila. and Reading Coal and Iron Co., the Locust Mountain Coal Co., and the Lehigh Coal and Navigation Co., respectively.

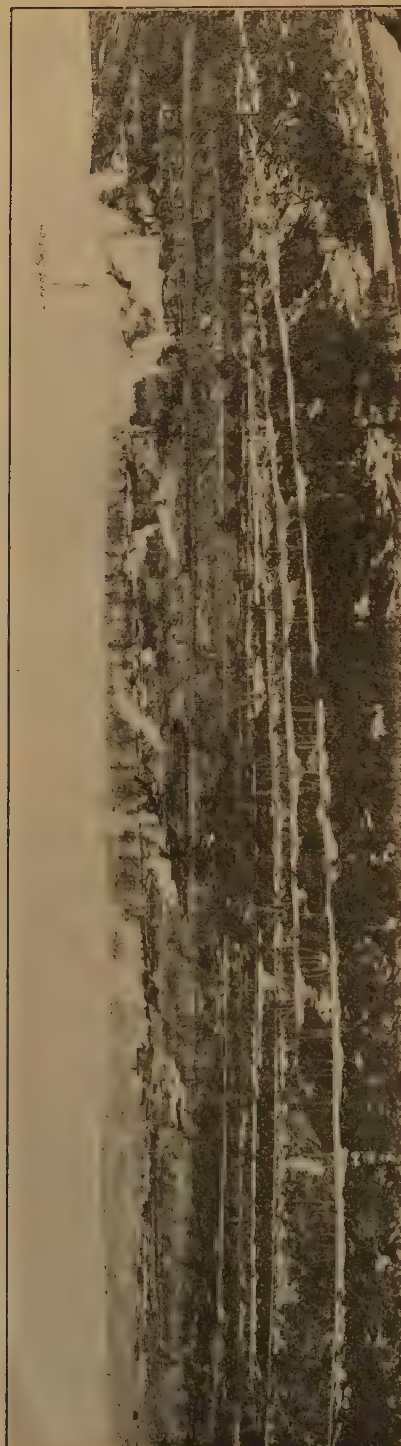


PLATE A.—PANORAMA OF STRIPPING SHOWN IN FIGS. 6 AND 6A.

There is also an early record of a similar quarry at Beaver Meadow and there may have been others whose record has not come down to us. After these first strippings, however, there is no record of such operations for many years. The first strippings, as we now know them, were commenced about 1864 when William Milnes carried on a hand-stripping operation at Jeanesville. Even this was a comparatively isolated instance and not until the late '70s did such operations become a general practice. About 1874, strippings were carried on at Beaver Meadow and in the Panther Creek Valley, as well as in other parts of the region. Thereafter, throughout the middle and southern fields until the advent of steam shovels, and afterward, immense quantities of overburden were removed in such operations, as is attested by numerous overgrown and forgotten spoil banks yet remaining along the hillsides of the coal valleys.

The literature of anthracite strippings, which is very meager, begins about this time with articles in technical journals descriptive of certain individual operations. Also, in the records of the Second Geological Survey of Pennsylvania there is a report by H. M. Chance on Anthracite Mining Methods and Appliances, published in 1883, which gives interesting plates and descriptions of early strippings.

The U. S. Geological Survey has recently commenced the collection of stripping data for the anthracite and bituminous fields, but the anthracite record is yet very incomplete, owing partly, at least, to a lack of detailed information in the reports furnished by the anthracite operators.

The earliest strippings were all hand excavations and the methods employed were much the same as those now used for very small operations. The operation consisted merely of digging out the stripping area with pick and shovel and removing the excavated material to the nearest available dumping place in carts or wheelbarrows. Horse-drawn scrapers were used in the larger jobs, some of which were extensive. Where the stripping became of considerable depth, and especially in rock, a horse-operated derrick was used to elevate material from the lowest levels.

The first steam shovel came into the anthracite region in 1881 and was used at Hollywood by L. E. Klotz, contractor for C. Pardee. This was 6 years earlier than the first date on which steam shovels were used in ore mining in the United States. It was one of the early type of Oswego shovels with a dipper of 1 yd. capacity and weighing 30 to 35 tons. The next shovel was used by J. W. Crellin at Silver Creek. This was larger, and operated a dipper of 1¾ yd. capacity. Shovels came into use rapidly after this and this general type of equipment has remained in use since, though the sizes of the shovels have gradually increased until now 70- and 80-ton shovels are the rule and larger equipment is coming in.

During the latter part of the '90s some experiments with aerial tramways were made, but their capacity proved low and they were soon abandoned. These have also been tried in ore-mine stripping with equal

lack of success, though they are used almost to the exclusion of other equipment in quarries.

Anthracite strippings are a notable example of the way in which labor-saving devices have held down operating costs in the face of steadily advancing labor costs. In the early days when $1.10 was the regular wage for a 10-hr. day, unit stripping costs were around $0.26 per cubic yard for clay excavation and $0.60 for rock. With present wages nearly double the early rate, stripping costs range from $0.16 to $0.20 for clay and from $0.35 to $0.40 for rock. This is due, largely, to the use of steam shovels and the increase in the size of the shovels employed.

Stripping is at the present time, and has been for very many years, one of the most important adjuncts to the mining of anthracite coal. In the northern regions, that is, the Lackawanna and Wyoming Valleys, stripping methods have been employed but little, because the veins are, as a rule, of limited thickness, and also because the pitch is so light as to render crop strippings unnecessary. In certain parts of these regions, however, these generalities do not hold good and stripping methods could be employed to economic advantage. Already a few areas have been stripped and others are contemplated.

In the middle and southern fields, on the other hand, strippings are an essential part of mining operations and form about 5 to 10 per cent., on a tonnage basis, of all mining done in these regions.

During the year 1914, the total yardage removed in stripping operations in the anthracite regions was 8,370,174 cu. yd. The division of this yardage between hand and shovel strippings was about 5 per cent. of the former to 95 per cent. of the latter. Due to the fact that a great deal of this work is done by contract on an unclassified basis, the percentage of rock removed as compared to other overburden is not obtainable, but is approximately 35 per cent. In the years since stripping operations were commenced in the anthracite region it is probable that at least 300,000,000 cu. yd. have been removed by stripping. Moreover, this figure does not include coal excavated by hand and by steam shovels from stripping areas, which is a substantial quantity.

Immense quantities of overburden have been removed in individual stripping operations in the anthracite field and some of these operations have extended over many years. One or two million cubic-yard strippings, and larger, are not unusual, and some of these have been in continuous operation from 12 to 20 years. To arrive at the record in respect to continuous operation, however, it is necessary to go back to the old Quarry mine at Summit Hill again. This was in operation nearly 30 years and the total output of coal from it was nearly 2,000,000 tons. In cubic yards of overburden removed, however, and in depth of excavation, this old operation does not compare at all with more modern ones. Total excavation in the largest modern anthracite stripping is 4,500,000 cu. yd.

exclusive of coal, with a surface area of 53 acres and a maximum depth to the top of the coal of 150 ft.

Reasons for Stripping

The economical problem of producing the largest quantity of coal for the least expenditure of money can be assigned as the fundamental reason for the employment of stripping methods. There are, however, many factors that contribute to this fundamental one. A higher yield of prepared sizes is usually obtained by mining the coal in the open under conditions of natural light; a cleaner product is obtained for the same reason and also because of the removal of overlying impurities; also, the total output of a colliery can be increased by this method and fluctuations of output eliminated for the reason that the entire body of coal is thus exposed to view and can be attacked in few or many places as desired. It is usually one or the other of these considerations that determines the stripping of any area. In some cases these are such potent reasons as to induce the stripping of certain areas even where the profitableness is otherwise doubtful. The doctrine of conservation also, is a consideration of some influence in certain cases.

Strippings on an economic basis, however, fall into two general divisions: Those that are undertaken to obtain coal at a cheaper cost than by ordinary mining, and those that are undertaken to recover coal that is unobtainable by ordinary methods, or obtainable only at a prohibitive cost. The latter is the larger division, because unless the coal to be uncovered is to be mined by other than the ordinary chute methods of inside mining—and often when the coal is to be mined by other methods—the output from a stripping is at a higher price than the average for the colliery. This is because the cost of inside mining is not appreciably decreased by uncovering the coal and because the cost of removing the cover must be added to the mining cost. There are, of course, many stripping operations, wherein the unit cost or the cost per ton of coal obtained or recovered is less than the average cost of inside mining. These occur, however, when conditions are exceptionally favorable; when, for instance, the coal after being uncovered can be excavated by hand or by steam shovel at a very low unit cost, which, added to the cost of removing the cover, is still lower than the cost of mining by ordinary methods. These cases, though numerous, are exceptions, and are undoubtedly becoming fewer in number.

It is, nevertheless, the preconceived idea of a great many people, even of some more or less intimately connected with coal mining, that coal is produced by stripping operations at a greatly reduced cost over general mining operations. That this is erroneous in the majority of instances is shown by the fact that lessors of coal lands often lower their royalty

rates in order to induce the stripping of areas that otherwise would remain unmined.

Limits in Area and Depth

The economical limits of stripping in area and depth are affected by so many factors, that this question is without exception the most important to be faced in stripping operations, and the variations in the values assigned to these different factors by those engaged in stripping operations are the most marked of all differences in practice. Some of the factors can be named, as: Quality of coal; margin of profit; classification of material to be removed; excavation of coal by chute mining, or by steam shovel; refuse to be handled, etc. As a rule the sum of the equation containing all of these is resolved for each operating company or district into an empirical ratio of cubic yards of cover to tons of coal which

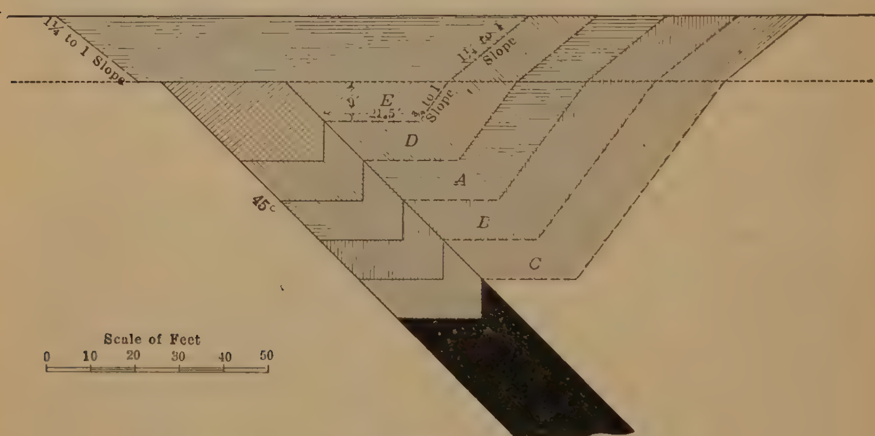


Fig. 1.

is then applied to other stripping work with perhaps slight variations to allow for local changes in conditions. Such a ratio once established for any region or operation sometimes remains unchanged for a long period and is applied in turn to each new stripping operation that presents itself. This method is not incorrect and in most cases it serves as well as a more complicated calculation, but there are many possibilities of a failure in the basic principle if a false value has been assigned originally to certain factors, or if certain factors have been neglected. Suppose, for instance, that it is decided to expend on coal recovered from the stripping shown in Fig. 1, an amount per ton equal to the average margin of profit of the colliery, the return on the investment being considered to be secured by certain factors or advantages that do not lend themselves readily to calculation in exact figures. Perhaps the output is falling off, or possibly the coal to be uncovered is of exceptionally good quality, and may improve the

car yield of the colliery a point or two. Then this cost per ton figure is translated into a ratio of cubic yards overburden removed per ton of coal uncovered, say, still merely as an instance, 2¾ yd. per ton. Then the limits in area and depth are set to produce this ratio. This, on its face, appears to be satisfactory, but not when the operation is resolved into component parts as is shown on this diagram by the dotted lines. There, the lowest component part which comes within the limits of the ratio is

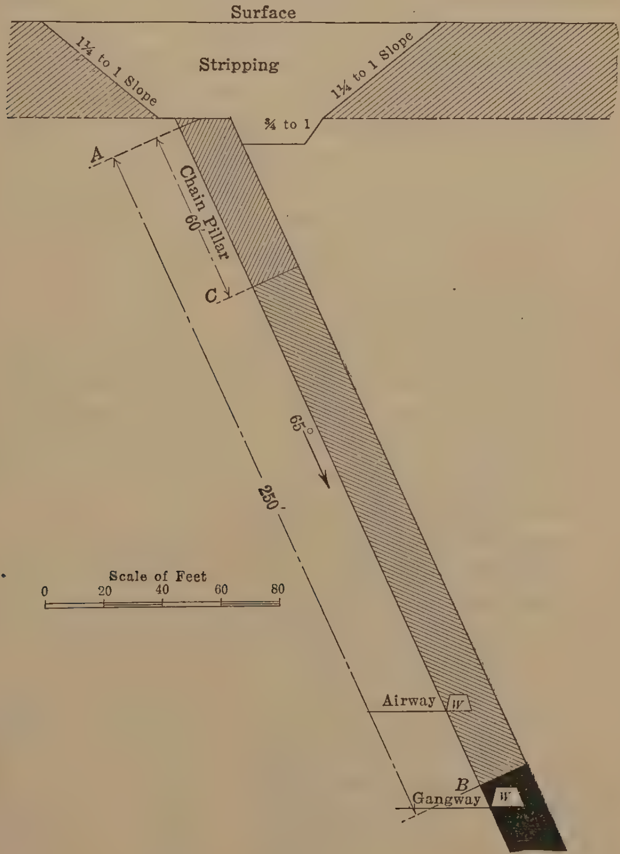


FIG. 2.

marked A. The component parts B and C are 3 cu. yd. per ton uncovered and 3½ cu. yd. per ton uncovered, respectively, and their average is 3¼ cu. yd. per ton. In other words, they are operated at a loss, regardless of the fact that component parts D and E are operated at a considerable profit. It is required that certain marked advantages be gained by the removal of B and C to justify this operation. In these days of increasing costs of coal and narrow margins of profit, conservation can hardly be considered one of these advantages.

Another condition is illustrated in Fig. 2. This is a crop stripping of a character common to the southern fields, where the clay and gravel overburden must either be removed or a chain pillar of coal left below the surface to prevent the contamination of the prepared coal. It is to be assumed that the stripping is for a virgin area.

In this illustration it would be necessary to leave a chain pillar of about 60 ft. if stripping were not resorted to. The coal below this chain pillar can be mined at as low a unit cost for cutting and loading as would be obtained by mining all the coal from the gangway to the surface provided the cover were removed. By the latter method, lower unit development costs would be obtained, but the reduction would amount in this case to only a few cents per ton. Still, taking 2¾ cu. yd. of overburden to 1 ton of coal as our economical ratio, we find the ratio in this illustration to be 4 cu. yd. to 1 ton of coal in the chain pillar, and 1 cu. yd. to 1 ton of coal in the entire area between the gangway and the surface. The coal that can be strictly classed as stripping coal is, therefore, secured at a loss, and assuming that no other factors need be considered, the stripping should not be undertaken.

Of course, in presenting this illustration, I do not consider certain veins in the southern anthracite fields that are very thick and in which the character of the coal is so loose and friable that any excavation in them is likely to cause a run of the coal that is not stopped short of daylight. A chain pillar is obviously of little value in such a case, and if the coal is to be mined at all stripping must be resorted to.

These are only two instances of possibilities of fundamental error in preliminary calculations, but they serve to illustrate the point. In general, in the anthracite region, veins that are too thin to be worked at a

Table 1

Pitch, Degrees	Thickness of Vein, Feet	Remaining, Per Cent.
1. 15–25	10–20	55
2. 15–25	20–30	60
3. 15–25	Over 30	65
4. 25–35	10–20	50
5. 25–35	20–30	55
6. 25–35	Over 30	60
7. 35–45	10–20	40
8. 35–45	20–30	50
9. 35–45	Over 30	55
10. 45–60	10–20	40
11. 45–60	20–30	50
12. 45–60	Over 30	55
13. 60–80	10–20	50
14. 60–80	20–30	55
15. 60–80	Over 30	60

profit are not mined, even though such mining might not cause the colliery to be operated at a loss, and stripping operations should undoubtedly be treated in the same manner.

The above illustrations were assumed to be of virgin veins in order to simplify the explanations. Unfortunately, the majority of large strippings are of veins worked and reworked in days past in such a way that an accurate calculation of the amount of coal remaining is impossible. This problem can only be solved by a guess, as is done where contracts are let on the basis of payment by the ton for coal recovered. Fortunately, however, many strippings of worked-over areas have been completed at this time, or are so far advanced that a record of the coal actually recovered or remaining in the worked-over areas can be obtained. A provisional tabulation has been made of all these records that are available, which gives the average results shown in Table 1. Some of the figures from which this tabulation was derived are given in Table 2.

Table 2

Kind of Stripping	Thickness of Vein, Feet	Pitch, Degrees	Extent Worked	Percentage of Original Coal Remaining
Anticline	55	20	Robbed and rerobbed	60.0
Crop	50	60–80	Mined and robbed	60.0
Anticline	40	15	Mined and robbed	50.0
Crop and basin	40	40	Mined and robbed	40.0
Crop	40	30	Mined and robbed	60.0
Crop	35	40	Mined and robbed	50.0
Crop	30	40	Mined and robbed	50.0
Crop	25	45	1st mining	82.5
Crop and basin	25	20–50	1st mining	70.0
Crop	25, 11	20	Mined and robbed	50.0
Crop–3 splits	10, 15	50	1st mining	70.0
Crop	16	20–35	1st mining	55.0
Crop	16	25–35	1st mining	50.0
Crop	15	50	1st mining	68.0

Tables 1 and 2 consist entirely of examples of stripping from areas mined 20 years or more ago. In more modern mining better extraction has been obtained, and this fact must be taken into consideration where the stripping of such an area is contemplated.

With all these factors to be considered, and with the varying margins of profit at different collieries kept in view, it can readily be seen that no universal ratio of yards of cover to tons of coal can be set as the profitable limit for stripping operations. At each operation this limit has to be worked out for itself. The ratios in use throughout the entire region

vary usually from 2 to 4 to 1, with an average of about 3 to 1. In some instances ratios as high as 5 to 1 have been used, but this means that the cost per ton, for stripping only, would be $0.80 to $1.50, to which must be added mining, preparation, and overhead costs.

Equipment and Operation—Characteristic Strippings

The equipment required for a one-shovel operation is about as follows:

1 70-ton shovel.	1 steam drill.
3 18-ton locomotives.	1 water tank.
20 5-yd. dump cars.	1 boiler.
1 star drill.	1 blacksmith shop.

Necessary rails, sills, pipe lines, tools, etc.

The total capital outlay for such an outfit is approximately $30,000.

The average force required to operate a one-shovel stripping consists of about 35 men, roughly as follows:

1 foreman.	4 jackmen.	2 drillers, 8 helpers.
1 shovel engineer.	3 locomotive engineers.	1 boiler fireman.
1 craneman.	1 dump boss.	1 blacksmith and helper.
1 fireman.	6 dumpmen.	2 coal diggers.
1 watchman.	1 track boss.	1 driver.
2 laborers.	2 trackmen.	1 switchboy.

The wages paid these men amount to $2,100 per month. The shovel engineer is paid $140 a month, the craneman $95, locomotive engineers $0.25 per hour. These rates are all subject, however, to the recent increases granted the mine workers, ranging from 7 to 15 per cent.

When a stripping is decided on and its limits staked out by the engineers, an inspection of the ground determines the method of opening it. Usually the cuts at the higher elevations are made first. After that the problem is almost entirely a transportation one. Steady operation of the shovel or shovels is the object to be secured. Everything must contribute to this end—tracks and rolling stock must be in good condition, turnouts must be maintained, and the grades must be as easy as the nature of the ground will permit. If in rock, drilling and blasting must be kept well in advance.

The method of opening a stripping with either a Bucyrus 70-ton shovel or a Marion 60-ton shovel, which are the two types most widely used in anthracite stripping work, is as follows (Fig. 3A): For the first cut the track is laid on the surface along one limit of the stripping, usually the bottom rock side, and the shovel cuts down grade alongside the track until a depth of 9 ft. is reached, this being the maximum cut that the shovel can take and load overhead. When the first cut is completed for the length of the stripping, the track is laid in this cut and the shovel

again cuts down grade until a depth of 9 ft. below the first cut is reached. The shovel then continues cutting toward the other limit, the additional depth being determined by the depth of surface over the vein up to 30 ft., which is considered the proper maximum height for a clay cut. In working by the above method, it is necessary to leave a bench at least 13 ft. in width for the laying of the track. Local conditions, as a rule, render it impossible to maintain any such plan for the entire life of a stripping.

The first cut as described above is always the first made in a stripping except in the case of what is known as a side-hill stripping. Here the track is laid on the surface and the shovel started at an elevation that will give the required cut at the vertical limit.

Rock cuts are usually made from 22 to 25 ft. in height, though more recent practice is to keep the height down to 12 or 15 ft. This height depends somewhat on the nature and hardness of the rock. The lower height seems better for very hard rock, the reason being that the rock is broken up by blasting better than in the case of a higher bank, especially the upper portion of the bank, and consequently is more easily loaded. A saving of 25 per cent. in cost per cubic yard is claimed for this method.

Drill holes for blasting are arranged in parallel rows, usually three, with the holes 12 to 20 ft. apart and the holes in each row staggered. From 15 to 25 holes are fired in a battery, or more if possible. After the holes have been drilled a charge of from three to eight sticks of 40 per cent. dynamite is used to "spring" each hole. By "springing" is meant the blasting of a pocket at the bottom of the hole to take a sufficient charge of black powder. It is sometimes necessary to spring a hole two and three times. The charge of black powder then used varies greatly, according to the nature of the rock and the amount of powder that can be put into the hole. The pocket itself, and the hole for about 2 ft. above the pocket, is filled with powder and the remainder of the hole with clay or coal dirt.

For 25- and 30-ft. holes, "Star" type churn drills are usually used with a customary diameter of 4 in. In solid rock the progress is about 30 ft. per shift of 9 hr., or an average of 3⅓ ft. per hour. For holes 12 ft. deep or under, a steam tripod drill is used and about double the progress of the churn drill, or 6⅔ ft. per hour, is made.

Average costs for drilling and blasting are as follows per cubic yard of rock excavated:

Labor drilling and charging, depreciation equipment, etc....	$0.045–$0.065
Powder....	$0.055–$0.080
	$0.100–$0.145

The tracks to the dump are always on an ascending grade of at least 1 per cent., though usually higher. Four per cent. is common and grades as high as 7 per cent. have been used. The grade of the tracks in the

stripping pit is governed by the necessary rise in elevation to reach the dump. The locomotives used vary in size up to 20 tons, the latter being about the heaviest type that can be used safely on a dump of any height. A 20-ton locomotive will push:

10 4½-cu. yd. cars on a 1 per cent. grade.
8 4½-cu. yd. cars on a 3 per cent. grade.
6 4½-cu. yd. cars on a 4 per cent. grade.

The general, and best, practice for stripping tracks is to use 60-lb. rails and nothing under a No. 6 frog. Curves should be kept to under 10°, though 20 to 25° curves are used, especially in forming a dump.

Dumps are made of all heights and sizes, though there is less maintenance cost with heights of about 25 ft. Dumps of greater height settle and slip easily, especially in wet weather.

The cars most widely used in stripping work are the Eastern and Western type side-dump cars. The Eastern type is of 4½ to 5¼ cu. yd.

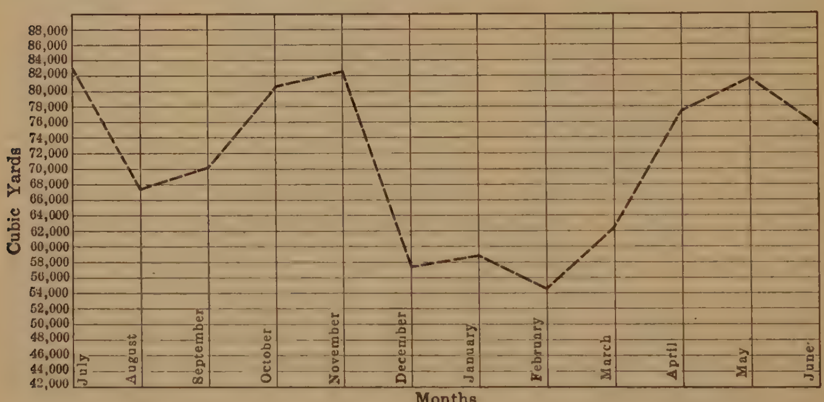


FIG. 3.

capacity and the Western type of 4 cu. yd. Some 8- to 10-cu. yd. cars are in use and the results obtained seem to be satisfactory.

Under proper conditions, outputs as high as 30,000 cu. yd. per month have been obtained for one shovel in clay. The average, however, is only about 18,000 cu. yd. for clay and 10,000 to 12,000 for rock. The output from strippings varies considerably according to the season of the year. The curve shown on Fig. 3 illustrates this for the entire stripping operations of one company averaged by months over a period of 4 years. This curve, while affected by other factors, serves to illustrate the drop in output during the winter months.

If the stripping is not too deep, all the excavated material can be removed by locomotives. In many cases, however, this is not feasible and hoisting planes must be resorted to. Practically without exception,

even in the largest operations, these are single-track planes operated by small second-motion hoisting engines with a capacity of about 150 dump cars per day, or about the output from one shovel. The practical problem involved in putting these planes down along the steep sides of the average pit is often a serious one. Some of the planes are anchored on a slope of 50° to 60° pitch by bars sunk into the solid rock to which the roadbed is tied, presenting a very interesting sight. While nothing can be said against these small hoists for a one-shovel stripping, it is undoubtedly bad practice to use them in the larger operations employing two or more shovels. There are practically none of these that cannot be laid out so that the output from two shovels can be brought to the foot of one plane, and this plane should be equipped with a hoist capable of handling with ease 300 and more cars per day. This plane can be either single-track or double-track, but the grade should be maintained at about 20°, which is the average for the single-track planes now in use. Some figures have been worked up showing the comparison of the cost of the two varieties of planes, taking a double-track plane handling only the output of two shovels which would allow the greatest advantage of comparison possible to the small hoist. The first cost of the small-hoist job is very low, as the hoist itself is usually picked up second-hand around the collieries. It would be something as follows for a 300-ft. length of plane:

Hoist	$500
Tracks, track material, rope, etc	700
Grading for hoist and plane	1,000
	$2,200

For the double-track plane with the larger hoist the figures would be:

Tracks, track material, ropes, etc	1,100
Hoist	5,000
Hoist house, pipe lines, etc	800
Grading for hoist and plane	3,000
	$9,900

To operate the single-track plane two top-men, two bottom-men, one locomotive engineer, one hoist engineer, four men and a boss on the dump, are required, while the double-track plane would require three top-men, three bottom-men, two locomotive enginemen, one hoisting engineer and seven men and one boss on the bank.

The comparative cost would be as follows:

	Single Track	Double Track
Labor per day	$17.88	$26.21
Power	4.30	6.48
Interest and depreciation, 15 per cent.	1.00	4.00
	$23.18	$36.69

Figuring 150 cars for the single-track plane, the operating cost per car would be $0.155 and at 300 cars for the double-track plane $0.122 or a difference of $0.033 per car.

The location of the limits for a stripping are set on a line where the normal slope of the overburden figured from the bottom of the final cut intersects the surface. Naturally a shovel cannot cut to any such slope and must accomplish the same result by a series of steps such as shown in Fig. 3A. The normal slope that earth of a clayey nature will take is about 1 to 1. Sandy ground requires 1½ to 1 or even 2 to 1, while rock can be cut nearly vertically if the height of bank does not exceed one shovel cut. For greater depths, ½ to 1 must be allowed or even 1 to 1 if the rock is of a shaley nature. The importance of having the foot of the stripping slope well back from the bottom rock of the coal, to prevent

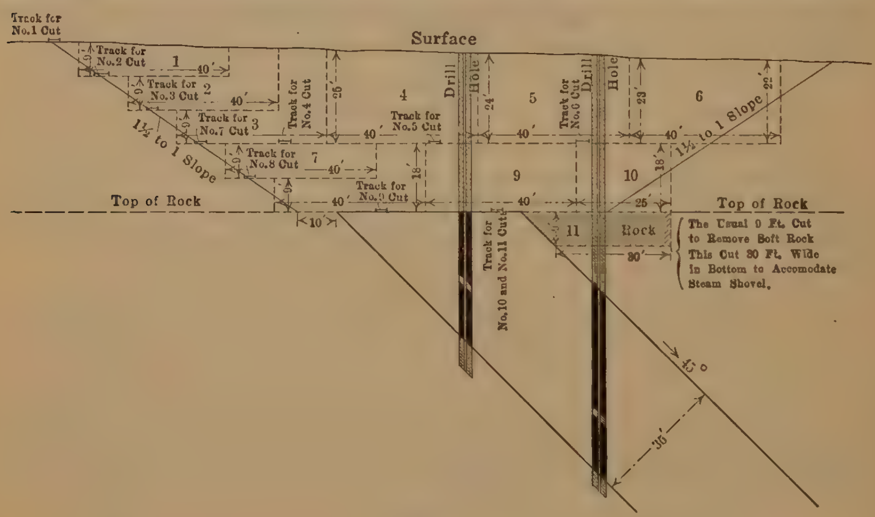


FIG. 3A.

the washing of overburden into the exposed vein by rains, is very great. The standard width for this ledge or berm is 10 to 15 ft. This also is shown in Fig. 3A.

Fig. 4, 5 and 6 illustrate crop, basin and anticline strippings, into which divisions all strippings fall. Fig. 4, showing the crop stripping, is interesting in that it is also an illustration of a chain pillar left in early mining under the surface wash, which here was 40 ft. and more in thickness. Breasts were driven up in the old days until the roof caved in and were then abandoned. The exact width of the chain pillar remaining has not yet been determined, but it is at least 150 ft. Here the object is not to uncover all the coal but merely to remove sufficient of the clay and rock to make possible the mining of the coal from inside with minimum loss. To do this it is impossible to drive chutes up in the old vein

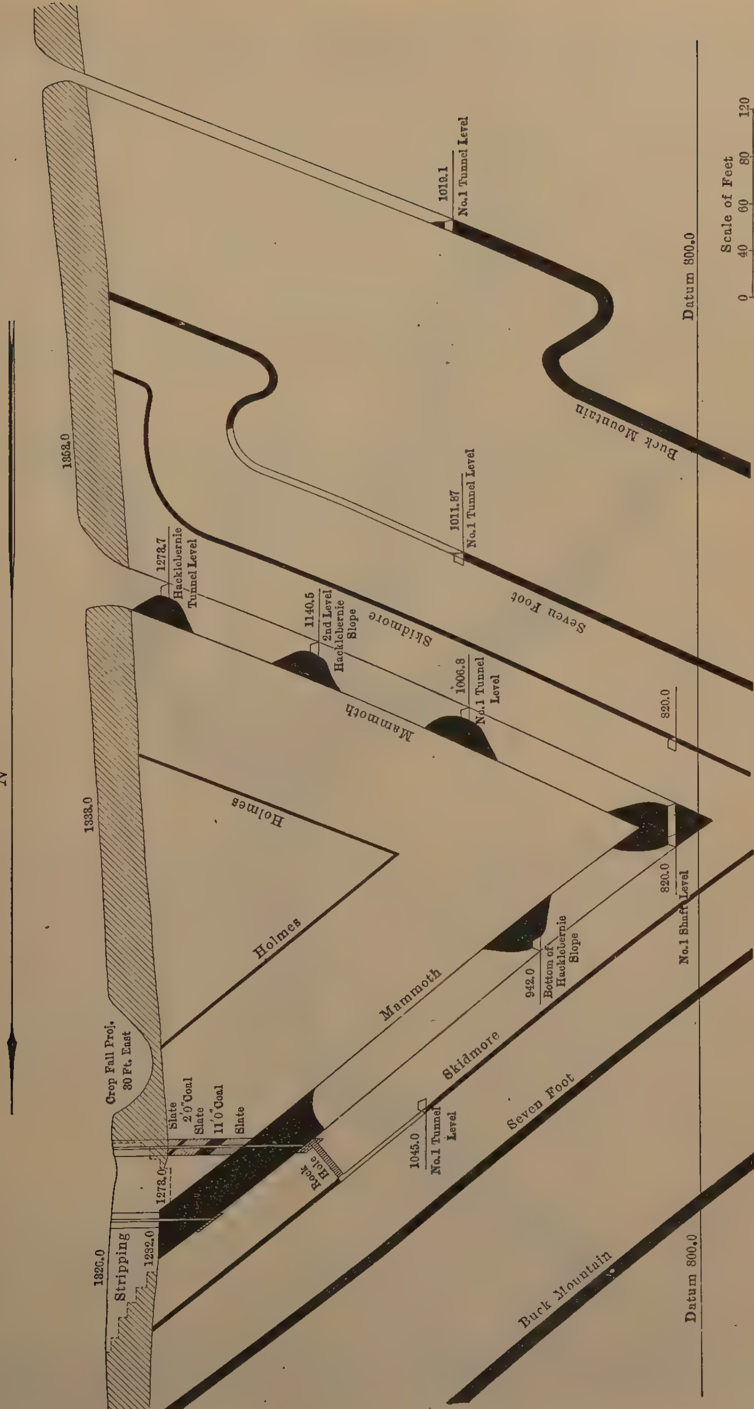

N
1352.0
1333.0
1326.0
Stripping
1232.0
1278.0
Crop Fall Proj.
30 Ft. East
Slate
2'0" Coal
Slate
11'0" Coal
Slate
Rock Hole
1045.0
No.1 Tunnel Level
Holmes
Mammoth
Skidmore
Seven Foot
Buck Mountain
1278.7
Hacklebernie Tunnel Level
1140.5
2nd Level Hacklebernie Slope
1006.8
No.1 Tunnel Level
942.0
Bottom of Hacklebernie Slope
820.0
No.1 Shaft Level
1011.87
No.1 Tunnel Level
1019.1
No.1 Tunnel Level
Datum 800.0
Scale of Feet
0
40
60
80
120

FIG. 4.

and therefore a gangway is driven in a small underlying vein from which chutes are driven up to a point opposite the lowest edge of the chain pillar and rock holes are then driven through into this pillar.

Fig. 5 is of a large basin stripping. This was operated for many years and the various stages of excavation are shown illustrating the characteristic methods of opening strippings of this kind. This stripping is of a virgin vein. Its width is 300 ft. and its length, 4,800 ft., maximum depth 100 ft. of cover, and cubic yards of cover per ton of coal, 3.46. This stripping was in operation from 1900 to 1915, starting with an Oswego shovel and changing in 1909 to a 70-ton Bucyrus.

Fig. 6 illustrates an anticlinal stripping. It is of a worked-over area where it is estimated that 60 to 70 per cent. of coal remains. Upon this

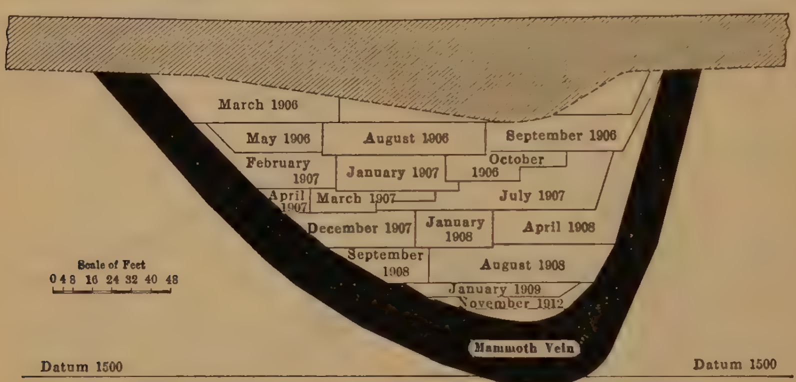


Fig. 5.

percentage depends its profitableness, as it has been undertaken primarily to form a final barrier against a fire that has been raging to the east of it for many years. The vein is 55 ft. thick and is on 20° pitch. It has been robbed and rerobbed and robbed again, but because of its thickness and the unhandy pitch, as well as the time of the mining, which dates back to early '50s, it is thought that not over 35 per cent. of the coal has been extracted. Fig. 6A shows a plan view of this stripping and Plate *A* is a panorama photograph taken looking north from the south side during the progress of the operation.

Engineering Methods

Engineering methods begin with the location of a stripping and the preliminary estimates of its profitableness, and continue through the calculations of yardage removed in the various stages of excavation down to the final tabulation of results obtained. In laying out the stripping and making the periodic estimates of excavation, a high degree of standardization prevails throughout the anthracite region. A base line is laid out

parallel to the length of the stripping, with stakes at regular intervals of 20, 25 or 27 ft. These are numbered in order, beginning with one. At each of these stakes lines are laid out at right angles to the base line

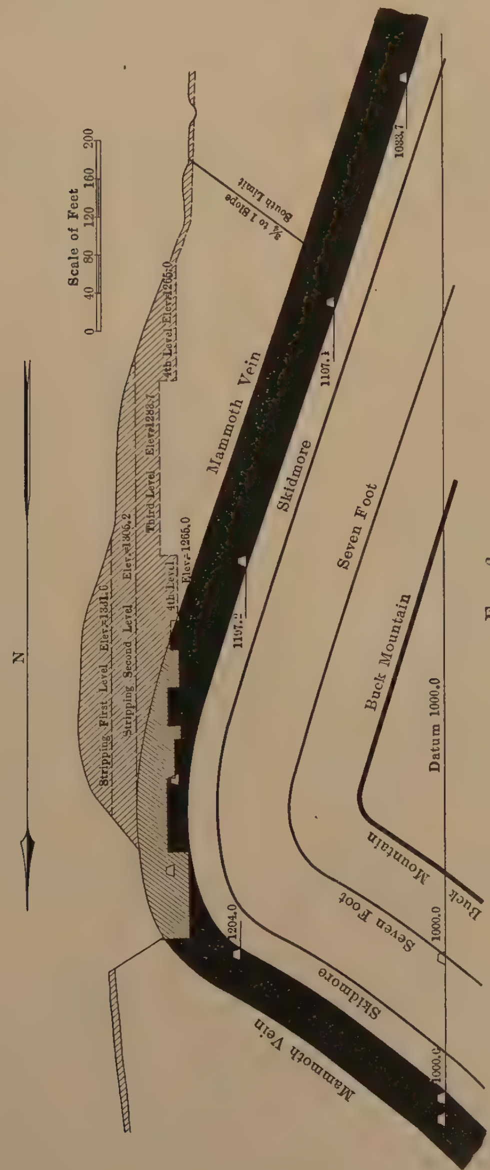


FIG. 6.

across the width of the stripping and stakes set at 20- or 25-ft. intervals. Sometimes these are numbered, but the best way is to assign letters to them, beginning with A. At any stage of the stripping operation these

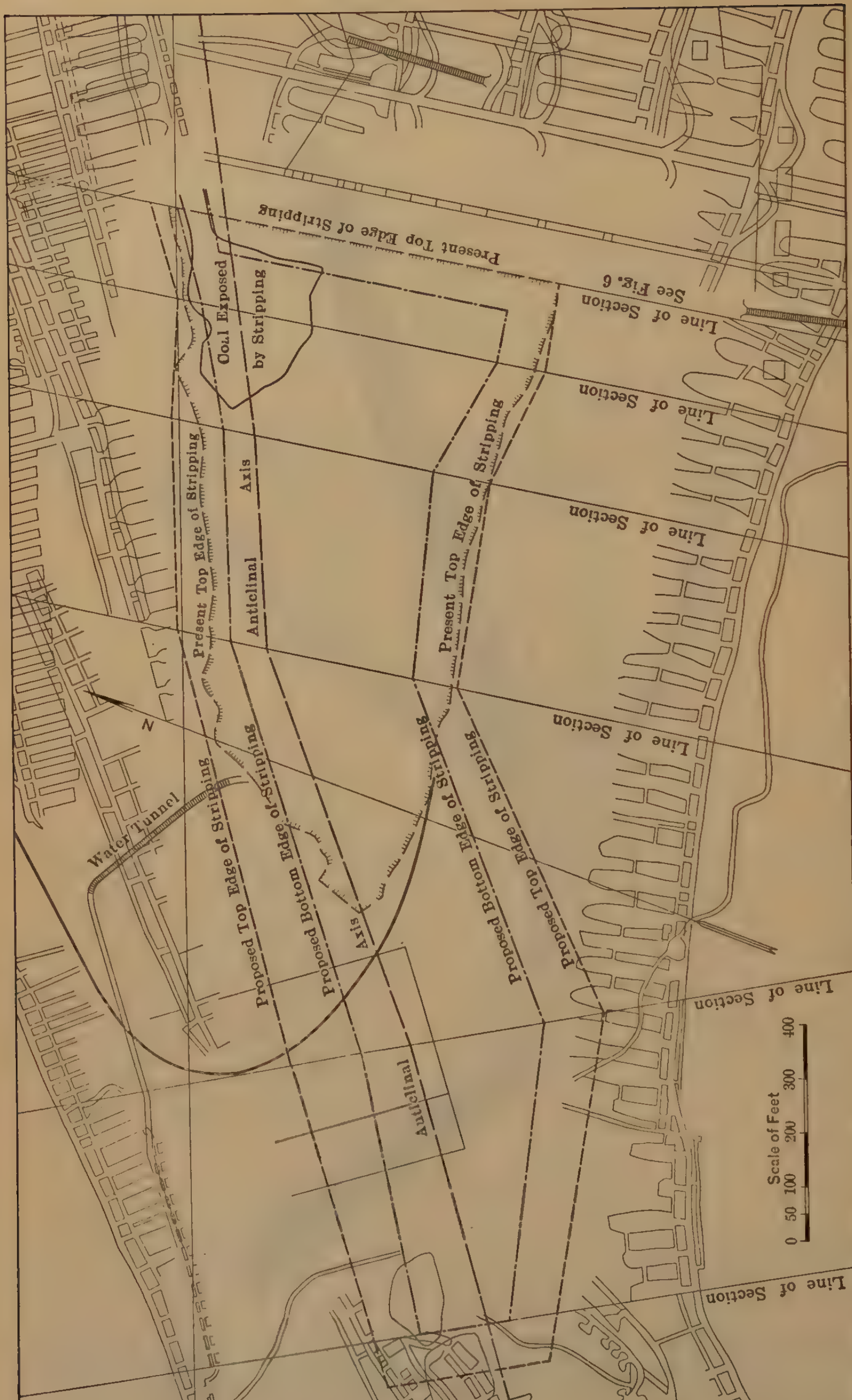


FIG. 6A.—MAP SHOWING PLAN OF STRIPPING.
(To accompany cross-section shown in Fig. 6.)

points can be readily relocated and levels run over them to ascertain the yardage removed. Cross-sections on each of the right-angle lines are plotted in the office on rolls of printed cross-section paper or tracing cloth. These plottings are invariably on a scale of 1 in. to 10 ft.

The use of 27-ft. intervals along the base line is worthy of note, as it is rapidly coming into practice. By this method the area in square feet of the cross-section on each right-angle line can be translated into cubic yards without any multiplication.

At the lateral limits of a stripping it is customary to place limit stakes, using large stakes painted red or white to give them special distinction and permanence.

Classification

The overburden in stripping operations is classified usually as rock and earth. This classification is purely for convenience in carrying out the contractual relations when the contractor has undertaken to remove earth at one price and rock at another. Where the price of removing rock and earth is the same, no classification is required and the rate established is known as an unclassified rate. The definitions of rock and earth vary. As a rule, boulders of 1 cu. yd. and over in size are classified as rock, and all smaller as earth. Most variations in classifications occur at the gradation zone between earth and rock where the material to be excavated, though stratified and of the appearance of rock, is soft and of the consistency of earth. In some cases a third classification known as loose rock is employed, as is witnessed by the following definition:

"Loose rock will include all stone and detached rock found in separate masses, containing not less than 3 cu. ft., nor more than 1 cu. yd.; also all slate, coal or other rock, soft or loose enough to be removed without blasting, although blasting may be resorted to; also stratified rock in layers of 8 in. thick and under, separated by strata of clay."

As a rule, however, rock and earth are the only classifications needed. One particularly complete definition of these two classifications is as follows:

"Excavations shall be paid for under the following classifications:

"EARTH, which shall include clay, sand, gravel, loam, decomposed rock and slate, whether lying in place or not; stones or boulders containing less than 1 cu. yd., indurated clay or other earthy material, cemented gravel, and all coal, shale, slate, soft friable sandstone, and all other material in place except rock as hereafter defined; also stratified solid sandstone in layers of 8 in. or less in thickness when separated by stratified earth as above defined.

"ROCK, which shall include all solid sandstone in place in layers greater than 8 in. in thickness, whether separated by layers or earth as above defined or not, and all boulders containing more than 1 cu. yd."

The loose-rock classification is undesirable and the tendency is, properly, away from it.

The average rates paid for classified strippings have been touched on earlier in this paper. For unclassified work the average rate varies from $0.20 to $0.25 per cubic yard.

1916 ____ Stripping ____ Colliery

Monthly Statement showing Cost of Stripping

Month		Description of Cut	Average Depth of Cut			Surface Area	Excavation						Mining of Coal		No. Cars Loaded	Breaker Car Yield	Tons of Coal	Excavated per Ton of Coal	Total Cost of Stripping and Mining	Cost of Stripp'g per Ton			Cost of Mining per Ton			Tot. Cost of Min'g & Stripp'g per Ton			Remarks
		of Cut	Earth	Rock	Total	Sq. Ft.	Earth		Rock		Total		Coal							A	B	C	A	B	C	A	B	C	
							Cu. Yds.	Cost	Cu. Yds.	Cost	Cu. Yds.	Cost	Cu. Yds.	Cost															
Year 1915	A																												
	B																												
	C																												
January	A																												
	B																												
	C																												
December	A																												
	B																												
	C																												
Year 1916	A																												
	B																												
	C																												

A — Stripping for Month — Coal Mined During Month
B — Total Stripping to Date—Total Coal Mined to Date
C — Completely Stripped and Completely Mined Section

FIG. 7.

Prospecting

Prospecting by diamond drills is of the utmost importance and the most successful stripping operations are those that have been most thoroughly laid out as the result of drilling. Holes are placed at proper intervals on cross-section lines sufficiently close to each other to insure, so far as possible, avoidance of error in the location and character of the vein or veins to be stripped. These cross-sections in badly distorted areas should often be as closely placed as 100 ft. apart. Prospecting by steam shovel is not to be recommended.

Special Devices, Methods, Accounting Systems, Etc.

Various special methods for keeping a record of the progress of strippings, their cost, economy, and other related data are in use. Some of the anthracite companies have kept records in great detail for many years covering the actual cost of stripping work, and the results obtained in the greater efficiency of their stripping operations are strong evidence of the value of such thoroughness. In some instances the contractor's costs and profits are watched as closely as are those of the company.

Keeping a careful record of the actual results from each individual

stripping cannot be too strongly urged. Fig. 7 is a reproduction of a sheet of this kind kept by one anthracite operator. The first columns can be passed over as merely giving tabulated data of value as reference. The final columns, however, show at each successive stage of the stripping operation the unit costs, based on the coal tonnage recovered, or to be recovered, of results to date, results for a completed section and probable final results. As the stripping progresses, a glance is sufficient to show the exact financial standing of the particular operation covered.

FIG. 8.

Another special form which is of great labor-saving value to the engineers engaged in calculating the monthly yardage excavations is shown in Fig. 8. The stripping area is laid out in squares by parallel and right-angle lines. The illustration shows squares of 25 ft., but a similar chart could be worked up for any dimension. By means of this chart the intersection of any two lines can be found with the minimum effort and with field calculations eliminated.

Yield and Waste

The yield from a stripping in per cent. of the total is naturally high. There are, however, certain losses such as the fuel loss; dump losses, where the coal is excavated by steam shovel; and various losses in digging and handling.

The fuel loss is, as a rule, the most important of these, as a large amount of coal must be used to supply fuel to the various shovels, locomotives, drills, etc., employed in stripping work and in a majority of cases this coal is cut by the contractor from the uncovered areas of the vein. In one large stripping employing eight shovels the consumption per day ran about 50 tons, which means a total consumption of nearly 15,000 tons a year. Another employing four shovels used 20 tons per day or about 6,000 tons per year. This loss is never less than 2 per cent. of the total coal uncovered and runs over 5 per cent. at times. A steam shovel of 60-ton class will use 2 to 3 tons per day. A 12-ton locomotive uses about 1 ton and churn drills and blacksmithing forges use about ½ ton each. Little economies can be effected in this connection, such as requiring the contractor to furnish well-constructed boxes or bins wherever coal is deposited or stored. These not only save considerable tonnage in the course of a year but also have a desirable moral effect on all concerned.

When fuel is not cut from the strippings, and always until coal is uncovered, fuel must be furnished from the colliery or from other strippings. This question of fuel is a rather important one and there is some difference of opinion as to whether or not all coal used in the operation of the stripping should be furnished from the colliery, rather than cut from the uncovered vein. In cutting from the vein there is always waste, because only the best coal is taken. Moreover, the shovels and locomotives use none of the fine coal that is produced. In spite of this, it is impossible to figure the value of coal in the ground at anything except the profit that would be obtained by mining and preparing it for market, while against coal furnished from the colliery all the costs of handling it in the various stages of mining and preparation must be charged and the comparison in cost is distinctly in favor of allowing the contractor to cut his own coal from the exposed vein.

The second loss or the dump loss applies only to vein areas previously worked over. There the loss occurs in the zone between pillars and old openings filled with gob and coal that has sloughed from the ribs of the pillars. No satisfactory separation between rock and coal can be made here with a steam shovel and the percentage of loss is high—perhaps 10 per cent. would not be an abnormal figure in some instances. This loss can be reduced by various methods. All cars containing an appreciable percentage of coal can be sent to the breaker and the separation made there, or a separate cleaning or separation plant can be erected for

the handling of all cars too high in refuse to be handled economically at the breaker. Still another way is to dump all cars containing an agreed on percentage of refuse on a definite section of the dump and to employ laborers to pick the coal from the refuse by hand. This method has been employed with success at various places. A single laborer has loaded as high as 2 tons a day by this method and if this average can be obtained the method is profitable. The dumps for the old Summit Hill Quarry mines, as mentioned earlier in this paper, are an example of the amount of coal that can be wasted into stripping dumps, as these dumps afterward took fire and burned with intense heat for many years. Evidently some progress has been made since that time in avoiding waste, but much yet remains to be done.

The other losses such as handling and blasting run as high as 5 per cent. at times, and the sum of all losses varies from 5 to 20 per cent., though the latter figure is unusual. Any reduction secured in any of these losses is true conservation.

Mining Coal from Strippings

The consensus of stripping experience is that it is more economical to mine coal from an uncovered area by hand loading into cars, or by steam shovel, than by the ordinary methods of inside mining. This is undoubtedly true as regards virgin coal areas and is usually true as regards worked-over areas, though here the difference is not so marked. When excavating coal in worked-over areas, it is necessary to excavate also all refuse and rock in old breasts, some of which might be left behind in chute mining. The recovery of coal from the strictly stripping area might also be less in a steam-shovel operation because of the various factors of waste that have been discussed previously. If, in addition, it is necessary to construct expensive planes to handle the product from the steam shovel, such an operation may be the more expensive, all factors considered, of the two. Ordinarily, however, the outside transportation cost will be found a little cheaper than the inside, unless the inside transportation can be handled without additional equipment, or men, especially if planes have been installed for the handling of the overburden.

It is a peculiar fact that loading into mine cars by hand in the open costs about the same per ton as coal recovered in excavation by steam shovel. At least such is the case in worked-over areas. There are no data on similar work in virgin areas, but the difference there would probably be slightly in favor of the shovel.

The objection to hand loading is, of course, its limited capacity, and its advantage is a cleaner separation of rock and slate from the coal.

In steam-shovel excavation of the coal, practice is about equally divided between contract and company work and circumstances to a large

extent determine the procedure. If the cover has been removed by a contractor it will usually mean a complete change of track gage and equipment for the company to take over the work, and if the cover stripping is still in operation, as is frequently the case, such a division of operations would be very undesirable. If possible, however, coal excavation in worked-over areas should be handled by the company and, if the output required will permit, the smallest size of shovel should be used. A cleaner product will be obtained with less waste in the dump and less handling of rock at the breaker. By the use of mine cars, also, a rehandling of the coal between the stripping and the breaker is often avoided with attendant breakage and rehandling cost. The rehandling cost is a considerable item and has equalled 7 c. a ton in certain cases. The minimum is not below 5 c. per ton.

Contracts

Written contracts between company and contractor are now nearly standard in essential form in spite of various local differences. It is, for instance, the universal practice for the company to furnish fuel and water though the mining of the one and the distribution of the other are generally required of the contractor. Where planes are required the company also furnishes, in most instances, the power to operate them. Practically all contracts contain certain standard clauses that aim to prevent waste of coal by the contractors. Enforcement of such clauses, however, varies.

The contractor in assuming the risk of profit or loss in the operation is supposed to assure himself so far as possible of the local conditions to be encountered, and no claim for extras is ordinarily allowed for unforeseen difficulties met with. On the other hand, no reduction in contract rates is asked by the company in operations where the natural conditions especially favor the contractor. As a rule, contracts are not made for each individual operation, but instead, cover all work done by the company over a period of years.

The usual form of contract provides for payment by the cubic yard, for all material excavated, at certain definite rates. Some contracts have been made, however, that provide that the contractor shall be paid by the ton, delivered at a fixed point to the company, a sum supposed to cover all costs of removing cover and mining coal. While such contracts are justified under certain conditions, they are not to be recommended. In their nature they are a gamble and their only virtue is that they sometimes secure the stripping of areas that would otherwise be left untouched.

Rates by the cubic yard are sometimes on a classified and sometimes on an unclassified basis. These terms have already been dis-

cussed. The unclassified rate is like the rate per ton of coal, somewhat of a gamble, and the classified rate is probably more satisfactory in the long run.

Contract vs. Company Strippings

For actual stripping of the overburden from coal deposits, the nearly universal practice among coal companies is to let the work out on a contract basis. This has been the accepted practice since the earliest days of hand stripping. Indeed, this probably accounts partly for the fact that such is still the practice, as customs that once gain a foothold in conservative mining regions are likely to go unquestioned for long periods, and to spread from small beginnings until they become the dominating note in immense operations. The contract system for stripping operations, however, has had other arguments than this in its favor. Coal operators have always been reluctant to have their attention distracted from straight coal mining by the injection of interests foreign thereto. Moreover, the margin of profit on coal recovered by the stripping method is so small, at the best, that it is practically essential before undertaking such work to know, within a cent or two per ton, just what the final cost will be and such is accomplished when by the contract system all risks of fluctuation in such costs are assumed by the party of the other part. Another argument is the usual one in favor of contract work, that such work is pushed more vigorously than company work, due probably to the fact that there is more individuality to the management of a small contracting company than to that of a large mining corporation. This argument does not apply to what is known as the individual operator, but another and more potent argument in his case is the high first cost of stripping equipment and his inability to keep it steadily employed. Some, however, of the individual operators do maintain their own equipment and operate their own strippings, and many of these have succeeded in excavating overburden at a figure under the usual contract rate for the region. That the large companies could secure similar results is probably true, though to do so it would be necessary to devise some method of handling the work entirely outside of the regular organization of the company. The question is an uncertain one, however, and the solution of the problem of securing decreased stripping costs must probably be looked for elsewhere. It is becoming more and more essential to the larger operator in the anthracite regions to consider carefully means of meeting the increasing cost of producing coal, which would be helped by stripping the remaining available areas to greater limits in both area and depth than is economical under present practice. If this increasing demand cannot be met by the contractor, then the operator himself must assume the burden.

Possible Improvement in Practice

Past history of stripping operations, as recited briefly at the beginning of this paper, has shown that in spite of large increases in the wages of labor and in the cost of materials the cost of stripping has been reduced by improvement in methods, and the employment of adequate labor-saving equipment. It can be confidently expected that the future of anthracite stripping will repeat this history. Already at one operation in the region this is the case. The problem was met here by the installation, at a cost of over $30,000 of one of the large drag-line excavators that have been so successfully operated in Middle Western coal strippings and in ship-canal excavations. Fig. 9 is a cross-section showing the measures stripped at this particular operation. Briefly, this machine is about 255 tons in weight and is of the revolving type. It is electrically operated through a complete control system which protects the motors from disaster due to overloading or sudden strain of any kind. The machine revolves in a complete circle with a radius of 125 ft.

Where the ground is at all regular this machine can be readily moved along when a cut is finished by casting the bucket to a point in advance and with the anchorage thus secured dragging the entire apparatus over wooden rollers. For uneven ground, or ground broken up by crop falls, the machine can be equipped as a shovel and results nearly as satisfactory secured.

Wherever it is possible to cast the excavated material to one side, rather than to load it into cars, labor is largely dispensed with. The labor item in ordinary strippings is probably 70 per cent. of the total cost of stripping, but in such an operation as described this percentage is reduced to about 50 per cent. From the results obtained to date it is believed that present stripping costs can be reduced by these advanced methods at least 10 c. per cubic yard, allowing amply for interest and depreciation items. The cost for power has been proven to be only 1 c. per yard and the first cost of the equipment is little greater than for an ordinary 70-ton shovel operation requiring a full complement of locomotives, dump cars, rails, etc. The principal item of cost is the moving of so large a machine from one stripping operation to another, or to and from the railroad. If the distance is great or if hills intervene, the machine must be taken down and carried piecemeal to its new location and there set up. This represents no insuperable difficulty, however, and it is probable that the number of these machines in the anthracite region will gradually increase. It is reported that in Kansas ratios of 10 cu. yd. of cover to 1 ton of coal have been handled profitably by means of this equipment. Many areas remain to be stripped in the anthracite region, for the end of stripping operations is not yet in sight, and if costs can be reduced to the extent outlined above there will be many

areas not now considered as stripping propositions, that will be the big operations of the future.

Various methods now employed in other stripping and excavating operations could perhaps be used to advantage in anthracite stripping

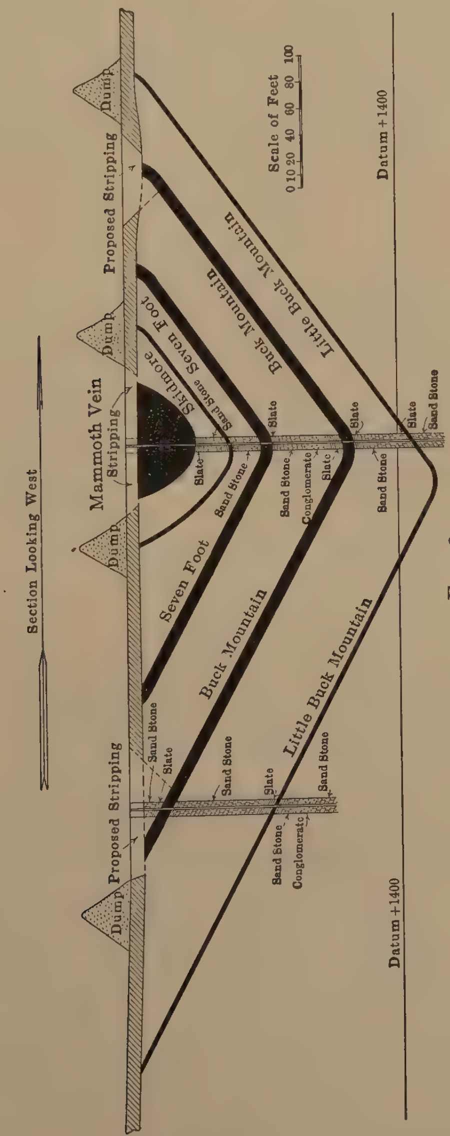


FIG. 9.

operations. Space, however, permits only the briefest mention of two of these.

Geared locomotives are practically unknown in the anthracite region.

There is a record of a stripping operation on an area of only 20 acres, where by the use of these locomotives a difference in elevation of 300 ft. between the bottom of the stripping and the top of the dump was overcome without passing outside the lines of the property.

Another possible improvement would be the use of hydraulicking methods. Scarcity of water and of areas on which to deposit and settle the refuse render this method in most sections of the region out of the question. Very low costs have been secured, however, in the West by this method and areas where it would prove feasible might be discovered. Where the refuse or spoil could be flushed into mine openings to support the surface a double value would be obtained.

Postscript

J. B. Warriner (communication to the Secretary*).—My attention has been called to the fact that in certain places in this paper my figures, which are merely average figures, might be constructed as including extreme limits. For instance rates for rock excavation are given as from $0.35 to $0.40, whereas rates as low as $0.32 and as high as $0.45 have been paid in isolated cases. Also, in the case of clay yardage excavated by one shovel in one month a maximum figure of 30,000 yd. is given. This represents a figure that is fairly frequently reached, but the extreme output recorded is 42,000 yd.

Discussion

S. A. Taylor, Pittsburgh, Pa.—What was the distance of haul?

J. B. Warriner.—There is practically no limit to the haul. The length is determined entirely by the feasible location for a dump. I have known of hauls up to 2 miles in length.

R. V. Norris, Wilkes-Barre, Pa.—This paper was originally presented to the Anthracite Section as the result of the appointment of Mr. Warriner as the Chairman of the committee to study this matter and present this paper. As he has stated, he obtained data very freely from all the stripping operations in the region, and it is a type of paper of which we should have more in the *Transactions*. Two or three things in it strike me as particularly valuable.

The two tables, 1 and 2, showing the coal remaining, have been the result of a great amount of study, as it is not an easy matter to obtain data as to the amount of coal left in mining. As an example, one of the anthracite mines was mined and robbed out in 1880, so the maps show. That mine is today producing 250,000 tons of coal, over 50 per cent. from the bed which on the maps is marked as exhausted. What actually happened is that the seam was on a light pitch inconvenient to mine.

* Received Apr. 25, 1917.

There was ample thickness of coal and the chambers were driven at a convenient pitch through this 30-ft. bed, leaving top and bottom, taking out about 10 ft. The pillars were removed and the working caved and marked as gone.

That is the type of area that is now being stripped, which explains the reason for the enormous amount of coal left in, and the fact that Mr. Warriner has worked out the actual results from a large number of strippings is of great interest to the operators in any region handling thick coal which has been mined over in the past generation. The modern mining, of course, is different. We do not have all the coal we want in big veins, and we do not mine it that way.

Another interesting feature of the paper to me was the study of losses. It is not uncommon to assume that after you have stripped off the overburden all the coal goes to market. Mr. Warriner has shown that there is a loss ranging up to 10 or 15 per cent. that does not go, so that estimates made on stripping, figuring on all the coal, are subject to this very material correction.

The use of planes has not been given the weight in the paper that the history of stripping in the anthracite region would seem to warrant. The great majority of the strippings have been in the very narrow, sharp basins of the Hazleton district, where it was practically impossible to get out graded tracks, and the greater part of the stripping has been done by planing. Overburden is removed by shovels and then pulled up on wire rope planes. The plane has a further advantage in that the coal is loaded directly into the mine car and taken into the original car to the breaker, so that there is no dumping between loading and final delivery. Every time anthracite coal is dropped the resulting breakage means more small sizes and therefore a less average price for the shipped material.

The tendency, of course, is to larger machinery and larger operations. It may be seen that Mr. Warriner has not given the cost per ton of coal. That is a figure which has but little value, and is generally unobtainable, at any rate until a stripping is completed, because the cost of moving overburden is confused with the total cost until the stripping is absolutely completed and there is no coal left. Further, the amount of overburden removed and the cost per ton is dependent upon the price received for the coal, because stripping will be continued as long as profitable and no longer, except in the few cases indicated, where stripping is a matter of safety for the mine or for the prevention of getting rock or waste into the mine.

George S. Rice, Washington, D. C.—I notice that special reference has been made to the stripping operations in the district where recovery of pillars in old mines has been attempted. I notice, however, that the author speaks of virgin fields, and in one or two illustrations, as for instance the last one on page 167, shows some illustrations of apparently un-

mined beds. I was wondering whether or not the factor of drainage of surface waters would afterward make the open excavations a detriment of any serious nature to future underground operations, so that perhaps it might be wise not to begin the stripping until after underground mining has ceased. My reason for that inquiry is because I have been in certain bituminous districts with pitching beds where the stripping of the outcrop forming canal-like excavations had later proved to be detrimental in subsequent underground mining operations down the dip. The water seeping through pillars and running down and into the mines caused continuous pumping operations.

J. B. Warriner.—That point, though a very important one, is not dwelt on particularly in the paper for the reason that in the anthracite regions early mining operations failed to take drainage into consideration at all. The early openings were entirely water-level openings and the workings were pushed through to the surface and water allowed to run in and find its own way out. Most all our strippings are of areas so worked over in earlier days, and, therefore, the stripping does not increase the drainage area materially. Even where the stripping is of virgin veins, there are usually other veins in the vicinity that have been worked through to the surface sufficiently to form an inlet for all drainage water, so that even such strippings do not materially increase the amount of water to be handled. It is regrettable that in the early mining operations such methods were used, as a chain pillar left under the surface, or even all the water-level coal left in place, would have greatly reduced the difficulties of present-day operation.

Harrison Souder, Cornwall, Pa.—This paper is extremely interesting to me, as our company is doing some work in the same line. Our business is mining iron ore but we have a problem of moving something like 3,000,000 yd. of stripping, practically all rock. I think the figures given here are quite valuable. I would like to ask Mr. Warriner about the operating hours. He gives the average monthly yardage here. Does that mean working one 10-hr. shift daily, or day and night?

J. B. Warriner.—On contract stripping work, one shift only of 9 hr. is worked daily.

H. Souder.—Name some tentative figures.

J. B. Warriner.—The rate of 45 c. mentioned is higher than the usual rate for rock on a classified basis. By a classified basis I refer to a contract that specifies one rate for clay excavation and another for rock excavation. The percentage of rock to clay in each stripping operation determines the average rate, or the unclassified rate, per yard.

H. Souder.—That puts the price pretty low, it seems to me.

J. B. Warriner.—Those are the rates that prevail in the anthracite region. We have stated the limits to be between 35 and 40 c. for rock, but these are only the average limits. There are some few exceptions, both higher and lower, that occur.

H. Souder.—Your suggestion of heavy equipment, I think, is in line with present practice not only up in the Mesabi country but in our own operations. We originally started with small, 65-ton shovels and now are putting in 100-ton shovels; started with a 5-yd. dump car and now have 20-yd. air dump cars. I notice you mention the height of the benches. That is a very important item in shovel work and we started originally with 16-ft. benches and are gradually working them up and now have adopted 40-ft. benches as the proper working height in rock.

J. B. Warriner.—The difficulty of that is blasting the rock so as to break it small enough to be handled by an 80-ton shovel.

H. Souder.—That is where the large equipment comes in.

J. B. Warriner.—In a 40-ft. bench the holes required for blasting must be nearly the same depth; naturally, therefore, only the lower part of the bench is broken up at all finely. The surface rock will be found in immense boulders after the blast and a great deal of labor and powder must be expended in redrilling and shooting these boulders. For that reason our contractors have come to consider the lower bench as more nearly ideal.

H. Souder.—We find that to be the case sometimes. Since the introduction of the jackhamer and larger equipment we have increased the output of a shovel I think something like 20 per cent.

J. B. Warriner.—Our contractors also use jackhamers.

H. Souder.—Our high bench makes for fewer moveups and bigger loads.

J. B. Warriner.—That is the usual argument in favor of the higher bench.

H. Souder.—Those are things which have been tried out in our operations at Cornwall and which have proven to be correct. Of course the tonnage moved at a fair cost is the thing that counts, in the long run.

Robert Peele, New York, N. Y.—When Mr. Norris spoke of the relation between overlying and wholly or partly worked-out veins and subsequent stripping operations, my thoughts reverted to a case in the bituminous regions that I saw some years ago, and although not quite pertinent to this subject I thought it might elicit some further discussion of the paper.

A seam lying horizontally and 8½ to 9 ft. thick was being worked.

It was overlaid by five other seams, included in a total height of 100 or 120 ft., and ranging in thickness from about 3½ to 5 ft. The mine was on the borders of the Pocahontas field, lying close to the line between West Virginia and Virginia. In working the main seam, the roof was allowed to cave, with the result that all the other seams were probably ruined so far as any future recovery of coal was concerned.

This case occurred to me also in connection with the remark made by Mr. Warriner himself, regarding the stripping of a series of seams, the upper ones of which had already been wholly or in part exhausted. It would seem as though the large tonnage of coal included in these thinner seams would be absolutely lost in the subsidence and breaking up of the overlying strata, in the caving following the working out of the main 9-ft. seam.

J. B. Warriner.—The point brought up by Mr. Peele is not a matter of particular importance in those sections of the anthracite region where stripping is mainly resorted to. The veins in those sections are highly inclined and can be mined to a large extent without disturbing the overlying veins, at least to the extent that they would be disturbed in flatter measures particularly in the bituminous region. In certain sections of the anthracite region, however, it is very important that the overlying veins should be worked off first before the underlying veins are mined at all and even in the regions of highly inclined strata some importance must also be attached to this point.

The Chairman (R. M. Catlin, Franklin Furnace, N. J.).—Increase of water due to the stripping has been touched on but I did not understand that it was considered much of a factor from the discussion here. Is that correct?

J. B. Warriner.—That is correct, because the watershed area is not increased appreciably by stripping for the reason that the previous, or early-day mining operations have already breached the surface to such an extent that stripping an area does not add much to the watershed.

Chairman Catlin.—I could imagine that in some cases where the surface had settled you might be able to divert a good deal of the surface water falling on that watershed.

J. B. Warriner.—That is true, and that is done in some cases, but it has not been an important point with us. The quantities of water handled in anthracite mining are enormous and I know of few instances where stripping in any area has greatly increased the total quantity to be handled by the mine.

Chairman Catlin.—Its chief danger is due to sudden water falls, for instance a very severe thunderstorm will crowd an enormous amount of water there temporarily.

J. B. Warriner.—Considerable money has been spent in constructing ditches outside the outcrops of our veins to prevent surface water in flood times from flowing into the crop falls or surface breaches. The same is also done with stripping areas where feasible.

T. M. Dodson, Bethlehem, Pa.—I have had some experience in stripping a large virgin vein—average thickness 65 to 75 ft. The depression caused by mining this vein would be so great that its effect, so far as to render impossible the drainage of the area, would be equivalent to the stripping process. In both instances the surface water would have to go into the mines. Of course, wherever possible, we ditched the periphery of the stripping and endeavored to prevent any outside water flowing into the pit.

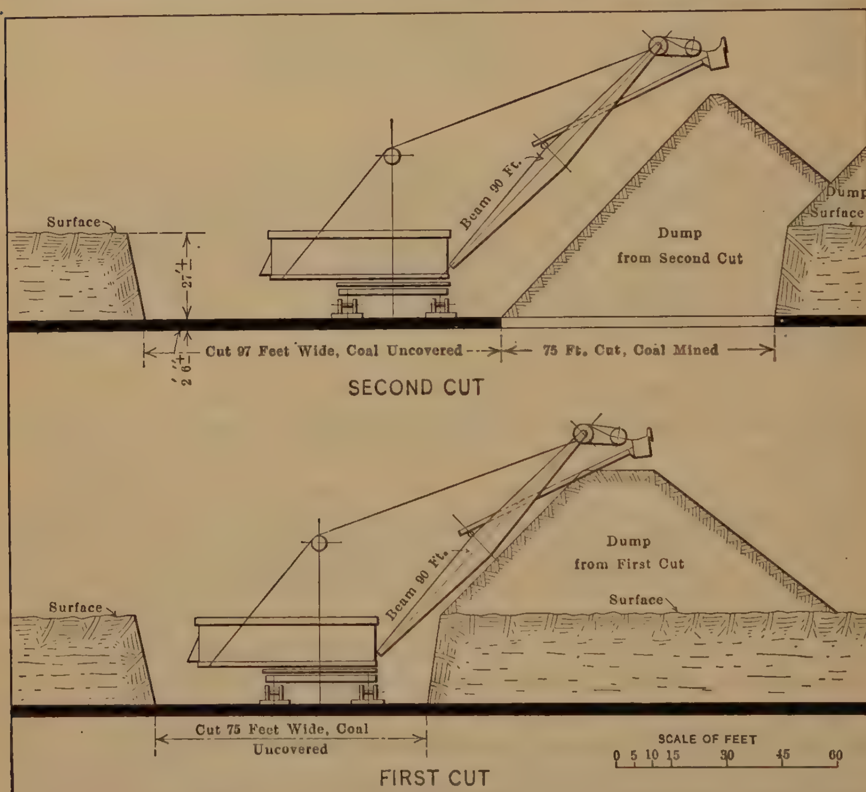
H. M. Crankshaw, Hazleton, Pa.—How far can the large shovels which are being used in the soft coal region be applied to mining and stripping in the anthracite regions? Some of these shovels now have 90-ft. booms, a 5-yd. dipper and can make a pile of refuse over 60 ft. high. It seems to me if these shovels go on increasing in size the anthracite stripping of the future might not have nearly so much transportation to contend with as it has now.

J. B. Warriner.—If you will turn to Fig. 9 you will see that that is what is being done where the 250-ton drag-line excavator that I speak of is being used. The drag line in that instance was able to make its dump outside of the stripping limits, as is shown very plainly in the illustration. The width of the basin being stripped is such that a drag-line excavator of this size can handle it in two cuts, each cut extending from one outside limit to the center and one bank being thrown off on one side and the other on the other side. When the stripping area is wider than is shown it can readily be seen that this cannot be done unless the coal is removed first, as is done in certain cases, I understand, in the Kansas field. If the coal is removed from an area that has already been stripped, then the spoil or the refuse material from the next cuts can be dumped into the area from which the coal has been removed, and in that way, by mining directly behind the stripping, it has been possible to strip very large areas in Middle Western practice, at a low cost.

T. M. Chance, Philadelphia, Pa.—The Pittsburgh region in Kansas, to which Mr. Warriner has just referred, was the first place, I understand, where they used really large self-stacking shovels. The machines in use about four years ago were not drag-line machines; they were simply shovels with a 5-yard dipper carried on a boom some 90 ft. long and stripping a cut 70 to 100 ft. wide. The coal, at the locality which I have in mind, was about 2 ft. 6 in. thick and the over-burden around 27 ft. in depth. This over-burden was composed of partially decomposed sedi-

mentary shales and sandstone, in no way similar to the hard sandstones and slates encountered in anthracite stripping.

The method of operating the stripping was to take a preliminary cut of any desired length and from 70 to 90 ft. in width, placing the spoil from this at the side of the pit and necessarily covering some coal that could not be recovered. The coal uncovered by this preliminary cut was then loaded up and a space left for the spoil from the next cut, approximately as shown in the accompanying sketch. The shovel was



operated on a pair of double tracks, about 29 ft. gage from outside rail to outside rail, this track being laid on top of the coal. By operating the shovel in this way the difficulty of an unstable track foundation in times of wet weather was largely obviated, as a good floor was always furnished to carry the load of the shovel and its equipment.

The costs of this method of stripping are exceedingly low, as no extensive transportation is required for the spoil resulting in removing the over-burden. On a basis of 27 ft. of stripping to recover 2 ft. 6 in. of coal I understand a cost as low as 5 c. was easily reached and could be

considered conservative. The principal difficulty encountered in this field was in loading the coal after it had been stripped by the shovel. If this work were done by hand it should not have cost much over 35 c. a ton to load, but a cost such as this would hardly be comparable to the low unit cost reached in the stripping of the over-burden. I understand that small power shovels were tried for lowering this loading cost but certain difficulties were encountered in their operation simultaneously with that of the large stripping shovel, and I believe that a form of drag-line loader was eventually worked out that largely eliminated these difficulties and greatly lessened the final cost of handling the coal itself.

J. B. Warriner.—The point I have tried to make is that drag-line machines cannot be used except under favorable conditions such as the one described in the paper or possibly where the stripping is of large enough area to make it profitable to remove the first cut only by this method. Such a machine, however, can be equipped with a boom and used as a shovel. That has not been tried out at all, but I believe it is contemplated, and I make the point that probably it will be tried out and probably it will be successful. It must be remembered that in our strippings the depth is variable and successive vertical cuts must be made. Most stripping pits have been excavated to a depth of 60 ft., and some even to 150 ft., from the original surface to the top of the coal.

The Chairman.—Where would the spoil go in such cases?

J. B. Warriner.—The spoil would have to be loaded onto cars and hauled out.

T. M. Chance.—Then you lose the advantage of a 90-ft. boom.

J. B. Warriner.—Yes, you lose that advantage, but I think the big dipper would be the chief advantage. You would still have the transportation problem to contend with in any anthracite stripping where those machines are used, except in a few cases where they can be used as the one described is being used. The machine described in this paper is carried on wooden rollers.

George S. Rice.—About ten years ago I was interested in phosphate mining in Florida. My attention was directed to a moving turntable which had recently been developed. This turntable was mounted on rail trucks, with long boom and a drag bucket, the drag engine being placed on the turntable. Upon my recommendation, my client, the Armour Fertilizer Co., decided to try it in mining phosphate. A plan was worked out for a certain tract adjacent to an extensive swamp, where the water level was in many cases higher than the bottom of the phosphate, by which the stripping of the overburden 5 to 16 ft. thick and the phosphate, 10 to 15 ft. thick, could be handled from one lateral position of the track and

moving turntable. The special track for the turntable was placed on top of the bed of phosphate on the edge of the bank, parallel with a channel cut in the phosphate. The boom which was first tried was 60 ft. in length and permitted the excavation of the overburden by the drag scraper, for a lateral width of about 50 ft. from edge of cut to top of bank. after filling the dredge bucket the boom was swung through 180°, by pull of the drag engine on the turntable, and the waste deposited in the pit from which the phosphate had been excavated. The pebble phosphate is mined by scraping up behind the excavator, swinging the boom and bucket through 90°, and loading on cars, or belt conveyors as the case may be, which run along the edge of the overburden cut. The excavator track being placed on top of the pebble phosphate, which is as firm as a bed of gravel, no difficulty is experienced even in wet weather in maintaining the track. The dredge bucket used is of a type that can be used under water, so that if the bottom of the phosphate lay below water level, it could still be mined, though of course not so efficiently. The boom first employed, as stated, was about 60 ft. in length, but subsequently, I understand, an 80-ft. boom was used.

Before the plant was completely installed I entered the service of the Bureau of Mines and have not visited it since. I understand, however, that it has been working successfully and with satisfactory costs, the overhead charges being relatively small.

A. O. Ihlseng, Joplin, Mo.—Reference has been made to the large 5-yd. shovels used in the Pittsburgh, Kan., field. The Marion Steam Shovel Co. has a 7-yd. and the Bucyrus Co. has an 8-yd. dipper operating in this field making a 40-ft. cut. I think they have been bringing the cost of mining the 3 ft. of coal underlying it down to a very small figure. With the large over-burden of 13 yd. of strip. per ton of coal mined, the stripping cost must be less than 5 c. a yard.

T. M. Chance.—Five cents a yard is a fair, safe estimate, with a 5-yd. dipper.

J. B. Warriner.—The Geological Survey Statistician gives 10 yd. of cover to 1 ton of coal as the maximum result for 1915.

A. O. Ihlseng.—This 8-yd. dipper was put in operation in March, 1916.

J. B. Warriner.—Even 10 yd. to 1 ton is more than double the results obtained in the anthracite region.

H. M. Chance, Philadelphia, Pa.—It is hardly fair discussion to compare methods of work done under conditions that are quite unlike those in the anthracite regions.

Conditions in the Kansas field, where more of this kind of work has been done than in any other district, are peculiar to that field and to some

small areas in the Western or Central Western coal fields, in that the formation overlying coal is usually a rather soft slate or shale with interbedded sandstones. The sandstones are true sandstones but are generally more or less disintegrated. I have seen one of these Marion shovels cut right through what looked like 2 ft. of solid sandstone. No shovel could do that in sandstones such as we have in the anthracite region. The cost figures are not applicable to conditions such as are present in the anthracite field. In these Western strippings it is customary at times to shake up the stuff with a few holes when the country rock is relatively hard but in many cases they take off the whole country rock covering without shooting.

In Kansas the coal is relatively thin, 2½ or 3½ ft. thick—very rarely 4 ft. thick—and that coal is worth, as a crude proposition, we will say 80 or 90 c. or $1 a ton at the outside. In the anthracite region the coal may be worth about $3 per ton in its crude form, so that in the anthracite region the material to be mined is worth perhaps three times as much as Kansas coal and it requires perhaps three, four or five times the kinetic energy to break up that rock and remove it as a stripping proposition.

It seems to me that it would seldom be possible to adopt the suggestion of Prof. Peele to attack bituminous coals by this method with the idea of working several coal beds that lie one above the other. I assume Prof. Peele had in mind the idea of stripping right from the top of a hill and taking vein after vein, thus winning the coal from several veins. That proposition has been suggested before but there are very few places where it could successfully be used for the reason that Nature has not generally placed our coal beds close enough together, the workable beds usually being separated by intervals of 30 to 60 or more ft.

If we adopt 10 cu. yd. as the maximum quantity of waste material that can be moved for each ton of coal recovered, that would require a coal bed 3 or more ft. thick to every 30 ft. of rock and there are few places where coal beds of this thickness are so close together with the top bed close to the surface.

W. S. Ayres, Hazleton, Pa.—I quite agree with Mr. Chance that we cannot, from the data given, get at the relative merits of the different systems of stripping referred to in this discussion. We have not taken into account all of the elements which enter into the problem. Not only the cost per cubic yard, but every possible phase that enters into the cost should be set forth. The nature of the material, ranging from soft drift material not requiring any blasting, to that which is very hard, such as sandstone or conglomerate rock, and which requires the most extensive use of powder, should be clearly and definitely stated. The length and system of haulage, the system of dumping, and whether the material when dumped does or does not adhere more or less to the bottom and sides of the car, should be stated. A very important feature is the thick-

ness and the quality of the vein of coal, and also the market value of the coal recovered. Without all of the facts before us we cannot get at an intelligent comparison, nor can we select from the systems described the one that will suit our particular problem best.

Years ago—15 or more—we had at Hazleton, Pa., a stripping which was considered at that time a very large one. The cross-section of it was almost identical with the cross-section of the Panama Canal at Gold Hill, and it was so deep—300 ft., the coal being 80 ft. thick at the bottom—that it was possible to remove a part of the rock by the milling system. We ran one section of the stripping by the steam-shovel system, using seven shovels, and the other by the milling system. The cost per yard by the milling system was 5 c. less than by the steam-shovel system. The milling system cost 17 c. and the steam-shovel system 23 c. per yard. We used at that time the 1½- and the 2-yd. dippers, which seemed to be about as large as we could use to an advantage on so deep a stripping as this. A larger machine would have enabled us to handle the large blocks of sandstone rock with less subsequent blasting. There were serious disadvantages seen in the larger machine at that time, and it was not considered as adapted to the conditions.

J. B. Warriner.—I recognize very clearly the truth of the remarks made by the two previous speakers. My idea is not that we can ever equal the Kansas results, but that our results at the present time when we are only doing about a third or a quarter as well as they do, are not highly creditable. If we better these results to do even one-half as well as they, we will have accomplished a great deal for the industry. This can be done only by the use of large-sized equipment, and that means that the equipment all the way through the stripping must be changed; wider gages and bigger dump cars must be used as well as larger shovels. The usual gage for stripping tracks at the present time is 36 in. and in certain cases 42 in., and the dump cars are of about 5 yd. capacity or even less. To use a shovel handling a 5- or 8-yd. dipper with small equipment of this kind would be out of the question. To get the full economic benefit out of the cost of such a machine it would be necessary to use 20-yd. dump cars at least and standard-gage tracks.

Chairman Catlin.—This matter of costs, I think, unless it is reduced to common denominator, is largely a state of mind. I believe comparisons between two localities widely separated are unsatisfactory, as conditions are probably very different.

D. B. Reger, Morgantown, West Va.—How does coal mined by stripping compare in quality with that mined by the ordinary mining method?

J. B. Warriner.—There is practically no difference in the quality of the coal itself. The quantity of refuse that must be excavated with

the coal, however, varies. Naturally, all the stratified refuse of the vein must be excavated with the coal; also nearly all strippings are of areas that have been previously worked over and the old breasts and chutes distributed through this area are filled with rock which has fallen from the roof. This rock, or refuse, is mixed with coal which has sloughed off the ribs of the opening, or perhaps a bottom fender, or a top bench of coal may have been left in. In the gradation zone between the virgin pillar and the old openings it is of course impossible to excavate with the shovel, or even by the chute method, and prevent contamination of the coal with refuse. The practice, therefore, is to make as clean a separation as possible and if cars contain too high a percentage of refuse to warrant sending through the breaker, to dump them elsewhere and to pick the coal out by hand. Where there is as high as 50 per cent. of coal in a car it is usually sent through the breaker.

B. F. Tillson, Franklin Furnace, N. J.—In extenuation of the Chairman's remarks, I might suggest it would be of much value, I think, to the members of the Institute if cost data could be placed on a basis of units of labor and units of material used in operation. I think it is of extreme importance to have our discussion on a basis of labor and material in order that we can compare it to our own practices.

Chairman Catlin.—That eliminates only part of the trouble. We still have the differences of local conditions to contend with and whether you take it in terms of dollars or terms of hours of labor, it is still the difference in local conditions that makes a wide divergence.

Portable Miners' Lamps

BY EDWIN M. CHANCE,* WILKES-BARRE, PA.

(New York Meeting, February, 1917)

DURING the past 10 years, the safe and efficient lighting of the coal mines of this country has received an ever-increasing amount of attention. Several States have passed laws attempting to regulate the type of lamp to be used and the nature of the fuel to be burned, and the mining departments of coal-mining States have generally shown a keen and intelligent interest in this subject. The passage of the recent Employers' Liability Act in Pennsylvania has made it necessary for many coal-mining companies to take out liability insurance, and the companies underwriting such insurance have made it desirable for the insured to permit the use of none but illuminants of established worth. While these conditions have not obtained for a sufficient length of time to permit the statement that the illumination of coal mines by portable lamps has been standardized, still, considerable progress has been made and the direction of future practice in this branch of coal-mining technology is very evident. Under these conditions it seemed that a review of the methods now used in coal-mine illumination, together with a brief consideration of the principles underlying these methods, might be of some interest.

Miners' lamps may be divided into three classes: The open light, the electric cap lamp, and the flame safety lamp. It will be desirable to consider each of these classes separately, as each has properties peculiar to itself and one class is hardly comparable with another.

In this country, without question, the open light is the most generally used of all miners' lamps. This fact is explained by the relative freedom of a large proportion of our mines from gaseous conditions, and by the admirable systems of ventilation installed in those that show a tendency toward such conditions. In the metal mines, the miner's candle has had and still has considerable vogue. Because of the small amount of ventilation usually supplied in metal mines, the freedom of the candle from any tendency to produce noxious gases or offensive odors, and the small amount of air it consumes, are valuable assets, while, because of the light color of the rocks in which workings are driven, the meager light is not a serious handicap.

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* Consulting Chemist and Engineer.

In coal mines, however, conditions are very different. Because of the considerable volume of air passed through the workings, the small air consumption of the candle is of little moment, and its feeble rays are so completely absorbed by the dusky background that its light is entirely inadequate. Moreover, the cost of the candle is relatively high. For these and other less obvious reasons the metal miner's candle is practically unknown to the coal miner.

Up to a few years ago the open oil lamp had no rival in non-gaseous mines and it is still very largely used, though its use is becoming limited, as will be shown later. The oil lamp has many disadvantages and the legislatures of a number of States have endeavored from time to time to remedy these defects by law. As the service given by an oil lamp varies largely according to the character of the oil, efforts have been made to fix by law the quality of the oil that may be sold to miners for use in these lamps. These efforts, unfortunately, have proven rather ineffectual, as the result has generally been to increase the cost of oil to the miner without increasing its quality in anything like the same proportion. These laws, therefore, may well be considered one of the most potent of the forces that have driven the oil lamp out of its once strong position.

An example of the manner in which a law, that was honestly intended to be beneficial to the miner, failed of its purpose is found in the Bituminous Mining Law of 1911 of the State of Pennsylvania. This law stipulated that oils sold for use in miners' lamps should not yield more than 0.11 per cent. of soot when burned in a miner's lamp under standard conditions. One of these conditions was that the flame of the lamp should be 1½ in. long. Now, low-grade oils when burned under these conditions yield as much as 1 per cent. of soot, while high-grade oils will give as little as 0.03 per cent. Thus it would seem, at first glance, that this law would considerably better conditions in the mines.

Such is not the case, however. Oils to pass this test must be very largely composed of costly fatty oils and this so greatly increased the cost to the miner that he was obliged to look for some cheaper illuminant. Moreover, instead of a flame 1½ in. long, the miner burns one of a maximum length because he wants as much light as he can get. The writer has found that while costly oils, containing high percentages of fatty ingredients, will produce much less soot than oils of medium price, and less fatty material, when burned under legal test conditions, these differences very largely disappear when these oils are burned under the conditions that obtain in the mines. With very long flames the high-priced oils still show a superiority to the medium grade, but the differential is so slight as to be of little real moment.

Indeed, the soot-forming propensities of both these oils under the conditions of use are so great that it is idle to attempt to classify one as

better than the other. They are both very bad. Thus with a legal requirement of 0.11 per cent. soot or lower, we find the oil passing this test will give about 8 per cent. of soot when burned as it would be in the mines—that is, with a flame 5 to 6 in. long—while the oil that will not pass the legal requirement, giving under test conditions, let us assume, 0.5 per cent. soot, will make under actual working conditions about 9 to 10 per cent. soot. Thus we may say that despite the greatly increased cost of the legal-test oil it is practically no better than many oils that may be secured at half or one-third the price. It is to be understood that many oils are of so low a grade as to be entirely unsuited for use in miners' lamps and, of course, these remarks do not apply to them. The point is that the tendency of many of the State laws is to increase the cost of oil very considerably to the miner and mine operator without proportionately improving its quality.

One of the drawbacks to the use of the open oil lamp is the greatly increased fire risk where such lamps are used in dry workings. It is necessary for the user of such a lamp to renew its wick or lamp cotton at frequent intervals, and it is customary to pull out the old cotton and insert the new while the old lies blazing on the ground. When the new cotton is in place and alight the miner places his heel on the blazing remains of the old, and perhaps extinguishes it; at any rate, he goes away and leaves it, to burn or not as may be. Another source of fire is the shower of sparks that is blown from the wick when the wearer of the open oil lamp is traveling against a strong ventilating current. Together these are the possible causes of mine fires that have disposed thoughtful mine operators to look with disfavor upon this source of illumination.

Of recent years a substitute for miner's oil, called "Miner's Wax" and a host of proprietary and brand names, has been placed upon the market. It is a paraffin wax obtained in the refining of petroleum and possesses the property of burning with a whiter flame than miner's oil and giving somewhat less soot. It must be used in a special lamp, however, as means must be provided for keeping it in a molten condition in the lamp fount. This is accomplished by conducting heat from the flame to the fount. Its use, though considerable, is decreasing because the fire hazard with this illuminant is as great as with miner's oil, and it is troublesome to handle.

Undoubtedly the greatest advance made in the illumination of non-gaseous mines is the acetylene or "carbide" miner's lamp. This lamp has come into general use during the past 7 years and is now probably the most widely used of miners' lights. The reasons for its popularity are not far to seek; in brief, it gives far more light than any other portable miner's lamp and costs less to operate. It gives a clear, white light in which objects have very much the same color value as in daylight. It

makes no smoke or soot and its demands on the oxygen of the mine air are moderate. It gives more reliable indications of the presence of dangerous proportions of black damp than the oil-fed flame. It gives off no sparks and hence decreases the fire hazard very considerably. It may thus be seen that this type of miners' lamp has benefits for the mine operator and the worker and is liked by both. The writer understands that insurance companies underwriting the insurance of many coal-mining companies under the new Pennsylvania Compensation Act have recognized the safety features of the acetylene miners' lamp by giving credits on the insurance rate where such lamps are used in non-gaseous mines.

Some years ago, considerable uneasiness was felt among mining men because it was thought that the acetylene lamp failed to give adequate warning of the presence of black damp. In the past, black damp had been believed, by many, to be an atmosphere in which a lamp would not burn, the reasoning being along these lines: If an oil lamp goes out, it is because there is not enough air (meaning oxygen). Now it is a fact that the acetylene lamp will burn where an oil lamp will not. Hence, if the oil lamp will not burn there is no air, and as the acetylene lamp continues to burn, this indicates that the acetylene lamp will burn without air. Therefore, a man may carry an acetylene lamp into an atmosphere containing so little air that he may be rendered unconscious, and still his lamp will give no indication of the dangerous condition of the atmosphere.

The facts of the matter are these: The oil-fed flame requires a minimum of about 17½ per cent. of oxygen for its maintenance; the acetylene flame requires about 12½ per cent.; and a man's life is endangered should the oxygen content fall much below 10 per cent. At about 14 per cent. of oxygen, however, the color of the acetylene flame changes markedly. It loses its brilliance and illuminating power, and becomes greatly elongated and unstable. From these data it will be seen that the miner is given obvious and adequate warning of the vitiation of the atmosphere through deficiency in oxygen. While this warning is not so peremptory as that given by the oil lamp, still it is of ample distinctness for men to appreciate and value, and above all, it is essentially a real danger warning.

On the other hand, the warning of the oil-fed flame is given with so high an oxygen content that miners have learned to disregard it, and will frequently go into workings containing air in which their oil lamps will not burn. They know that they can live in an atmosphere in which these lamps will not burn, but do not realize that, once in the dark, they have no further guide to the quality of the atmosphere and that in a few feet it may become lethal. Such is not the case with the acetylene lamp; its warnings are given so near to the danger point that men will have a wholesome respect for them, to the great increase of their own safety.

While the writer has heard of cases in which men after working with acetylene lamps in sections in which oil lamps would not burn became

sick when brought into fresh air, these always proved to be based upon a fallacy. Upon investigation, the fact has always developed that the disability of the men was due to other causes; too high a temperature of the workings or carbon monoxide being the most usual. In any case the presence of the disturbing agency would not have been detected by the use of oil lamps. Indeed, the writer has many times seen men fall like flies in a place they had considered safe, because their oil lamps gave no indication of anything abnormal in the composition of the atmosphere.

The supposed danger from the use of acetylene lamps where black damp may be encountered has caused their use in pillar work and in robbing to be discontinued to some extent, thus exchanging a danger of the imagination for a very real peril. Those familiar with the coal-mining industry know what a curse miner's asthma has been. If not caused, it is at least aggravated, by the greasy soot and oily emanations given off by the oil lamp. Now these foul vapors are at their worst in pillar work and in robbing where the ventilating current is apt to be at its lowest ebb and where black damp is obviously most likely to be encountered. Hence, as a result of trying to safeguard the miner against the hypothetical danger—of a combination of black damp, the acetylene lamp, and his own stupidity—we expose him to the very real and pressing danger of miner's asthma, against which the most careful and most thoughtful is helpless.

The deepening of our mines, and the increased length of airways, together with the greater attention now given to the danger of gas and dust explosions, have all tended to increase the number of safety lamps in use. Safety lamps may be divided into two classes: The electric cap lamp and the flame safety lamp.

The electric cap lamp is a development of the last 7 or 8 years and its growth has been very lucidly traced by J. T. Jennings in a paper read at the 1916 meeting of the Coal Mining Institute of America. It will, therefore, be unnecessary for the writer to go deeply into this matter. European practice has tended toward the development of an electric hand lantern. Because of the more general use of flame safety lamps there, their mine workers were satisfied with this type, being accustomed to the inconvenience of hand lamps.

In this country, however, conditions are radically different. The miners are, as a general rule, accustomed to open lights worn on the cap and rebel at the inconvenience and inefficiency accompanying the use of the electric hand lantern. Hence, when an electric miners' lamp was developed it became essential that it should be such that the efficiency and convenience of the cap lamp would be retained. Progress was made along these lines with the result that the present very convenient equipment has been developed.

It is remarkable that so complete a standardization in general design

as is now found among the product of the numerous manufacturers of this type of lamp should have been possible. The credit for this should be given without stint to the Bureau of Mines. This bureau has worked hard and faithfully with the manufacturers of electric cap lamps for the past 4 years or so, and has had a very definite vision of what such a lamp should be. The result of this pre-natal influence is a startling similarity in the various equipments offered to the mining industry. While this method has perhaps sacrificed a little individuality, it has undoubtedly increased the average excellence of the product, and made the whole industry more robust by weeding out the abnormalities.

Technical literature has been so full of descriptions of various types of miners' electric cap lamps that it will not be necessary to describe any in detail; it will suffice to touch upon the general advantages and disadvantages of this type of lamp.

Many of the underwriters of insurance under the Employers' Liability Act have shown a marked preference for the electric cap lamp when compared with the flame safety lamp. This preference has led to the penalization of companies using the flame safety lamp, to the extent of 11 c. for each $100 of pay roll, whereas, were electric cap lamps installed, this penalty would be wholly removed. This premium has led to the installation of many electric cap lamps.

This lamp throws its light into the plane of vision of the wearer so that its light is efficiently utilized. It leaves the miner's hands free and the light requires no attention; indeed, the outfits are so arranged and locked that it is impossible for the wearer or any unauthorized person to tamper with them. There is no fire risk with these lamps and we are assured by the Bureau of Mines that the danger of their originating a gas explosion is practically nil. These are also the safest of all lamps with which to handle explosives. They have three chief disadvantages: Their upkeep is high, the flux of light they furnish is not so great as it should be, and their wearer is in absolute ignorance, so far as the lamp enters into the matter, of the nature of the atmosphere surrounding him. It is very probable that the next few years will see these faults abated to a considerable extent.

That the flame safety lamp should be so old, so widely used, and so little improved in all its years of service reflects but scant credit upon the human mind. To all intents and purposes we still have Sir Humphrey Davy's invention in actual use, development having practically stopped after introducing the use of a cylinder glass to surround the flame. In this country the bonneted Clanny and Wolf type are practically standard except where heavily pitching veins are encountered, as in the Schuylkill anthracite district. Here, because of its lightness and the convenience with which it is handled on the pitches, the old Newcastle Davy is still supreme.

The flame safety lamp is principally handicapped by the meager light it gives. Its use is absolutely essential as an indicator of the quality of the mine air in all gaseous mines, and the writer would suggest that even in non-gaseous mines, where electric cap lamps have superseded open lights, the installation of a number of flame safety lamps would be an added safeguard. The principal difficulty in the use of the flame safety lamp as an indicator of atmospheric conditions, where electric cap lamps are relied upon for illumination, is that unless the flame lamp is the source of light it will come to hang unnoticed on the miner's belt or be left neglected in a corner of his working place. In other words, unless his attention is automatically called to it from hour to hour, he will, in the long run, cease to note its warnings.

While the variations of the Davy principle on which flame safety lamps are constructed have been endless, these variations have been slight, and the principle has not been diverged from with any success. As a result, the flame safety lamps in actual use in this country are of two very similar types, the Clanny and the Wolf, as has been mentioned above, ignoring the Davy lamps used in the southern anthracite district of Pennsylvania, for these will soon disappear.

With the design of the lamp fixed, we have but one variable to consider and that is the fuel burned. Even this disappears in the case of the Wolf lamps, as these will operate only with naphtha whose composition may vary within but narrow limits. With the Clanny type lamp, however, a wide variation in the nature of the fuels is possible. Among these are sperm, peanut, lard, rape, seal, cotton seed and mixtures of these with mineral burning oils. The writer has spent much time in investigating the question of improving the quality of safety-lamp oils and has met with some success in this direction. It has been found that some of the most costly and highly prized safety-lamp oils were really inferior to mixtures containing high proportions of high-grade mineral burning oils. These mixtures burn with a whiter flame, give appreciably more light, do not crust the wick, and are much cheaper than the standard safety-lamp oils.

There is another type of lamp that holds out promise for the future. This is the acetylene safety lamp invented by T. M. Chance of Philadelphia and described in the discussion of R. P. Burrows' paper on coal-mine illumination.[1] As this lamp is not yet a commercial fact, but little can be said of it definitely. It would seem, however, that it combines the safety and indispensable gas-detecting properties of the flame safety lamp with many times the illuminating power of the electric cap lamp.

A table is appended containing data that may make more intelligible some of the statements in this paper. These data have been accumulated

[1] *Trans.* (1916), **54**, 34.

during the past eight years and are general averages. The photometric determinations were made upon a United Gas Improvement Co. 60-in. bar photometer. The photometric standards used were 10-volt tungsten lamps, prepared and calibrated by the National Lamp Works, and standard sperm candles. At times a secondary standard was used, consisting of a long-time kerosene burner, similar to that used in railway signal practice, standardized against one of the primary standards noted above.

It will be noted that no estimate of the cost per day of electric cap or flame safety lamps is given in the table. In the writer's opinion, the modern electric cap lamp has not been in use long enough for an intelligent opinion of its upkeep cost to be formed. Moreover, the labor charge on both the electric cap and flame safety lamps is so large and varies so much with the size of the installation that such figures as could be given would have but little meaning.

Conclusions

1. The open oil lamp has outlived its general usefulness. It still has a field, however, in special cases, such as those of drivers, motormen, trip runners and the like, who are obliged to work in swift air currents.

2. The use of the open acetylene lamp is growing in all non-gaseous mines because of its cheapness, the powerful light it gives, its reliability in the presence of black damp, and its freedom from soot and noxious vapors.

3. The electric cap lamp is best adapted to use in gaseous mines. Its flux of light is superior to that of the flame safety lamps now in use. Moreover, as it leaves the hands free it is more convenient than the flame safety lamp. Under especially drastic conditions in non-gaseous mines, where the fire risk is unreasonably high due to peculiar local conditions, its freedom from fire hazard recommends its use. In all gaseous mines its use must be accompanied by that of flame safety lamps, in order that the condition of the atmosphere may at all times be known in all parts of the mine. Where it entirely replaces open lights in non-gaseous mines it is imperative that a few flame safety lamps be supplied along with the electric cap lamps for the same purpose.

4. The flame safety lamp should be used in all gaseous mines irrespective of the use of electric cap lamps. These lamps form in themselves the best and most trustworthy gas detector yet devised and their presence is absolutely essential to the safe operation of such a mine. Even in non-gaseous mines, where electric cap lamps have replaced open lights completely, it is good policy to have a liberal proportion of flame safety lamps. The flame safety lamp is doubtless an unsatisfactory working light, but its rôle is twofold. It may be used as a working light, but it must be used as a gage of the safety of the mine atmosphere.

5. There is a new acetylene safety lamp that gives promise of being an admirable working lamp as well as an excellent fire damp and black damp detector.

Candlepower of Various Types of Portable Miners' Lamps

	Candlepower	Cost per Shift, Cents
Miner's open oil cap lamp	1.50	2.4
Miner's open acetylene cap lamp	5.00	1.5
Electric cap lamp	1.10	
Davy safety lamp	0.12	
Clanny safety lamp	0.35	
Wolf-type safety lamp	0.65	
Akroyd and Best safety lamp	1.10	
T. M. Chance acetylene safety lamp	3.80	

NOTE.—The above candlepowers are in no sense maximum but are the average values over the field illuminated by the lamp in question and have been obtained from many determinations. These are the values that may be expected to be realized in practice under working conditions.

DISCUSSION

HERBERT M. WILSON, Pittsburgh, Pa. (written discussion).—Permit me to endorse the author's conclusions and their form of presentation as being, in my judgment, the last word on the subject of lights for the illumination of coal mines. I also feel that the open-flame oil lamp has outlived its usefulness, and that it should go, in the interests not only of safety but of efficiency.

Where an open-flame lamp can be used with safety in non-gaseous mines, the acetylene light adds materially to efficiency by its better illumination, and to safety for the numerous reasons given by Mr. Chance. The latter, however, lays chief stress on the value of the acetylene because of its non-fouling and non-obscuring effect on the atmosphere. Another important advantage over any other form of illumination is the clear, bright light projected at the point where the workman may look for danger. The better illumination materially safeguards the worker against obstacles over which he may stumble when traveling, and against dangers from falls of roof or of coal, weakness in timbering, and similar conditions. The acetylene lamp, where it may be used with safety, has advantages even over the present electric cap lamp, in that it gives a higher luminosity, and when the miner comes to learn its signals, as he will in time, just as he learns those given by the open-flame or the safety lamp, he will find that the acetylene lamp is a fair detector not only of methane but of carbon dioxide, as clearly shown by Mr. Chance.

The only serious objection raised against the use of carbide lamps has

been recently in Pennsylvania, due to the belief of the State Mine Inspection Department that it endangered miners working on pillar work, or other places where carbon dioxide might accumulate, or where there is lack of sufficient ventilation, and because it was believed that the indications of the carbide lamp would not give the miner sufficient warning. On this point Mr. Chance has clearly shown the fallacies of the objections. I may add that for those unable to detect carbon dioxide with the acetylene lamp, the maximum of protection would be afforded by using an open oil lamp or a flame safety lamp in such places; or, because the miner might not watch his oil light, a better protection still would be offered by a sufficient number of daily inspections of such working places by a fire boss or other official, with flame safety lamp.

I agree with Mr. Chance that in gaseous mines the electric cap lamp as now perfected is superior to any other form of illumination. The workers using it must, however, be protected against the accumulations of explosive quantities of gas by sufficiently frequent testing with a flame safety lamp by a mine official. With this protection, the electric cap lamp gives not only greater efficiency and greater protection, because of its more clear illumination, than the flame safety lamp, but it also renders the user far more safe from the possibility of ignition of gas. I have knowledge of a number of gas flares or explosions occurring within the last few years, when the only possible source of ignition appears to have been a defective flame safety lamp. Time and again I have had brought to my attention defective safety lamps used by men unskilled in their use, which lamps were, in some cases, in the presence of moderate quantities of gas, and which were in such a condition that the flame would have ignited an explosive mixture of gas. This could not happen with an electric cap lamp.

The States of Kentucky, Kansas and Colorado are in advance of some others in respect to the use of open-flame oil lamps, which are prohibited in the former State. I hope that similar prohibition may ere long be made in Pennsylvania, Illinois, Indiana and other States. Like Mr. Chance, however, I agree that the effects of legislation in respect to mine lights has not always been successful, and that the safety measures and the favorable premium rates based thereon adopted by casualty insurance companies under Workmen's Compensation, have been most effective in improving the illumination in mines in the last year or two. A recent ruling by the Industrial Board of Indiana provides, under the Indiana Workmen's Compensation Act, that "No compensation shall be allowed for an injury or death due to the employee's wilful failure or refusal to use a safety appliance, or performing a duty required by statute," and the Board holds that refusal to use electric safety lamps comes under the provisions of that section of the Act.

M. D. COOPER, Curtisville, Pa. (written discussion).—The interesting paper contributed by Mr. Chance on Portable Miners' Lamps brings out several valuable points. More particularly, his clear explanation of the comparative values of oil and acetylene lamps as regards the indication of black damp ought to go far in the direction of removing a rather widely held notion prejudicial to the acetylene lamp. Mr. Chance clearly demonstrates the safety features of the acetylene lamp.

The author's discussion of the work of the Bureau of Mines in bringing about a remarkable similarity in construction of the American types of electric cap lamps is well made. In fact, some of us have come to believe that there is less importance attached to the particular type selected from among the several well-known makes than to the proper care and handling of the lamps after they are placed in service. Careful study of the causes of lamp failure has revealed the fact that most of these may be traced not to defects in the lamp, but to carelessness in handling and use.

There are two general causes of the failures of electric cap lamps. For one, the lamp-house attendant is responsible; for the other, the miners are to blame. Care in manufacture may be speedily offset by lack of attention or ignorance on the part of the lamp man. Therefore, it is essential that a competent and careful person be placed in charge of the lamp-house. I know of a case where the results obtained in the use of electric cap lamps at two adjacent mines operated by the same company and similarly equipped, were satisfactory in one, while there were constant failures at the other. The difference was due entirely to the lack of a skilled and careful attendant at the mine where the failures occurred.

The miner himself, or the user of the lamp, is largely to blame for the loss of time resulting from the failure of the lamp to give an adequate light throughout the shift. At best, mine conditions offer a severe test of the construction of the lamp. But the miner has, for instance, a popular habit of removing his battery from his belt and carrying the apparatus by means of the cable. Such practices as this must be stopped before entirely satisfactory results can be expected. But with proper treatment both in the lamp-house and in the mine, it has been found that very good service may be obtained from the electric cap lamp.

ROBERT P. BURROWS, Cleveland, Ohio (written discussion).—Mr. Chance has given a very interesting review of the portable lighting situation. To one not directly connected with the daily operation and application of the various types of portable lamps, Mr. Chance's data seem to be very practical and are undoubtedly of considerable value in considering the question of such lighting equipment. The opinions of the miner, together with practice data relating to the operation of the

apparatus, will always be the largest factor in the final decision. It is true, however, that the more fundamentally scientific analysis of this problem may contribute much.

Mr. Chance has given much practical information, but has not attempted to give complete technical data nor comparative costs of operation. This information would be of considerable value, it seems to me, and worth much further work. The data Mr. Chance has given, of course, do not completely specify the illuminating power of the lamps. His method of test must assume the limiting angles of useful field and the relative value of the light at various angles in that field. Unless we come to some standard method of comparison, involving both the question of light output and cost of operation, I am afraid that many of us will be confused. For instance, the total light emitted from one type of cap lamp may probably be higher than the total light emitted from another of different light distribution. If, however, the two are compared on the basis of a working plane, the efficiency of operation might show to the disadvantage of the one giving the greatest total light. This may be due to the variance of the distribution of light from the two different units.

In view of the importance of this class of lighting and the various types of equipment, perhaps it would be possible, through coöperation of the large mining interests, together with manufacturers of portable mine lamps, and laboratories, where all the necessary equipment is available, to obtain data that will give such final results as cost per unit of light on some standard basis, or the cost per total unit of light emitted from the various equipments, including such items as upkeep and depreciation, interest, etc., which are not now considered.

I am sure there are manufacturers or groups of manufacturers who are sufficiently interested in this subject to be willing to furnish such photometric and comparative candlepower information as they could, and to obtain from large mining interests such average costs of operation as could be obtained from large installations. These data when carefully analyzed would undoubtedly be of considerable value to everyone concerned.

H. H. Clark, Washington, D. C. (written discussion).—The table of relative candlepowers and costs of operation is especially interesting. The electric cap lamp has been in general service for so short a time that accurate data with respect to the cost of operation are not available. I believe, however, that some of the lamps can be operated at a cost of less than 2 c. per shift. I will just mention in passing that operating costs may be computed upon so many different bases that a statement of cost operation has no absolute value unless accompanied by data with reference to the basis of computation, and so far as I know there is no standard basis of computation.

I have always been interested in statements of safety lamp candlepower and have found that they do not always agree with one another. The candlepower will vary with the flame height, the fuel, and the basis of computing the candlepower; *i.e.*, whether it is mean horizontal, maximum in one direction, or average over the entire illuminated zone. In tests that I have been interested in we have found considerable difficulty in checking our own results and in checking the results of other investigators. We have not yet done any very extensive work with respect to the candlepower of flame safety lamps, but some time we hope to, and then we plan to measure the total light produced by the lamps after standardizing very carefully the height of the flame and the quality of the fuel.

Some time ago I made some measurements of the average candlepower over the light stream of Wolf-type lamps, using as a fuel naphtha having a specific gravity of about 0.70, and trimming the flame to a height of 1 in. The average intensity of the light stream under these conditions was determined as between 0.4 and 0.5 candle-power.

More recently, some tests have been made with the Ackroyd & Best lamp, burning mineral seal 300° oil with the flame trimmed to 1 in. in height. Under these conditions the average candlepower over the stream of light was determined as between 0.5 and 0.6 cp. It is hardly likely that the discrepancy between these results and those given by Mr. Chance is entirely due to personal equation or to inaccuracies of observation, but rather to conditions under which the tests were made.

It seems to me that the best way to compare the light-producing capacity of flame safety lamps is to compare the total amount of light that such lamps give, and this is most easily done by comparing the total light flux expressed in lumens. While some may balk at the new term when candlepower has been used for so long, I believe that it is the proper term to use, and, moreover, believe it will be used more and more as time goes on. Makers of incandescent electric lamp bulbs are urging the adoption of the lumen unit for the comparison of the product, and for this reason I believe that in a few years we shall be comparing no longer by candlepower but by lumen ratings.

E. M. Chance.— Mr. Clark's figure of 0.4 cp. for the Wolf type is interesting, and I think I know where the discrepancy, if any, exists. In the first place I am decidedly skeptical as to whether the integrating sphere is applicable for use in photometering flame safety lamps due to the fact that a very slight fouling of the air from lack of ventilation causes an enormous drop in candlepower. I am not quite sure whether I am quoting correctly, but my impression is that it has been shown with ordinary lights the candlepower falls off 3½ per cent. for each 0.1 per cent. decrease in oxygen.

Mr. Clark's data are exact, precise and of unquestioned accuracy, as is the case with all the work that he has done, as I can bear witness. The figures that I quoted were based on approximations. This paper is in a sense non-technical and the average mine superintendent or mine foreman or mine engineer, for whom I wrote that paper, is not interested in the point Mr. Burrows brought up, that one electric lamp will give 10 per cent. more light than the other. My figures on the Wolf type are practically maximum; that is, the lamps were driven at as great a light intensity as it was possible for me to secure from the lamps. They are undoubtedly far in excess of what is obtained in practical work. I do not doubt that the ordinary Wolf-type lamps so widely used in bituminous regions of Pennsylvania, under working conditions, will give not to exceed 0.3 cp. The Davy lamp will probably not average, in the hands of the miner, above 0.05 candle-power.

The mining industry has been so educated by advertising literature to look for excessive candlepowers in safety lamps that it was desirable, to my mind, to bring the matter down to a basis more closely allied to fact. With reference to the question that Mr. Clark raised regarding cost, I do not think we can quote costs. We can give instances of what certain lamps cost in certain mines. It is certain that with the electric miners' lamp we are in ignorance of what costs will be and will remain in ignorance for a matter of 18 months or 2 years.

G. S. Rice, Washington, D. C.—Is your test made with a round-wick Wolf?

E. M. Chance.—Yes.

G. S. Rice.—And the flame is perhaps more than an inch in height?

E. M. Chance.—Yes. The miners ordinarily carry their flame as high as they can, especially with a Davy lamp or a lamp burning a compound fat oil. They hang the lamp up and the flame gradually decreases in height, and if they start with an enormously high flame, they do not have to attend to it so often. It is the height of inefficiency and smokes the lamp badly, but that is the way they do it.

The fact should be borne in mind that a very slight difference in the height of the flame of a safety lamp when being photometered will make a great difference in the values found. This probably explains the discrepancies between Mr. Clark's values and mine. He has carefully measured the candlepowers given when the flame was exactly 1 in. high, while I measured the candle powers given when the flames were burned at the height that my experience had taught me is usual in practice.

E. C. Lee, Pittsburgh, Pa. (communication to the Secretary*).—I would like to add a word to the arguments presented by Mr. Chance

* Received April 7, 1917.

and others regarding the superiority of carbide lamps over open-flame oil lamps.

The chief argument by advocates of oil lamps, is their sensibility to the presence of black damp. This is entirely correct. The oil lamp is sensitive to the presence of black damp, so sensitive, in fact, that it goes out in the presence of a low percentage of black damp. The question in my mind is whether the oil lamp, by going out in the presence of a low percentage of black damp, is more conducive to the safety of the miner than a carbide lamp, which indicates the presence of black damp to those who have been instructed as to its behavior in the presence of black damp; but which does not go out until the proportion of black damp present is considerably higher than that required to extinguish an oil lamp, while still being lower than the percentage which is dangerous to men. Personally, I would much prefer having a lamp that I know would go out before the atmosphere became too contaminated to support life, rather than have my light go out in the presence of the relatively small amount of black damp necessary to extinguish an oil lamp. I believe that any man who has had the experience of groping his way along the rail after his lamp has been extinguished by black damp will agree with me in this statement.

Edwin M. Chance (communication to the Secretary†).—The subject of miners' lamps has long interested me and it would seem that the thought that has been given to it by the Bureau of Mines, and the users and the manufacturers of these lamps, is about to bear rich fruit. Not only will the solution of this perplexing problem mean greater safety and comfort for the mine worker, but it will also mean decreased rates of insurance because of this increased safety. There should also be induced greater efficiency of the individual worker, with accompanying greater output per man. Under the present labor conditions and those of the immediate future, this is all-important. The attainment of these results can be looked for in the immediate future with confidence.

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† Received Apr. 20, 1917.

Shot-firing in Bituminous Mines

BY M. D. COOPER,* E. M., CURTISVILLE, PA.

(New York Meeting, February, 1917)

For the purpose of obtaining some first-hand data in regard to the shooting down of coal in bituminous mines, it was the writer's good fortune to be employed as a shot-firer for almost one year. In all, 6020 shots were fired by him during this period.

All of the work was done in what is known as the South Main section of the No. 2 Mine of the Ellsworth Collieries Co., Ellsworth, Pa., a subsidiary of the Lackawanna Steel Co.

Position of Shot-firing in the Cycle of Operations

The established practice of the Ellsworth Collieries Co. calls first for the undercutting of the coal by electric chain machines across the face of the entry or room. This is followed by replacing any posts removed by the machine-runners, or the setting of new posts or timber, as may be required. After this has been done the machine cuttings are loaded into cars. Then the coal is spragged, and a shot hole drilled to the depth of the undercut. Next in order, the shot-firer loads the hole and fires the charge. The working place is then cleaned up, timbered if necessary, the sides and face squared up, and loose material taken down. The face is again ready to be undercut.

Materials Used

Permissible explosives were used exclusively in blasting coal. These were Carbonite No. 2, a nitroglycerine explosive; and three different makes of ammonium nitrate explosives, Tunnelite, Red H, and Mine-ite 5-D. The last-named was used during the greater part of the period. It is interesting to note that even at the beginning of this work, the gases resulting from the explosions of the charges caused no especial distress on the shot-firer's part and this continued to be the case except on the occasions when a change was made from one class of explosive to another.

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* Assistant Superintendent, Ford Collieries Co.

At these times the gases had an unpleasant, but not at all serious, effect for perhaps a day. The explosive was brought into the mine by the miners, each man being required to have a small wooden box for this purpose. The box was equipped with a grooved cover, and was of sufficient size to contain nine cartridges, each 1¼ in. in diameter and 8 in. long. Without a box, the miner was refused a supply of explosives.

For the breaking up of an occasional rock fall in an entry, straight 40 per cent. nitroglycerine dynamite was used.

All of the charges, both of permissible explosives and dynamite, were detonated by du Pont No. 6 electric blasting caps, equipped with 6-ft. iron wires. The uniformity of results obtained from these caps will be

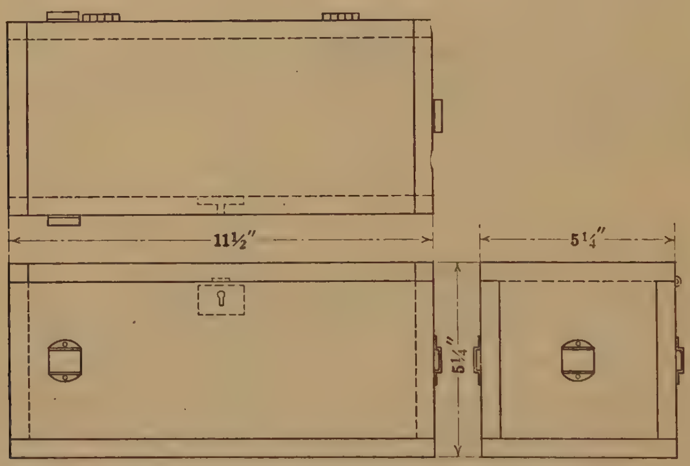


Fig. 1.—Box for Carrying Detonators.

commented upon later in connection with the subject of misfires. Formerly, the miners were required to carry into the mine their own detonators, inclosed in small locked boxes that held five or six caps. This was an insufficient precaution, however, against danger from the acts of careless or inexperienced men. It was found that the new men sometimes stored explosives and detonators in the same box, and in some cases attempted to load their own shot holes before the arrival of the shot-firer. To eliminate all dangers that could result from these causes, it was decided to have all of the detonators handled by the shot-firers only. Consequently, provision was made for issuing checks to the miners in place of caps. When the miners called at the supply magazine to purchase their daily stock of caps, they were given checks, each good for one cap. One of the checks was in turn given to the shot-firer each time he fired a charge for the miner.

A convenient type of box for carrying electric blasting caps is shown in Fig. 1. This is the type used by the Ford Collieries Co., and is patterned after the Ellsworth box. It is constructed of ½-in. boards, is varnished inside and out, fitted with a hinged cover and lock and with small sheet-metal slots through which a shoulder strap is run. The dimensions are such that a du Pont paper box containing 50 exploders will fit snugly inside. Each shot-firer is supplied with two boxes, one being left with the supply clerk to be filled, while the other is carried into the mine for

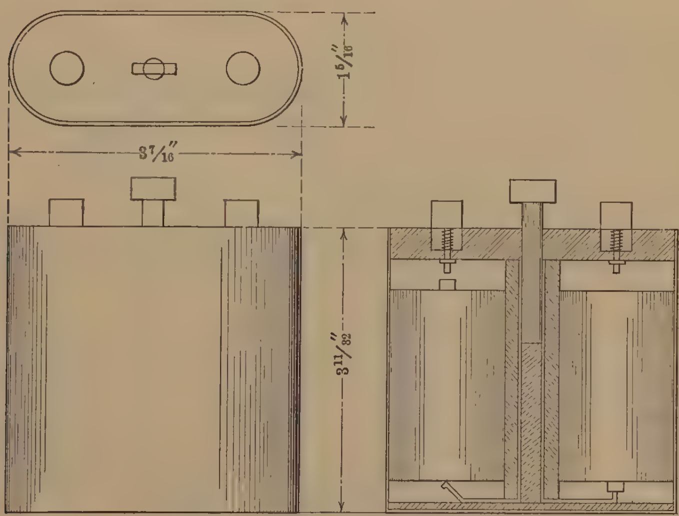


Fig. 2.—Shot-firer's Battery.

use. At the end of the shift, the shot-firer returns the box and a miner's cap check for each detonator taken out of it during the shift.

The lead wire used was about 110 ft. in length, of No. 14 B. & S. gage duplex copper wire. This length was found ample for safety. However, from time to time a short length is apt to be broken from the end nearer the charge and to replace this loss a corresponding length must be spliced on. This may be neglected in some cases, as there is a tendency on the part of shot-firers to carry no more wire than is absolutely necessary. Here is a point of danger, often overlooked, that may well be given attention by safety inspectors.

The battery used consisted of two dry cells inclosed in a suitable container, so arranged that the contacts were made inside and the danger of accidental contact avoided. At the time the writer took up this work, the battery in use consisted of two standard dry cells fitted into a wooden box made of ½-in. boards. This proved unnecessarily heavy and inefficient. Therefore, Andrew Boland, Chief Electrician of the Ellsworth Collieries

Co., was requested to have a new battery made, to hold two small cells of the type usually employed in flash lamps. He devised the battery illustrated in Fig. 2. This type of battery was found to be especially convenient, for it could be carried in the pocket. Moreover, it was doubly safe because before the circuit was completed it was necessary both to insert the key and press down the two contact buttons.

Clay was used for tamping all shot holes. The usual procedure required that the shot-firer observe the quantity on hand in his section of the mine. When this fell below a sufficient amount the mine foreman was notified. The latter had a mine carload sent in by the night shift to the point designated by the shot-firer. The clay was usually unloaded in a break-through closed by a stopping and conveniently located to a number of working places. Here the miner obtained his individual supply. If the clay proved too dry, enough water was added to make it plastic, in order that it might be worked by hand into roughly cylindrical masses about 1 in. in diameter and 6 or 7 in. in length.

In the case of rock falls, the explosive was as a rule placed on the upper surface of the piece to be broken, and then a shovelful of rather wet clay was packed over the charge before firing.

The miners were all required to provide a tamping rod. This was in all cases obtained by them from among the young trees in nearby woods. It was 6 ft. long and 1 in. in diameter.

In addition to the materials described above and used in the firing of shots, it was necessary, of course, to provide each shot-firer with an approved flame safety lamp, in order to enable him to comply with the law and the rules of the mine, requiring him to test for gas.

Method of Firing

The first step to be taken in an effort to safeguard the firing of shots is the selection of suitable shot-firers. The Bituminous Mine Law of Pennsylvania provides that in the portions of a mine where locked safety lamps are used, "the mine foreman shall employ a sufficient number of competent persons, who are able to speak the English language, to act as shot-firers, whose duty shall be to charge, tamp, and fire all holes properly placed by the miners, and to refuse to charge any holes not properly placed."

To meet these requirements fully, it is essential that the men selected for this work be of the sort who appreciate the responsibility placed upon them, who understand the proper use of explosives, and who are willing and able at all times to keep constantly in mind the fact that the reason for the creation of their positions is the prevention of accidents. It has been found desirable to have men with fire-boss certificates employed as shot-firers, if it is possible to obtain them. There are two good reasons

for this arrangement. First, a man with the certificate is generally competent to undertake the work. Second, if the regular fire boss is unable for any reason to make his run, a man is available who is thoroughly familiar with the regular fire boss's territory, and is able to go in and make the examinations as required by law.

The number of shots that may be fired in one shift depends upon the extent of the section covered, the proportion of narrow places to wide places, and the experience and ability of the shot-firer. Under average conditions, a shot-firer ought to be able to load and fire between 40 and 50

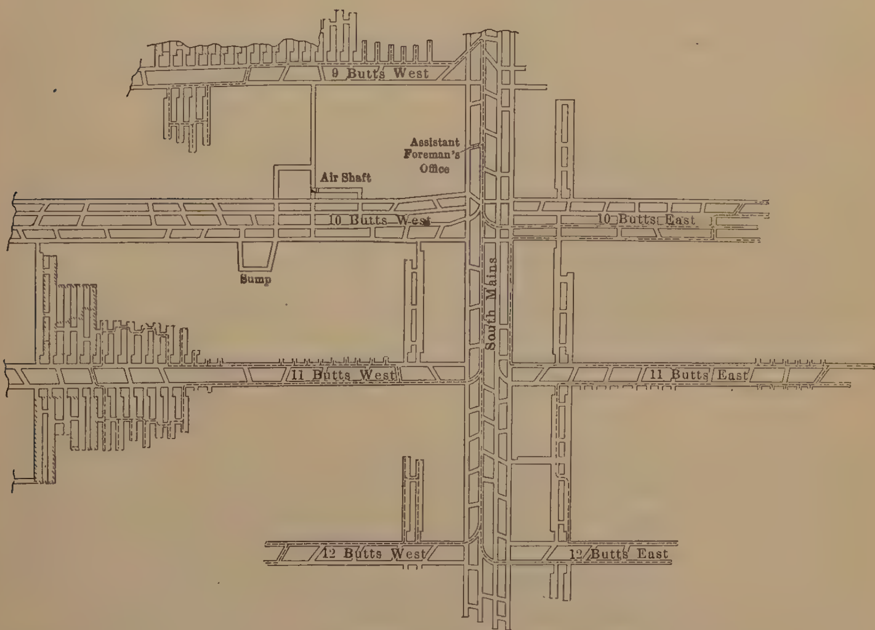


Fig. 3.—Part of Map of Ellsworth No. 2 Mine, Showing South Main Section.

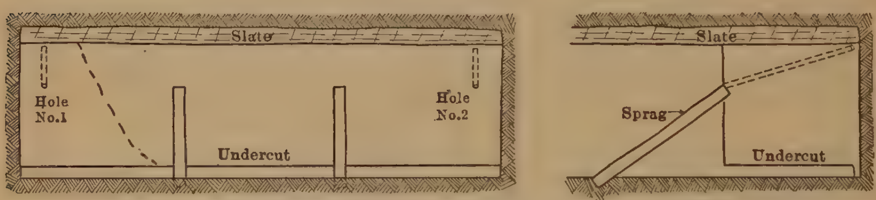
shots during an 8-hr. shift. This limit may be exceeded where the working places are concentrated; or, on the other hand, it may not be reached by a shot-firer who is required to visit a widely scattered group of narrow working places. Fig. 3 is a portion of the map of Ellsworth No. 2 Mine showing the South Main section. The route of the shot-firer is shown by the dotted line.

Cycle of Operations

In making his rounds, the first act on the part of the shot-firer is to find whether the loader at whose place he has arrived is ready to have his

coal shot down. It is possible to determine this by standing in the entry and calling or whistling to the man at the face, or by going into the working place and making a personal examination. By far the best method is the latter, as it affords more frequent inspection of the working places during the shift, but when a shot-firer must cover a difficult section, it may be impossible for him to go into each place on each round.

Having found that a shot is to be fired, the shot-firer is required to make a thorough examination of the condition of the place. First, the safety lamp is used to make sure that there is no trace of gas present. The roof and sides are examined to see that the required method of posting or timbering has been lived up to and that no additional timbering is required. Furthermore, two 7-ft. posts must be placed as sprags at the face in a room, or one in an entry, as shown in Fig. 4. The miner



FACE OF ROOM SECTION

Fig. 4.—Locations of Shot Holes and Sprags.

must load out all of his slack. The position of the place with respect to other workings ahead or at the sides must be noted, and the men in these places notified that a shot is to be fired. In the case of breaking up a rock fall in an entry, the same precautions must be taken and, in addition, men must be posted at safe distances on both sides of the fall to warn persons that the shot-firer is about to shoot.

The condition of the place having been found satisfactory, the shot-firer next examines the hole that has been drilled into the coal by the miner. First, he inspects it to be sure that the location of the hole conforms to the rules of the mine, these locations being shown approximately in Fig. 4. Second, by using the tamping rod he measures the length of the hole to see that it does not exceed the depth of the undercutting. Third, he examines the hole to see that it is rid of the drill cuttings.

The shot hole is then charged. The miner produces his box of explosive and the shot-firer selects from it the required amount of material, usually two and one-half sticks in the case of a "tight" shot, and from three-fourths to one and one-half sticks for a "butt" shot. The explosive is examined to see that it is in good condition and marked "Permissible." Keeping one full stick out, the others are carefully inserted one at a time and pushed to the back of the hole by means of the tamping rod. A

hole is then made in one end of the cartridge kept out. The hole is made parallel to the axis of the cartridge by using a wooden pin or a small piece of No. 0 copper wire pointed at one end. An electric detonator is then obtained, the 6-ft. iron wires straightened out and the cap inserted in the cartridge, the wires being wrapped once around the cartridge to prevent the cap from being pulled out. The cartridge being thus prepared is inserted in the hole, the end containing the cap going in last. The shot-firer allows the iron lead wires to slowly slip through his left hand while the cartridge is driven carefully in by the tamping rod held in the right hand.

Clay used in tamping the hole having been prepared beforehand by the miner, it is now ready for use. Kneeling before the shot hole, the miner inserts one piece at a time, while the shot-firer, still holding the ends of the lead wires, drives the clay home with the tamping rod, taking care to merely press the first two or three pieces against the explosive, while the rest are solidly packed in until the hole is completely filled.

After a shot hole has been properly loaded and tamped, the shot-firer attaches his cable to the ends of the detonator lead wires. This is done by tightly winding the latter five or ten times around the ends of the cable. Following this, the shot-firer walks slowly away from the charge, uncoiling his cable carefully to avoid breaking the connection between lead wires and cable. Also, to prolong the life of the cable, it is advisable to string it over cars, timber, a pick driven into the rib, or any other means of support that can be utilized to keep it elevated from the bottom and prevent its being buried under the coal about to be shot down. Of course this precautionary measure is found much more useful in the firing of the "butt" shot, but it is considered worth while even in the case of "tight" shots.

All the preliminary steps having been taken, the shot-firer then must select a safe place in which to stand while firing the charge. The place must be out of range of flying pieces of coal and also free from unsafe roof, or else properly timbered. Moreover, there is also the final precaution of seeing that all other persons have been warned and are out of danger and that no person, unaware of the shot-firer's presence, is walking into the danger zone.

Being satisfied that all is in readiness and that the men in adjacent places have been warned, the shot-firer calls out loudly his warning, "Fire!" Then he inserts the key in the battery, presses the ends of his cable against the contact buttons of the battery, pushes down on the buttons, completes the circuit, and fires the charge.

Following the firing of a shot, it is always advisable to wait a minute or two until the ventilating current has cleared away the smoke, and to afford an opportunity to listen carefully for any sound of roof movement.

Then the shot-firer proceeds carefully toward the point at which the shot has been fired, at the same time coiling his cable over the palm of his hand and the back of the upper arm. A careful examination of the roof follows. Then, by means of the safety lamp, any evidence of the liberation of gas is looked for. The face and sides of the place are examined for dangerous slabs that may drop off and injure the miner. Finally, the timbering and sprags are noted to see whether they have been disturbed by the blast. At the completion of his reëxamination, the shot-firer carefully explains to the miner each possible source of danger he has noted and instructs him how to remedy any dangerous condition existing. Or else, in the case of the discovery of any serious danger, the shot-firer fences the place off, fixes danger signals on the fences, and sends the miner affected to report to the mine foreman.

Misfires

Upon completing the circuit when firing with electric detonating caps, the charge may fail to explode for one or more of the following reasons:

1. Defective or exhausted battery.
2. Broken wire or connection or short-circuit due to imperfect insulation.
3. Defective detonating cap.
4. Deteriorated explosive.
5. Detonating cap detached from cartridge.

A battery of the type described in Fig. 2 has been found to give satisfactory results for an average of 600 shots, before it becomes necessary to renew the cells. Therefore, if the shot-firer keeps a record of his shots and the time at which a battery was placed in service, he can readily predict the date at which to expect the failure of his battery. On the other hand, some misfires are caused by imperfect contacts within the battery due to the presence of a small amount of coal dust at contact points. These difficulties can, of course, be avoided by frequent inspection and sandpapering of contacts.

Misfires due to a broken or disconnected, or short-circuited cable are the most frequent of all misfires, and are therefore generally the first suspected following the failure of a charge to explode. Proper care of the cable when about to shoot and again after shooting is quite necessary to reduce this source of misfires to a minimum. Fortunately, in most cases of this kind, the cause is easily discovered and quickly remedied.

In the firing of the 6020 shots on which this paper is based, only three misfires resulted from defective electric blasting caps, indicating quite

positively the dependability of this method of blasting. In order to avoid any danger from the subsequent handling of these caps, they were destroyed by placing them on the haulage road and running a locomotive over them. When subjected to this treatment, two of the caps failed to explode, while the third was exploded. The conclusions reached were that the first two had not been filled with fulminate, while the other was evidently filled but either the bridge wire or the iron lead wires were broken.

It sometimes occurred that a cartridge of explosive that had deteriorated found its way into a shot hole. Ordinarily the shot-firer may readily detect such cartridges because of the fact that they may be either unusually hard or soft. If, however, such a cartridge is used, the report of the blasting cap may generally be heard quite distinctly, but the charge of explosive fails to detonate. In two or three instances, misfires were the result of blasting caps becoming detached from the cartridge. This was due to the cap working loose during the charging of the shot hole.

In the case of misfires due to either of the first two causes, the remedy lay in the repair or replacement of cable or battery or in reëstablishing a broken contact. On the other hand, the last three causes of misfires always involved the drilling of a new shot hole. It was the general practice to circumscribe the hole with a rough circle at least 12 in. in radius and instruct the miner to drill a new hole outside the circle, and parallel to the original hole. In any case, the miner was forbidden to drill out the contents of the misfired hole, and reminded of the dangers of that procedure. When it became necessary to fire a second charge following a misfire, it rarely, if ever, happened that the first charge was exploded. Therefore, the miner was required always to search the coal shot down to locate the cartridges of the charge that failed. These were generally ready for inspection by the shot-firer on his next visit to the working place.

In the firing of the shots discussed in this paper there were no so-called windy shots and only one blown-out shot. The latter was in an entry and was a small charge intended to break down the coal remaining after the coal broken by the tight shot had been loaded out. The blown-out shot was apparently caused both by a charge that was too light and a hole that was slightly in the solid.

Shot-firer's Report

A form of report for shot-firers is illustrated in Fig. 5. It has been found that a report of this kind is useful in checking up the work of the shot-firers and in determining the cause of any difficulties in shooting. Also, it serves as a daily reminder of the necessity for exercising care.

FORD COLLIERIES COMPANY

SHOT-FIRER'S REPORT

————————————————Mine

Section No.————— Date ——————————

1. Did you examine each place for gas and other dangers before firing?
——————————————————————

2. Did you give warning to persons in adjacent places? ——————————
——————————————————————

3. How many shots did you fire today? ——————————————
4. Give location of any place in which you refused to shoot, and reason. ————
——————————————————————
——————————————————————

5. Misfires.

Location Cause

1——————————————————————
2——————————————————————
3——————————————————————

6. Blown-out Shots.

Location Cause

1——————————————————————
2——————————————————————
3——————————————————————

7. Did you examine each place for all dangers after shooting?——————————

—————————— ——————————

Mine Foreman Shot-firer

Fig. 5.—Shot-firer's Report.

Precautions in Shot-firing

The most essential precaution to be taken in an effort to safeguard the firing of shots is in the selection of properly qualified men to do the work. Moreover, it is especially desirable that one or two competent men be made available to act in the capacity of shot-firer in case of the absence of the regular incumbent of that position. It is a dangerous practice to place the battery and wire in the hands of a partly qualified man and tell him to shoot during the shift. Nor is it sufficient, as a rule, to assume that because a man was once considered competent to act as shot-firer, he will always continue to be careful in the performance of his duties. It is advisable to issue special instructions from time to time for the purpose of keeping shot-firers alert and informed of any dangerous practices to be avoided.

The importance of giving adequate warning before shooting ought never to be lost sight of by the shot-firer. The best practice is personally to warn all men on all sides within dangerous proximity to the charge, then to call out loudly to warn persons approaching from other parts of the mine, and wait 5 or 10 sec. after calling before firing.

So much has been written concerning the advantages of permissible explosives, that it is unnecessary to enumerate them here. For gaseous mines, they constitute a very important element of safety. To insure their most efficient and safe use, they should be carefully looked after in the magazine outside, examined before being issued to the miners, and transported into the mine in proper containers. Finally, they should be charged and fired by competent persons.

Discussion

Arthur LaMotte, Wilmington, Del. (communication to the Secretary*).—On p. 214, Mr. Cooper makes the following statement: "For the breaking up of an occasional rock fall in an entry, straight 40 per cent. nitroglycerine dynamite was used."

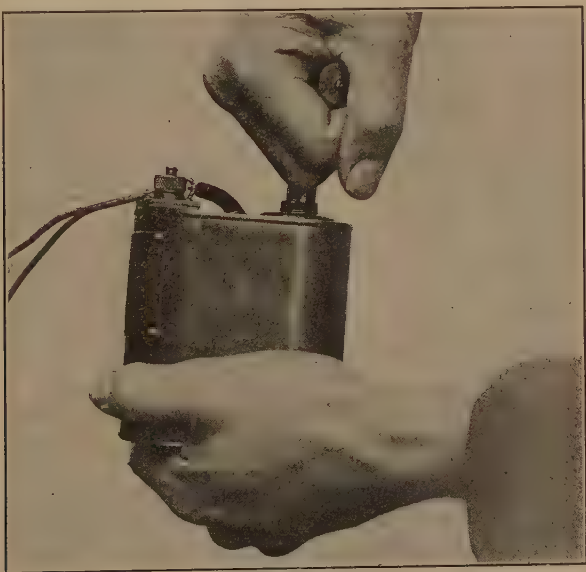


Fig. 1.

I believe that this practice is one of considerable danger and would suggest that the use of nitroglycerine dynamite should not be allowed in gaseous and dusty mines, inasmuch as we and other manufacturers of explosives make strong, quick-acting "permissible" explosives which are well suited for this purpose.

On p. 215, Mr. Cooper describes a battery used for firing the electric blasting caps, which consisted of two dry cells inclosed in a container, so that accidental contact was avoided. We have made a number of

* Received Jan. 26, 1917.

experiments with dry-cell batteries for blasting and have abandoned them all after exhaustive tests, as it is not possible, unless the blaster is equipped with rather expensive and delicate instruments, to determine whether the dry cell will develop enough current to fire the electric blasting caps. There have been so many instances of misfires due to their use that we have developed a blasting machine of much greater capacity and much smaller size, especially for use in coal mining, which is known as the Du Pont Pocket Blasting Machine (Fig. 1.) This will easily fire three or four electric blasting caps at one time, and is only about as large as the battery described in Mr. Cooper's paper.

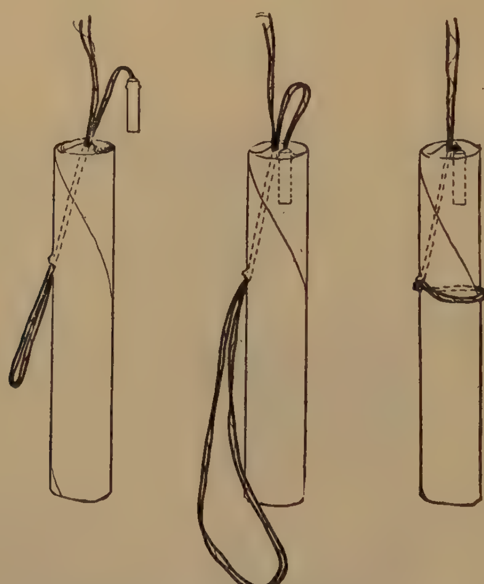


FIG. 2.—PRIMING METHOD FOR ELECTRIC BLASTING CAPS.

Mr. Cooper mentions the method used in making a primer, and also calls attention to the probability of misfires being due to the detonating cap becoming detached from the cartridge. His method of making a primer is described as inserting the cap in the cartridge and wrapping the wires once around the cartridge to prevent the cap from being pulled out. This is not very clear, but it corresponds to the practice known as half-hitching, the wires being brought at right angles to each other and a tension applied to the ends forming a hitch and sharp bend in the wires and a possible short-circuit of electric current at that point.

In our catalogs, and in the directions of almost all manufactures of high explosives, it is recommended that wires of electric blasting caps should not be half-hitched around the cartridge on account of the possi-

bility of short-circuiting or breaking the wires, with the attending danger of misfires.

A much better way of making a primer is shown in Fig. 2, which needs no description.

In Mr. Cooper's summary of the causes for misfires, the first one, defective or exhausted battery, would be obviated by the use of the small-size blasting machine previously described.

The second cause, broken wire or connections on short-circuit due to imperfect insulation, would be largely done away with by correct priming methods.

The third case, defective detonating caps, is one which very rarely occurs in our method of manufacture, as each electric blasting cap is tested twice before leaving the factory.

The fourth, deteriorated explosive, can be guarded against only by using common-sense measures in storing and handling the explosive when received from the factory.

The fifth, detonating cap detached from cartridge, can be entirely eliminated by proper method of priming the dynamite cartridge with the electric blasting cap.

The method that Mr. Cooper mentions for destroying defective electric blasting caps—that of placing them on the haulage road and running a locomotive over them—is a rather dangerous one, as the electric blasting caps are likely to fly for a considerable distance on exploding and embed particles of copper in the flesh of persons standing even 30 or 40 ft. away. A better way to destroy these is to bury them in close contact with a good electric blasting cap, several inches under ground, and fire the good cap. This will detonate the others.

Lucien Eaton, Ishpeming, Mich.—It is not the custom in the Lake Superior region, as far as I know, to employ shot-firers. Each man, or pair of men (most of the contracts are given to two men) do their own firing; each contractor keeps his powder separately. The general custom is to distribute powder (dynamite only is used) to the mines twice a week. The boxes, marked with the contract number and the level, are taken underground and the men take them to their working places. They are kept in boxes, which must not be nearer together than 50 ft. and are usually away from the timber roads or traveling roads. When holes have been drilled, one man usually takes down the machine, and the other man makes up the charge. He cuts the fuse, puts in the caps, etc., at a distance from the powder box. He then carries the powder charges to the working place where both men charge the hole. They do their own firing and warn others coming from any direction that the holes are being fired.

The customary manner of warning is to call "fire" and give the num-

ber of holes; when the holes have all been shot, they return to their working places.

We have discussed having special powder men and shot-firers. In fact, a good many years ago, when nitroglycerin was used, they had special men very often for charging. That requires usually the keeping of a large amount of powder in one place and we found it much safer to keep our powder supplies separate rather than in a magazine underground or one on the surface near the mines. We get our powder directly, except in outlying districts, from the powder companies' store houses. They deliver it right to the mines.

T. M. Chance, Philadelphia, Pa.—A few years ago, Mr. Vallat, of the Newport Mining Co., showed me a system of firing shaft rounds which they were using at that time in the sinking of the Palms shaft. This system employed fuse firing, but the fuses were lighted by the use of a battery together with a detonator, the ends of the fuses being led into a cardboard box which was partially filled with black powder covered with a small amount of rifle powder; the powder was fired by the detonator, which lighted the fuses. I understood they used from one to two boxes in the shaft to fire the round. The great advantage in the method was that it combined the advantage of timed shots secured through fuse-firing, with that of firing the round from the surface by means of a battery. The shaft was somewhat wet, but the boxes were paraffined and seemed to work very well.

A few months after I saw these boxes in use at Palms shaft, a contractor who was employed in sinking a shaft for us here in the East endeavored to make use of the same method, but without very much success, although I believe that the trouble his men encountered could have been overcome. I also understand that the Cleveland-Cliffs company used this same system of shaft-firing in sinking the Athens shaft, and I would like very much to have Mr. Eaton tell us what results they secured in that work with this method of shot-firing.

Lucien Eaton.—We used that for a very short time. This firing method which Mr. Vallat worked out consists of a box about a foot long and probably 4 or 5 in. square in cross-section, with small holes about ¼ in., bored in it from the sides. The fuses were cut and inserted in these holes, the fuses being cut in different lengths for the order of rotation. Then inside this box was placed about a tablespoonful of coated blasting powder, also a little package of fine rifle powder that ignited with an electric squib.

This blasting powder, on firing, made a long flame, igniting all the fuses so they would go off all at once. That made it possible to do the blasting by electricity, merely by that ignition. We tried that method of blasting in high raises. Instead of using an electric squib, although we did use that in some places, we merely inserted a fuse in the end of this box which con-

tained the powder. For convenience, the blasting powder for each firing was made up and kept in a sealed envelope. It was then merely necessary to cut a hole in the envelope, put in the fuse and when everything was ready, cover over the raise (we were supposed to have the end of this ignition fuse hanging down); the last man would light the fuse and go down. He left a long fuse so he could go down the shaft. It made quite a saving and was a great convenience. Recently we have been sinking some other shafts. We used what are known as electric fuse igniters. These consist of an electric squib to which is attached a short piece of fuse. The fuses come in lengths of 2, 4, 6 and, I think, 8 in. We used only the 2, 4 and 6. The igniters are made up by crimping on a blasting cap (we used number eights on the lower ends of the short piece of fuse); then wrapping the whole thing, squib, fuse and cap, with oakenite tape, to make it waterproof, and then dipping it in diluted paraffin paint. One man makes up all the powder and fixes the priming sticks. The shots were fired with the battery in series and the results have been very satisfactory.

We have had considerable water in the shaft but as long as the water does not get above the joint made in the leading wires we do not have any difficulty. We gave up the use of the blasting box such as Mr. Vallat used in Newport, except in dry places. I think these electric fuse igniters will take its place entirely.

H. M. Crankshaw, Hazleton, Pa.—It seems to me that the main point about shot-firing in bituminous mines has been almost lost sight of in this paper. The main thing in firing shots in bituminous mines is to fire shots in such a way that there shall be no possibility of an explosion occurring. Explosion after explosion in soft-coal mines has been traced to blown-out shots and the whole system of shot-firing must be so organized that if a shot does blow out it cannot cause a dust or gas explosion and so make a great deal of havoc.

In the first place, a bituminous mine should be zoned, that is to say, either by watering or by rock-dust protection, it should be so arranged that if an explosion occurs locally it cannot spread through the mine. This paper simply goes into the details of actually firing shots and does not touch the general subject of shot-firing in bituminous mines.

B. F. Tillson.—If it is not astray from the subject, the question which Mr. Eaton raised in regard to the use of delayed-action and electric exploders in one that is possibly of great interest to us. Our own experience at Franklin where we have been using them off and on for the past 6 years, has shown us that the use of the delayed-action exploder is now pretty satisfactory so far as the exploder is concerned but that the manipulation of the hand-operated blasting machine is something which is rather tricky, that is, unless the machine is level and the operator pushes the plunger down with an accelerated force, it is quite probable that all the exploders will not be ignited; and many times we have suffered loss

in the failure of a round being properly fired (some of the holes not being ignited) owing undoubtedly to the improper operation of a blasting machine.

This is particularly true where rounds include from 20 to 30 holes. We have used as many as five delays and the instantaneous electric exploders. Of course, what has seemed far superior, where you can employ it, is to use electricity from a direct motor-generator outfit, but that is particularly applicable possibly in shaft-sinking or limited operations and a little more difficult to perform where illumination of an electrical nature is not in general practice.

The Chairman (R. M. Catlin, Franklin Furnace, N. J.).—A good deal that has been said relates not especially to bituminous mines. I think we are nearly all interested in the subject of shot-firing and what has been said, I think is more or less applicable to all mines.

Lucien Eaton.—We formerly used delay-action electric fuses, which are different from the fuse igniters and had unsatisfactory results. Probably for the reasons just given, some of the holes would not go and some would go too soon. We tried it both with the battery and with the electric current. We found by test that there was a certain definite limit to the amount of current to which the exploders should be subjected to give the best results in blasting with the direct current but we found the best results to be with large hand batteries. We use the No. 5 DuPont battery which is good for 100 holes and we shoot as high as 150 holes with them; if the men are careful in making good connections all the holes will be blown, but if they are not, the resistance in the connections seems to be enough to cut down the capacity of the battery. If a battery is good for 100 holes, if they are all clean, it may not be good for more than 35 or 40 if they are carelessly done. We found that by having a battery of much larger capacity than theoretically needed, we get very satisfactory results.

Mr. Tillson.—My remarks applied to fuse igniters as well as delay-action electric detonators.

Chairman Catlin.—This is in connection with this subject, but not peculiarly applicable to the bituminous mines—particularly the question of the care and distribution of the explosive. I think it would be interesting if somebody could devise a way in which it would be possible to control the distribution of the explosive. Particularly in these times it is desirable not to have deadly material very widely distributed. The paper touches on that matter in relating to the caring for the explosive and carrying it in boxes. The difficulty of control is considerable, particularly where there are open quarries in the vicinity.

Powder is used indiscriminately and often gets into the hands of those

that should not have it. Is there any way by which the illicit use or the illicit possession of powder can be avoided?

E. T. Lednum, Joplin, Mo.—Under the heading "materials used," Mr. Cooper reports that permissible explosives were used exclusively in blasting coal, while on pages 214 and 216 he reports that straight nitroglycerine powders were used for blasting rock from falls in the entry, and that the explosive charges were simply placed on the boulders and covered with wet clay. A short hole drilled into a boulder with a very much smaller charge of permissible explosives, say a part of a cartridge, would not only give better results, but at the same time reduce the vibrations with possible subsequent falls of roof, and also make a safer condition in the mine, especially in the presence of gas.

As to blasting batteries, dry cells are uncertain and liable to fail without warning. The small blasting machine, such as made by the DuPont company, is safer and more efficient. It is practically indestructible, and will last indefinitely when properly handled. The wires leading to the blasting circuit are held in place by thumb screws on the blasting machine.

On page 219, the method of charging the priming cartridge and the fact that the end containing the cap is placed in the drill hole last, may also result in misfires. In addition, the sharp bends in the wires from the half-hitch around the cartridge, may also result in broken wires or short-circuits. An improvement on this method to hold the electric blasting cap in place would be to punch a hole diagonally through the end of the cartridge, double the wires and force them downward through this hole, and pull the wires through until a loop is formed through which the cartridge could be placed. After this, punch a hole straight in in the end of the cartridge at the top, parallel to the longest axis of the cartridge. Then place the electric blasting cap in this last hole and pull the wires tight. This should give a safer primer, and at the same time prevent breaking of wires or short-circuits from sharp corners.

Under the heading of Misfires, defective or exhausted batteries are a fruitful source of trouble. As I have previously outlined, they become exhausted without warning, and the wires are also merely held in place at the terminals. Proper priming and charging methods will overcome misfires from broken wires or poor connections or short-circuits, etc.

Defective detonating caps may be the result of improper storage and rough handling. Caps should be stored in dry places, as they are sensitive to moisture.

Deterioration of explosives is also caused by improper storage.

Detonating caps may become detached from the cartridge, as a result of the method of priming as indicated by Mr. Cooper. I feel quite sure that this condition can be overcome by proper priming methods, and I

have found to be very satisfactory, the method which I have attempted to describe.

EDWIN M. CHANCE, Wilkes-Barre, Pa. (communication to the Secretary*).—The point has been raised by one of the speakers that the object of shot-firing in bituminous mines is to prevent the explosive from causing a gas or dust explosion that will spread to considerable portions of the mine workings. I would submit that this conception is fundamentally faulty. To me it seems that shot-firing has for its chief object the cheap and efficient breaking down of coal so that it may be loaded into cars, and transported to the surface.

Now, to my mind, Mr. Cooper has shown us how such work may be carried out in detail and has thus rendered a valuable service to the mining industry by making this information a matter of permanent record. There is often a great temptation to be so carried away by the larger and more theoretical side of questions such as this of shot-firing that the writer loses his sense of perspective to such an extent as to forget to give the small and necessary details that in the aggregate form the basis of success. Mr. Cooper has painstakingly transferred these details from actual practice to the printed page, from which they may at any time again be transferred to practice. For this he deserves much credit.

J. J. RUTLEDGE, McAlester, Okla. (communication to the Secretary†).—In some mines in the Southwest, the miners' daily supply of permissible explosive is sent in to his working place in galvanized iron canisters of rectangular cross-section, in size about 9 by 6 by 3 in. These canisters contain only sufficient explosive for one day's use, and the canister with any unused explosive is sent from the mine before the close of the shift. In accordance with the mining law, the company handles the explosive both ways. This procedure prevents the leaving of any explosive in the mine over night, or for several days, where it would deteriorate. All shots in the part of the country above-mentioned are fired by special shot-firers after all other persons have left the mine workings. In some States the miner tamps the shots, in others the shot-firer tamps the shots as well as fires them. There has recently been brought to my notice a fiber canister for carrying permissible explosives which is said to be used in many Colorado mines. It is water- and weather-proof as well as a non-conductor of electricity, and for this reason very desirable in mines where there are electric wires. It can be purchased for from 60 to 70 c. It is 9 by 6 by 3 in. in size.

The author says that 40 per cent. straight nitroglycerine dynamite was employed for breaking up an occasional rock-fall in an entry. In my opinion, dynamite should not be employed in a coal mine. A per-

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* Received Mar. 5, 1917.
† Received Feb. 5, 1917.

missible explosive can be obtained which would perform the work mentioned equally well, or better than dynamite. The method of handling and distribution of the electric detonators is commendable, also the use of a safety-contact firing battery. Some idea of the difference between the conditions in Pennsylvania bituminous machine mines and in the solid shooting mines of the Southwest may be gained by a consideration of the relative number of shots fired by one shot-firer in the two districts in certain periods of time. In the Ellsworth mine, one shot-firer fired 40 to 50 shots in an 8-hr. shift, while in Kansas and Oklahoma one shot-firer frequently fires an equal number of shots in from 1 to 2, or 3 hr. time, but it is manifest that firing at this rate does not permit the proper inspection of preparation of the charges, where this is done by the shot-firer. In the former State, the miner prepares and tamps the charges but the shot-firer fires them; in the latter the shot-firer tamps and fires all the shots, the miner having prepared the charge.

The remarkably small number of misfires resulting from defective electric detonators as noted by Mr. Cooper certainly testifies to the reliability of the electrical method of firing shots.

The paper brings up for discussion the debatable subject, "Is it safer and more efficient to fire shots of permissible explosives, by means of an electric battery, by a special shot-firer, while all the miners and other employees are in the mine, or to fire them by special shot-firers after all other employees have left the mine?" The plan as outlined by Mr. Cooper could not be followed in Kansas or Oklahoma mines without coming into conflict with the mining laws of the States mentioned. Mr. Cooper's paper corroborates the experience of other fields, viz., that permissible explosives can be used successfully and with efficiency only when the shots are charged and fired by special shot-firers.

The suggestions in the first paragraph, under Precautions in Shot-Firing, are especially pertinent. Coal-mine officials should keep in close touch with their shot-firers and should caution them in regard to the dangerous practices to be avoided, and should give the shot-firers proper support when they take a stand against firing dangerous shots. It does seem sometimes as though the most reckless or ignorant man is employed as a shot-firer instead of the most careful and intelligent man. A mine owner or superintendent would not think of entrusting his property to the care of a drunken or sleepy night watchman, yet many operators and mine officials place their entire underground properties in the hands of incompetent or reckless shot-firers. They make the shot-firer the scapegoat and apparently quiet their consciences by the argument that, by the mining laws, the shot-firer is made the sole judge of conditions under which shot-firing is done; if he comes through the ordeal with his life, well and good, if he does not, then another man can be found who will be willing to take the risk for the high wages offered. The manage-

ment of the mine owes it to the owners of the property, to the general public, and to the shot-firer and those dependent upon him to make his work as safe as possible, and to that end he should be carefully chosen and his work regularly checked up to make sure that he is not following dangerous practices. The State Inspectors should also check up the work of shot-firers. There is one mine in the Southwest, in which during a period of 10 years, 15 shot-firers have been killed. Quite a number of mines have had four to six shot-firers killed at intervals varying from several months to several years. A mine superintendent should take as much interest in the work of his shot-firers as he does in the work of his fire bosses, since the safety of life and property depends on the carefulness of both.

George S. Rice, Washington, D. C. (communication to the Secretary*).—Mr. Cooper should be congratulated on the unusual data he presents. It is not often that we have from a trained engineer so interesting a record of daily performances in a special work like that of shot-firing.

The method employed by the Ellsworth Colliery Co. seems particularly good, and the example of the care taken in making examination of roof and gaseous conditions before firing a shot is worthy of emulation in the Southwestern coal field, where the shot-firers often fire as many shots in a couple of hours as are fired in a full shift at Ellsworth; but there are several points which seem to call for comment.

The evidence regarding fumes from permissible explosives given by Mr. Cooper is most interesting. It has been alleged by the miners' officials in some districts that the fumes from permissible explosives are bad. Mr. Cooper says that no special distress was given to the shot-firer, except in changing from one class of explosives to another, but the charge-limit (1½ lb.) of the explosive was not exceeded, which, of course, is important. Apparently he allowed a very brief time for the diffusion of the smoke and fumes, since he says "it is always advisable to wait a minute or two until the ventilating current has cleared away the smoke." It would seem best where a shot is in a heading or room where the air circulation may not be active, to wait 5 min.

For a mine that was regarded as gaseous, since locked safety lamps were used, and in a mining district in which the coal dust is of an explosive nature, it seems unwise to employ ordinary dynamite for breaking up falls of rock in adobe shots, as described, or even in "block shots." There are a number of permissible explosives that will do practically as efficient work in breaking up rock, and even if not exactly equivalent, the slight gain through the use of dynamite is not compensated for by the increased danger in a bituminous mine, potentially dusty or gaseous, in

* Received Feb. 2, 1917.

the case of neglect of some individual, through its use in blasting, or through accidental discharge in handling. But even when permissible explosive is used, every adobe shot should be covered with a good-sized heap of incombustible material, preferably with dry incombustible dust on top to form a dust cloud. Such covering greatly increases the efficiency of the explosive and lessens the danger of ignition of coal dust in the vicinity.

The diagram of the shot-firer's battery is not sufficiently detailed to make it clear about the position of the contact buttons. If, as it seems to show, these buttons project above the top of the dry-cell box, this does not seem so good a plan as to have them recessed, since an accidental blow or fall after the key is turned might cause a premature depression of both, whereas if recessed this danger would be much more remote.

The test made of the detonators by running a mine locomotive over them to determine whether the detonating compound, usually mercury fulminate, was in proper condition, and the conclusion that in two instances it was missing from the containers, is a method of testing open to question. The explosives engineer of the Bureau of Mines, S. P. Howell, made five informal trials of the method, by placing detonators on a rail and running a 2¼-ton truck with flanged iron wheels over them. The end of the cap that contains the mercury fulminate was squeezed out in each case without explosion. The detonators had been so placed that the electric bridge ends were just beyond the edge of the rail and in three instances were not destroyed, so that on connecting the detonator wires with a firing battery the fulminate remaining in contact with the bridge was discharged. It would seem best for the testing of detonators that failed in a mine, to take them outside the mine and try with a stronger current.

The best way to destroy uncertain detonators, Mr. Howell advises, is to tie several in a bundle with a fresh, presumably good, detonator and fire them with a battery, after placing them behind a barricade to avoid flying particles of the copper container.

The shot-firer's report to the company is an excellent one for recording data, and it would be very interesting if the company would give a summary for a year or more of the results obtained not only by Mr. Cooper, but by the other shot-firers, giving a summary of the data collected through the agency of these shot-firers' reports, especially the cause of misfires and blown-out shots.

D. Harrington, Butte, Mont. (communication to the Secretary*).—To one not familiar with the coal field in which the system described is located, a few additional facts would be of value: What is the seam thickness? What is the nature of the coal as to friability, partings,

* Received Feb. 26, 1917.

etc.? What is the nature of the roof? Is the coal wet or dry? What is the pitch?

Mr. Cooper's paper brings out the fact that many meritorious practices are followed in this mine. In general mining practice, as well as in shot-firing, it is pleasant to learn that the coal is undercut and that cuttings are loaded into cars before shooting, as against the only too abundant practice of leaving cuttings not only on the floor, but banked in the cut in such a manner as to practically negative the value of cutting. The use of permissible explosives in amounts less than 1½ lb. per hole is to be commended as against the use of black powder or dynamite, or against the use of permissible explosives in amounts up to 3, 4, or 5 lb. per hole and yet expecting the explosive to give no flame and not to shatter the coal. Handling of detonators by shot-firers exclusively is a most commendable practice and to my mind the carrying of explosive in insulated boxes in limited quantities into the mine by individual miners is by far a safer system than that of company men transporting into the mine large quantities of explosives to be distributed later on. The question of transportation of explosives underground and distribution to the working face is certainly one of importance. Carrying the daily supply by the individual miner is looked upon with disfavor by miners' organizations, as well as by many prominent mining men, yet in my opinion past records will prove that far fewer accidents are attributable to individual carrying than to the transporting of large quantities for later distribution.

The small two-cell battery is certainly worthy of extended use, in view of its small cubical content, cheapness and lasting qualities, as well as for the fact that closing the circuit is dependent on two separate and distinct connections, which practically eliminates the possibility of premature firing by accidental contact.

Removing drill cuttings from holes, refusing to fire holes of greater depth than that of the machine cut, use of wooden tamping rod, and of clay for tamping, are all commendable.

The matter of selection of shot-firer is of prime importance, and of scarcely less importance is the amount of authority to be delegated to him. While it is true that the shot-firer should be a man of judgment and discretion, local conditions, generally beyond control of the mine management, only too frequently result in the selection of men of only mediocre ability: in some localities wage-scale agreements are such that the earnings of miners, loaders, and others are so much greater than those of the shot-firer that the only men who will accept this work are the newcomers who do not understand conditions as to shooting and who soon demand more lucrative work, or leave; in other regions the risk of firing shots is so great that only dare-devils, frequently with little knowledge of explosives or of firing shots, will undertake the work, though as much as $15

per shift is paid for 3 or 4 hr. work, the shot-firer practically betting the employer $15 or more daily that an explosion will *not* occur even though such barbarous mining practices are in effect as "shooting off the solid" with black powder and fuse, using coal-dust tamping in gaseous places frequently covered with dry inflammable dust.

Assuming that the shot-firer is a competent man, his life is at stake every time he fires a shot and he should be allowed every available safeguard. Apparently the Pennsylvania Bituminous Law covers this at least in part by providing that the shot-firer may "*refuse to charge any holes not properly placed.*" In many localities where similar legal provisions apply, they are rendered inoperative by fines or other restrictions placed on shot-firers by miners' organizations, when the shot-firer refuses to charge certain holes deemed by him to be unsafe; in other regions mine officials force the firing of unsafe holes or compel the use of unsafe materials or methods. The shot-firer, if a competent man, should be able to dictate all details relative to drilling, loading and firing of shots as well as to safety considerations preliminary to shot-firing, such as sufficiency of propping, undercutting if done, condition of air, etc.; if he remains underground while firing shots, his life is endangered with every shot fired and he is entitled to be allowed to protect himself. As a matter of right and justice, the shot-firer should be independent of both miner and company, be paid adequate wages by the State or Government, and be appointed preferably by a district judge upon recommendation of miners' organizations and company officials and be amenable to court only.

The main criticism of the system described by Mr. Cooper would be that shots are fired during the day while all the men are in the mine, as against firing them by night when ordinarily few, if any, men are working; the ideal system, however, would be an electrical one in which all shots would be fired from the surface after all men were out of the mine. While the use of permissible explosives, in small quantities per hole with electric battery firing, use of clay tamping, undercutting of coal and removal of cuttings before firing of shots, are all commendable precautions, yet firing of shots with men in the mine carries with it many dangers. The working faces as well as air currents have considerable quantities of fumes from explosives, which, while possibly not dangerous, are certainly disagreeable and at least temporarily render air conditions in working places such that clearness of vision is obscured and accidents possible. Again, where shots are fired with men in the mine, frequently the shot-firer neglects to warn before shooting, or he will give warning when firing the first shot, fail to warn as to other shots, or will shout "fire" and without lapse of time complete the circuit through the battery, leaving practically no time for an oncomer to retreat. The successive concussions due to firing shots are inclined to cause loose pieces of roof or rib to fall

even in remote parts of the mine, and these frequently cause accidents. All of these sources of danger are eliminated if shots are fired on the shift when miners are not at work; in some mines where as many as three, five, seven or more shots are fired in a room face, it is customary to fire one, or possibly two shots and allow this coal to be loaded out before firing the other shots. Under such conditions it is generally desirable to have shot-firers on hand during the working shift; however, in the mine described by Mr. Cooper, but two shots are fired in a place and presumably they may be fired simultaneously, or at least in quick succession, and the presence of shot-firers on day shift is not a necessity.

Mr. Cooper does not state whether the sprag placed against the face of the coal after undercutting is removed before firing shots; possibly the influence of leaving the sprag would be small, yet it appears that leaving the sprag would to a certain extent operate as a bar to efficient work by the powder. The explosive used during the greater part of Mr. Cooper's observation is an ammonium nitrate explosive of comparatively slow rate of detonation. It is noticeable that the priming stick is entered last, with primer toward outside of hole, generally believed by authorities to be the proper method; however, this class of explosives is frequently somewhat insensitive to detonation and to assist in this matter frequently the paper ends of contiguous sticks of explosive are removed. It would be interesting to know whether this is done. It would also be interesting to learn whether the paper covering of sticks is slit lengthwise and whether the explosive is rammed to fill the full diameter of the hole. In wet holes, is the ammonium nitrate explosive used, and if so what precautions (if any) are taken toward waterproofing?

Mr. Cooper's paper is an admirable one, not only for the data it contains but also for the fact that it opens a broad field for discussion of a subject of vital interest to all those interested in the operation of coal mines.

The Pennsylvania Mine Fire, Butte, Mont.

BY C. EDWIN NIGHMAN, B. S.,* BUTTE, MONT., AND ROLLINS S. FOSTER,† BONNE TERRE, MO

(New York Meeting, February, 1917)

THE following is a description of the methods used in rescuing men and extinguishing the underground fire at the Pennsylvania mine, Butte, Mont.

This fire, which cost the lives of 21 men, began about 9 p.m., Feb. 14, 1916, at or near a ventilating fan on the 1,200 air-shaft station. This shaft at that time was downcast, while the main hoisting shaft was upcast. Owing to the proximity of the fire to the air shaft, the air in the latter soon became heated, and as the fan on the level was not running—the power having been cut off shortly after the outbreak of the fire—the air shaft became upcast to the 300 level. At this point a blanket of cold air from the surface caused the smoke and gases to be drawn in on that level and carried down again through the workings to the 1,200. A short time after the discovery of the fire, water was turned down the air shaft through a series of pipes around the collar of the shaft, which caused the smoke to recede below the 300, so that within 2 hr. it was possible to search that level for the missing men.

From the above it will be apparent why the air shaft was upcast instead of downcast, as under normal conditions, and why men were suffocated at considerable distances from the seat of the fire. The smoke and gases that came up the main hoisting shaft did not diffuse through the workings to any extent.

There are doors on all levels in the connections to the main hoisting and air shafts, but in the confusion and running around many of these doors were left open, and this undoubtedly caused the smoke and gases to spread more rapidly than they would otherwise have done. It was necessary for most of the men on the various levels to pass close to the air shaft in going to the hoisting shaft, and when they encountered the smoke and gases in that vicinity a number turned back and climbed down with the direction of the air currents and were then suffocated, while all those who passed through the pall of smoke to the main shaft were saved.

There were 220 men in the mine that night, and all but 25 were hoisted to the surface within 30 min. after the discovery of the fire. Of the 25, five went out through the Tramway mine, and one through the Moun-

* Fireboss, Anaconda Copper Mining Co.
† Late Safety Engineer, Anaconda Copper Mining Co.

tain View mine, while the remaining 19 were missing. Their bodies, with two of a rescuing crew, were found later. All the men who came to the surface through the main hoisting shaft were able to go to their homes, with the exception of two, who were sent to the hospital but suffered no ill effects.

Discovery of the Fire, and Rescue Work

About 8:50 p.m. of Feb. 14, two shift bosses who were on the 1,600 and 1,800 levels detected the odor of smoke. One immediately went to the fan on the 1,600 station and discovered smoke coming down the air shaft. He then went to the 1,200 level. The other, whose run included the 1,500 and 1,600 workings, sent men to notify others in that part of the mine. About the same time, men were sent through the remaining workings to notify every one to come to the stations and be hoisted.

The men were notified as follows: A station tender notified the shift boss for the levels above the 600, who went to the 1,000 and notified the assistant foreman and another shift boss. The latter were conversing with two miners who were sent to warn two others that were working on the 1,000 sill. The assistant foreman and the shift boss then climbed through 1,284 manway (See Fig. 1), which was the only one open to the 1,200, and walked out on the sill toward the air shaft. They opened the inside one of two doors in the air-shaft crosscut (1,266) and encountered a wall of smoke through which they could not pass. They then went out to the main shaft, were hoisted to surface and immediately had water turned down the air shaft.

No men were working between the 1,000 and 600 but a large number were employed in the west stopes above the 600. Two skinners were sent to warn the men in these stopes, but encountered so much smoke in the vicinity of the air shaft on the 600 that they were obliged to turn back. They then climbed up a manway to the 500, went back to the top of these stopes and down into them, warning the men as they met them. On the first floor one skinner met eight miners; he told them of the fire and said that they could not get to the main shaft on the 600 level but should follow him to the 500. He said, later, that the men seemed to realize the serious nature of this news, but for some reason failed to follow him. Without delay he returned to the 500 and lined up nine men who had hesitated to enter the thicker smoke in the direction of the air shaft, and led them to the main-shaft station. These men were then hoisted safely to the surface, though the skinner was unconscious when he reached fresh air.

The other skinner later climbed down through these same stopes and also met the eight men who had been warned, through by this time they had gone from the first floor to the 600 sill. He also told them that they could not possibly reach the main shaft through the 600 level but must

climb immediately to the 500. None made reply or attempted to follow him, so he returned to the 500 and proceeded toward the main shaft until he had reached the manway through which he had climbed from the 600. Once more he climbed down through this manway and attempted to go in on the 600, but found the smoke thicker than before and noticed that three horses in the barn were dead. He again returned to the 500 and found four men who had come out of the stopes above the 500. These he took to the main hoisting shaft and was hoisted to the surface with them.

About 9:15 p.m. a miner on the 300 station attempted to go in on that level to warn seven men who were working in stopes tributary to that level, but was stopped by smoke. He then went to the surface and down on the cage to the 300 station in the air shaft, with a shift boss and two others, but they found the smoke so dense that they were obliged to return to the surface.

After the men had been hoisted from the mine, which was about 9:30 p.m., the mine foreman, the foreman and assistant foreman of the Tramway, and others, kept going to the stations on all the levels above the 1,200, hoping to find some of the missing. About 9:45 p.m. a shift boss came to the surface and reported that he thought he had seen two men near the 600 station. The assistant foreman of the Pennsylvania and a volunteer from the St. Lawrence mine put on oxygen helmets and went to the 600 in search of these men. They failed to return in 20 min. and others went in search of them. They were found about 400 ft. from the station stretched out in a drift with their helmets still in place. They were removed to the surface in about 20 min., where every effort was made to revive them by using both the Pulmotor and the Schaeffer method of resuscitation, but without success. It cannot be positively stated, but it is thought that the oxygen tanks of these helmets were partially depleted and that they were not aware of it and did not examine the tanks when they took the helmets.

About 11:30 p.m., the air was clear on the 300 level and men climbed down the air raises from the surface without helmets and in a crosscut which was only about 100 ft. from the air raises they found the bodies of seven men. Resuscitation by the Schaeffer method was employed on the warmest bodies, but without result. About 4 a.m. the following day these bodies were brought to surface and no further effort was made to find more of the missing men until the air shaft had been made an upcast. By 3 p.m. of the 15th the upper levels were clear enough to begin a search for the missing men. About 2 hr. later the bodies of two men were found with their arms around each other on the fourth floor of a manway above the 800. Two more were found farther up in the same manway and two others on the sill near by. These were six of the eight who had been working in the 600 west stopes and who had been warned by the skinners. The bodies were removed to the surface by 10 p.m.

About 1 a.m. Feb. 16, the second day after the fire, the bodies of the other two men from the 600 west stopes were found near the bottom of 1,284 manway, together with those of four men who had been working on the 1,000. Because of the smoke these bodies were hoisted through this manway to the 1,000 and taken to the surface from there. On account of the poor condition of this manway, it was 2 p.m. on this day before the bodies reached the surface.

From the foregoing, it is evident that all the men in the mine were warned, except the seven who were working in the vicinity of the 300 level. Of the 19 who remained in the mine, it is certain that 12 would have been saved had they done as they were instructed.

Fighting the Fire

Preparations

As noted above, the fire broke out at or near the ventilating fan on the 1,200 air-shaft station. The air shaft was downcast because this fan, which had a capacity of 55,000 cu. ft. per minute, and a smaller one on the 1,600 level, were drawing the air down and forcing it into the workings below. The workings above the 1,200 were ventilated through a series of air raises, down which air was forced by a surface fan. Shortly after the fire started the power was turned off on the surface and the heat of the fire turned the air shaft into an upcast, igniting the timbers. The rapidity with which the fire traveled is shown by the fact that when rescue parties entered the crosscut to the air shaft on the 1,000 level about 3 hr. later, they found the shaft timbers already ablaze. The smoke and gases traveled with great rapidity, for although the adjoining mines were immediately notified of the fire, four men in the St. Lawrence, one in the Mountain View and one in the Tramway mine, were overcome before the doors and bulkheads in the connections to these mines could be closed.

The turning of water into the air shaft immediately after the fire was discovered was the only attempt made to fight it until a suction fan had been installed on the surface at the air shaft and all the bodies removed from the mine, which was on the afternoon of the second day of the fire.

As soon as the fire was discovered, preparations were made to insure the safety of the adjoining mines and to facilitate fire fighting by changing the ventilation, *i.e.*, the air shaft was to be made an upcast and the main hoisting shaft a downcast by the use of a fan on the surface. To further safeguard the adjoining mines, as well as to hold back the smoke and gases, concrete bulkheads with iron doors were at once built in all open connections to the Pennsylvania from other mines. Twenty or more of these bulkheads were built.

The air shaft carried the water, air, electric light and power lines—

with the exception of one power cable to the pumps on the 1,800, which was taken down the main shaft—and as these were destroyed it was necessary to replace them before any fire fighting could begin. Work to install these new lines was commenced on the 15th, but it was some days before they were completed, for an old 4-in steam line had to be first removed and there was no manway or pipe compartment in the main shaft other than a space about 6 in. wide between the end-plate and the "chippy" guide center. However, while the fire fighting was going on, this space was enlarged to 30 in. in the clear from the 1,400 to surface.

By the morning of the 15th, foundations for fan and driving motor had been placed at the mouth of a short fan drift, which connects with the air shaft about 25 ft. below the collar, and by noon this fan was installed and running. The fan used was a No. 13 Sirocco, whose sea-level capacity is about 83,500 cu. ft. per minute, but at this altitude (5,700 ft.) is somewhat less. So that there might not be any interference with the ventilation. a second of these fans having both motor and engine drives was installed, so that it could be operated independently of the first fan.

Locating the Fire

Up to the time the fan was started the air shaft remained downcast to the 300 and upcast from the 1,200 to that level, but it became entirely upcast very quickly and the main hoisting shaft a strong downcast, though it later became less so because of the caving of the ground around the air shaft above the 1,200 station. The effect of this change of air was at once noticeable, for that afternoon, the 15th, it was possible to go anywhere without the use of oxygen helmets. The extent and location of the fire could then be determined.

After the reversal of the air currents on the 15th, parties entered all the levels and found that the fire centered in workings north of the air shaft on the 1,200 level (see Fig. 1), and west of the main north-south crosscut (1,207). Also, that as far as could be determined it was only on the sill, in the stopes above, and in the drifts and crosscuts connecting to the air-shaft crosscut (1,299). These workings were connected through stopes with those of the St. Lawrence mine on the west, but were not accessible because the latter were worked out and filled with waste. The only other possible way of getting to that part of the 1,200 sill was by coming in from the Flat vein through A 1,256 crosscut.

On the 15th, a party went in on the 1,400 sill and climbed up through the raise on the east end of the Flat vein workings to the 1,200, but were able to get only part way to A 1,256 crosscut because the timbers at the junction of the vein and crosscut were already on fire. That night helmet men also climbed through this raise to the 1,200, but by the following noon the fire had reached the 1,400 sill and no further investigations

could be made. At the ends of the workings on this vein on the 1,400, pillars had been left which had some drifts through them. Solid concrete

FIG. 1.

bulkheads were at once erected in these drifts and the fire sealed off. This was the most important work done in the first days of the fire, for, on the

1,400 sill the Flat vein is separated from No. 3 vein by a short, timbered crosscut and the latter vein had been stoped from the Silver Bow on the east to the St. Lawrence workings on the west. Also, this vein crosses the main haulageway near the main hoisting shaft and had the fire penetrated these stopes it would have been impossible to get to the existing fire. It is well to note that the Pennsylvania workings are as a rule very dry and the upper workings are all square-setted and contain many timber bulkheads.

The west side of the fire, therefore, was not accessible and all that was known of its extent and location was that it was to the west and north of the air shaft, in the Anaconda, Air Shaft and No. 10 veins as well as in the Flat vein, which is about 400 ft. south of the Anaconda vein and is approximately parallel to it. It is worthy of note that the Flat vein and the crosscut to the air shaft are about 350 ft. apart and that the connecting crosscut (A 1,256) was not timbered and the only inflammable materials therein were the track ties and the insulated electric light wire with the occasional spreaders from which they were hung.

Methods of Fire Fighting

The methods used were in general those used at the Mountain View fire of 1913.[1] Drifts and crosscuts were driven from the nearest accessible workings to the fire and as much as possible of it was extinguished with hose lines and by pouring water into the workings above the sill and other inaccessible places by means of diamond-drill holes. The air currents were controlled with bulkheads, doors and brattices, so that the men would always be working with the air; *i.e.*, so that the smoke and gases would be carried away from them. As far as possible the air pressure was maintained at a point where it was just sufficient to hold back the gases without supplying fresh air to the fire. All places were securely timbered and air and water lines were kept in all faces. As many men as possible were employed in all the headings and frequent reliefs were given. The men worked 8-hr. shifts.

Where the gases were too strong, helmet men were employed. The helmets used were Draeger, 1909 and 1911 models. A number of extra sets and three ½-hr. helmets, as well as a good stock of accessories, were kept on hand at all times. The supplies were brought as needed from the company's rescue station at the Anaconda mine. Concrete for bulkheads was mixed on the surface in a motor-driven mixer and sent into the mine in regular mine cars. About 600 tons of concrete were used during the fire.

Fire fighting did not begin in reality until the 17th, as the water and air lines were not completed to the 1,200 level; the work of the first few

[1] C. L. Berrien: Fire-fighting Methods at the Mount View Mine, Butte, Mont. *Trans.* (1915), **52**, 534.

days being limited to the cleaning up of openings and the building of bulkheads. After building the two bulkheads on the 1,400, others, having iron doors and 2-in. and 3-in. pipe connections, were built just off 1,207 crosscut in the air-shaft vein crosscut (1,299) and in the air-shaft crosscut (1,266). These two were very important, for they made 1,207 safe up to 1,209 and 1,250, where the fire was burning strongly, and they also prevented the air from short-circuiting to the air shaft. Many other bulkheads, the locations of which are to be seen on the accompanying map (Fig. 1), were constructed as required in the later work.

Mining

In driving the headings great difficulties were encountered because of the gases and smoke, the heavy ground and the water pouring down from the diamond-drill holes. Wherever filling was met, spiling or forepoling was necessary and in many places breast boards were also required.

On Feb. 18, drifting was begun in 1,209 (No. 10 vein) and in A 1,250 (Anaconda vein). As the timber had burned out, both of these places were caved. An added difficulty was met when 1,292 crosscut was started from their intersection. Here the old sill was about 25 ft. wide, all the timber was burned out, and the back could not be seen because of the smoke, while the hanging wall (dip about 60°) would slab off in large pieces and crush the timbers. It was necessary to retimber this place several times during the fire fighting. It was also necessary to crib over the sets to the back as they were put in. One-inch pipe lines were connected to the fire hoses and thrust ahead to reach the hot ground and burning timbers. Temporary canvas brattices—and later, wooden doors and concrete bulkheads—were built to control the air pressure so as to hold back the gases.

A few days later a drift (1,249) was started north of the above and driven west through a pillar and into the workings on No. 10 vein. In the solid, rapid progress was made but, as was the case in the other headings, extreme difficulty was experienced in getting through the old gob. Charred timbers were found when the west or hanging-wall side was approached. A large amount of water from the diamond-drill holes made it very hard to hold the drift open. At times there would be a run of ground of about the consistency of thin mud which would come out in the drift for some distance, and several times the men had to run out to avoid being trapped in the mud. Whenever these flows occurred there was an influx of strong sulphur smoke which would drive the men out. This became of such frequent occurrence that only helmet men were employed there. This heading was stopped when the hanging wall was reached (39 ft. advance in 19 days).

On the same day a similar and parallel heading (1,263) was started just north of the above. Rapid progress was made here, the average

daily advance for 40 days being 4 ft., including the time lost while two concrete bulkheads were being built. In passing through the gob, no evidence of fire was encountered, other than an inconsiderable amount of gas. Some water from one of the drill holes was coming down on the foot wall. At the junction of 1,263 and the Anaconda vein, some fire was seen, though the ground near by was thoroughly wet down through the drill holes. Having found fire here it was imperative to have further openings to the west; accordingly a foot-wall lateral (B 1,253) was started between the two bulkheads in 1,263. This was driven in the solid because of the greater rapidity of advance possible there, but was not to take the place of an opening in the vein workings themselves. Such an opening was made later. From B 1,253, two crosscuts were driven through the Anaconda vein gobs, but no fire was encountered. Work was resumed in 1,263 on Mar. 10. This place was a difficult one in which to work, because it was wet and because a great deal of gas, chiefly SO_2, was present. The solid ground was so hot that it could not be touched, and blasting was delayed until the holes could be cooled down with water jets.

While this work was in progress it was found that the pressure from the air column in the downcast hoisting shaft was not sufficient to hold back the gases in all the headings, hence it was necessary to reduce the number of splits from the main air course, 1,207. This was accomplished by building a bulkhead in 1,207 just south of 1,249. Two canvas brattices were used until the bulkhead was completed. An immediate improvement was noted in the two splits, A 1,250 and 1,209. To maintain a counterbalancing pressure in the other workings, 1,249 and 1,263, a No. 2 Sirocco fan was placed in the frame of a wooden door on the west side of A 1,208 in A 1,274. By varying the opening of the door itself, the proper pressure was kept up. In order that only fresh air would be drawn in by this fan, two raises from the 1,300 were sealed off and the air brought down 1,284 raise from the upper levels.

On Apr. 5, after many crosscuts and drifts had been driven (see maps) a quantity of burning timber fell behind the bulkhead across A 1,250 at the junction with 1,207. This was quickly extinguished and was the last fire actually seen.

Work on the 1,000 Level

It having been determined that the fire was above the 1,200 sill, it followed that the best point from which to attack it was the level next above. Also, it was necessary to prevent the fire from getting to the upper levels. The first level above the 1,200 is the 1,000, the distance between the two being about 140 ft.

Other than in the air shaft, no fire was seen on the 1,000, but great quantities of gas filled the sill. A week after the fire had started, and


PLAN OF DIAMOND DRILL HOLES
1000 LEVEL
1200 LEVEL
LEGEND
Full Lines = 1000 Sill; also 1200
Dotted Lines = Stopes above 1200; 1300
CB = Concrete Bulkhead
W = Wooden Door
SCALE OF FEET
0 25 50 75 100
1252 Raise
Diamond Drill
Anaconda Vein
Air Shaft Vein
No. 10 Vein
1246 Raise Top Sets Filled with Concrete
Air Shaft
1221 Raise
1246 Raise
Cen

Fig 2.

on the fourth day of work in 1,067 crosscut (Fig. 2) a gas sample, which was analyzed by the Bureau of Mines, showed an oxygen content of only 14.58 per cent. while it contained 0.68 per cent. CO. A candle would burn here notwithstanding the low oxygen content, but that, with the large percentage of CO, made this an extremely dangerous atmosphere. This would indicate that the burning of a candle is not a test for a respirable atmosphere. This gas came up chiefly through two raises, A 1,246 and A 1,252. These were, therefore, sealed off with concrete bulkheads. Several other bulkheads were also constructed. The chief work done on this level was that of diamond drilling.

Diamond Drilling

As mentioned previously, one of the most important methods used in extinguishing the fire was that of turning water into the burning areas through diamond-drill holes.

Two drills were obtained on the 18th, but drilling was not begun until the 21st because the conditions existing on the level made it impossible to place them in the positions chosen. The first drill was placed in 1,039 crosscut just north of A 1,026, which place was directly over the No. 10 vein stopes and farther to the north than any fire had been seen on the 1,200. The second was placed at the end of A 1,026 crosscut. Considerable difficulty was experienced in getting the drills into place and in operating them, for the level was very gassy and there had been no time to prepare stations for them. At the first two setups it was necessary to use rods from 12 to 15 in. in length, and also to drill holes in the back in line with the drill holes, so that the rods could be raised and lowered. Furthermore, the waste water backed up so that the crosscuts were miniature lakes for some distance. Not alone were the setups inconvenient, but the nature of the ground itself made drilling very difficult. Practically all the holes from the 1,000 penetrated the Rarus fault, which is a wide complex fracture zone of finely crushed and altered granite with included blocks of sharp, rough, quartzose vein matter. In many of the holes two standpipes were necessary.

The first holes were drilled with the idea of cutting off the spread of the fire, particularly on the north and west where the stopes connected with those of other mines and were inaccessible. The remaining holes from the 1,000 were drilled at intervals over the fire area, as required by the conditions obtaining on the 1,200, and so that all portions of the stopes would be thoroughly wetted down, water being turned into them as fast as they were completed. In all, 37 holes having an aggregate length of 3,909 ft. were drilled from the 1,000 level.

On the 1,200 a series of short holes was drilled from 1,208 into the stopes on No. 10 vein below the 1,200. No traces of fire were found

with these holes. Also, a series of holes was drilled from a station in 1,209 to the air-shaft vein below the 1,200, and a few into the upper floors of the stopes on the Anaconda vein below this level. None of these showed any evidence of fire having burned below the sill, even where it had been most intense above. By the time these holes were finished, the 1,200 sill had been thoroughly opened up and it was considered unnecessary to do further drilling. Twenty holes with an aggregate depth of 984 ft. were drilled from the 1,200.

Work on the 1,300 Level

To determine the lower limits of the fire, repair work had been started on this level about Mar. 1. Under very difficult conditions 1.329 and 1,330 crosscuts (See Fig. 3) were driven across the gobs of No. 10 and the Anaconda veins. No signs of fire were found in these crosscuts, though considerable gas and much water were present at all times. It was later decided to connect these workings with the main hoisting shaft. The 1,311 drift on No. 10 vein was then started. Working 8-hr. shifts for a part of the time and 6-hr. shifts for 8 days, this heading was advanced 370 ft. at an average rate of 11½ ft. per day. By the time this drift had reached the old workings, conditions had so improved throughout the fire district that it was not considered necessary to connect at once with 1,329; the heading was stopped temporarily and a crosscut driven to the air shaft.

Work on Other Levels

Elsewhere in the mine the only work done was that of repairing and putting the mine in shape for the resumption of mining. This included the retimbering of the air shaft below the 800 and the pumping out of the three bottom levels which had been flooded with the water used in fighting the fire.

Results of the Fire Fighting

As mentioned above, the last fire was seen on Apr. 5. Since that time the air shaft has been retimbered, the 1,200 sill and stopes including the Flat vein have been thoroughly reopened and no trace of smoke or gas has been found, proving that the fire has been extinguished.

Origin of the Fire

The true origin of the fire is not known and probably never will be. Three possible causes have been advanced: (1) That a lighted candle was left on some timber; (2) that it was an electrical fire; (3) that it was of incendiary origin.

As to the first, it is known that it was practically impossible to carry a lighted candle through the double doors near the air-shaft station because of the strong draft induced by the fan there. That it was an electrical fire has been doubted by some, owing to the fact that the fire apparently jumped several hundred feet through the untimbered A 1,256 crosscut to the Flat vein workings. Because of this, it is argued that the

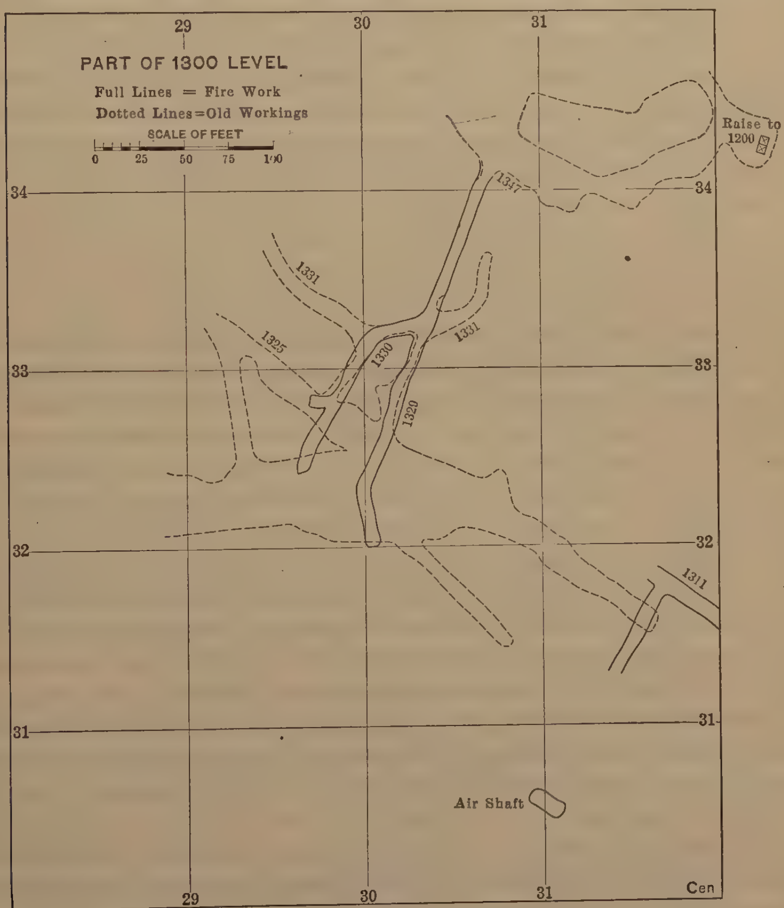

FIG. 3.

fire must have started in two places and was therefore of incendiary origin. But, about 2 hr. before the discovery of the fire the fan at the 1,200 air-shaft station had been oiled by a shift boss, and at that time everything was all right. Had the fire started at or near the fan the smoke would have been carried rapidly down through the Flat vein workings, as the air was being forced in that direction, and would have been noticed by the men at work there before it was noticed elsewhere. Also, it is

argued, the fire could not have been electrical, because the fan must have been running in order to carry the fire to the Flat vein stopes. Had the fire started from a short-circuit at the fan, it would not have been possible for the latter to run long enough to do that. However, about seven o'clock on the night of the disaster, the power went off and came on again with a big surge. The voltmeter at the main power station registered 2,800 volts, or 500 above normal. This overload caused the circuit-breaker on the fan circuit at the Pennsylvania mine to kick out. An electrician from the Leonard mine substation was sent to throw it in again and when he did so the lights glowed for a short time instead of burning brightly, indicating some trouble on the line. The power was then left on till shortly after the discovery of the fire. The above are the important facts in connection with the origin of the fire and they are too meager to justify the acceptance of any of the three probable causes advanced.

Methods of Protection Against Future Fires

The collars of all shafts have been provided with 2-in. sprinkler lines which will throw water into all compartments. About 18 months ago, at the Mountain View mine, a fire at the collar of a downcast shaft destroyed the loading platforms, headframe and engine room but was extinguished by water from two 2-in. hose lines before it had burned more than 25 ft. below the collar. It was concluded from this experience that the sprinklers as mentioned would afford adequate protection.

In addition to the sprinkler lines, it is also recommended that the upper portion of all shafts be made of some non-inflammable material, say concrete. When the above-mentioned shaft was reconstructed, the upper 30 ft. was made of concrete and the shaft timbers for about 60 ft. were covered with asbestos paper and steel roll cap.

All hoisting shafts are to be downcasts. This was to have been done at the Pennsylvania mine as soon as the 1,600 and 1,800 levels had been connected to the air shaft. At the time of the fire, raising from these levels was in progress.

At intervals of 400 ft. in all shafts, water tanks with a capacity of 3,000 gal. have been erected. These are connected with float feeds to the water columns and are always full so that they may be used in case of emergency.

Downcast shafts are to carry all pipe, electric light and power lines.

There are water lines in all the main drifts and crosscuts on the different levels. Also, water may be turned into the air lines by means of interchangeable connections between them. In connection with these, it is believed that it would be advisable to have connections for hose and pipe at all manways and chutes and an automatic or manually operated sprinkler system on shaft stations and at the tops of the principal man-

ways or raises. Tees with hose connections and valves should be placed in all water lines on the sills at intervals of, say, 100 ft. This has been done on some levels at one of the mines.

Hose, pipe and monkey wrenches, hammers and a sharp axe or two, should be kept for fire purposes only, on shaft stations and at other critical points on each level, depending on the extent of the workings on that level.

Watchmen or "firebugs" are employed at all the mines to examine all working places after the shifts have gone off. Abandoned workings are also examined occasionally to guard against incipient fires.

Many concrete bulkheads with iron doors now exist, but enough more should be used to make possible the isolation of any required section of the mine in so far as the workings will permit.

All surface and underground fan motors have been provided with the necessary protective apparatus. All underground fan stations have been lined with expanded metal lath and concrete. The fans are oiled at stated intervals, and cared for by either shift bosses or the mine electricians. It is recommended that the starting boxes, fuse blocks, etc., be so inclosed that they cannot be tampered with by any other than the persons responsible for their operation. This should also apply to switches and circuit-breakers on the trolley lines. Buckets of dry sand or Pyrene fire extinguishers should be kept near all electrically driven machinery.

Metal receptacles should be provided for oily waste. Recently a fire was started on a shaft station by a pile of waste, but was discovered before any damage had been done.

There are rigidly enforced regulations in regard to the careless use of candles and to smoking.

Surface fans are to be used for exhausting only. A short time ago a fire at the collar of a downcast fan shaft burned the shaft timbers for some distance even though the fire was discovered practically at its inception. Not alone is the danger that of burning out the shaft timbers, but the smoke is forced into the workings below and may cause the suffocation of workmen.

A telephone system is practically a necessity in any large mine. These are in use in many of the Butte mines. In general, the phones are on the shaft stations only, but it would be a great advantage to have others at important places on the levels. In case of danger, the men could be promptly warned by the use of the phone and the loss of life thus avoided. In connection with the phone system, or without it, there should be some means of warning men to come to the shaft stations or to leave the mine. This can readily be done in electrically lighted mines by the use of a special flash signal to indicate the existences of danger and then, by giving the station signal, to show where it is. Carmen or

dumpers would notify the men working in raises, winzes or stopes. The use of such a signal is not customary in Butte, though it is used elsewhere.[2] Connections to other mines and to shafts are plainly marked by signs on the main traveling ways in the Butte mines.

The systems of mining now being tried out in Butte make use of much less timber than the ordinary square-set method and, therefore, decrease the danger from fire.

In some of the mines, when working near fire zones, a row of timber has been removed from hanging to foot wall of the stopes at some convenient place, the floor mined out and this space filled with fine waste which is thoroughly wet. This operation is repeated at the same place in each successive floor until the level above has been reached. This makes a vertical wall about 10 ft. thick from level to level, and it has proved cheap and efficient. This practice originated at the Anaconda mine.

Oxygen helmets and lungmotors or pulmotors should be provided at all mines, and men trained in their use should be available. This is well taken care of in the Anaconda Copper Co.'s mines by two rescue stations and by many men trained in the use of apparatus. These stations are located at the Anaconda and Tramway mines, which are about a mile apart. The stations are very modern and are practically alike in equipment. At the two, there are 14 sets of 2-hr. 1907 type, 10 sets of 2-hr. 1911 type, 13 sets of 2-hr. 1914 type, and 12 sets of ½-hr. 1914 type, Draeger helmets. The apparatus, as well as all repair parts—of which a large stock is always on hand—is kept in zinc-lined lockers. The oxygen bottles are kept pumped to 130 atmospheres. A large supply of oxygen and several hundred potash cartridges are kept in stock. At each station there are hand- and motor-driven oxygen pumps. In the buildings there are a bath and toilet and a small smoke room. Outside, there is a timber gallery for apparatus practice. At the Anaconda station there are also three pulmotors and one lungmotor. Each station has a light automobile truck which is used for the transport of helmets or supplies to the mines where needed.

An attendant is on duty at each station at all times. These attendants work 8-hr. shifts. If the attendant has to leave the station at any time, he leaves word as to his whereabouts with the telephone central.

All the fire work at this mine was conducted under the supervision of C. L. Berrien, assistant general superintendent of mines, H. R. Tunnell, mine foreman, and C. E. Nighman, fireboss.

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[2] Since the above was written, all the Anaconda Copper Mining Co.'s mines have been equipped with special oil switches on the lighting lines, by means of which station tenders or other authorized persons may give an "all out of the mine" signal. This consists of nine flashes repeated three times, followed by the signal for the level on which the danger exists.

Report of the Secretary of the Committee on Safety and Sanitation

Being a Classified Synopsis of the Data Collected by the Committee

By E. Maltby Shipp, Secretary, New York, N. Y.

Your committee's secretary submits the following report, or summary, to the members of the committee, in an endeavor to lay before them a general review of the information so far received and also his own knowledge of the work as carried on at a number of plants, this being done in part to secure further information and suggestions from members and lead to a more effective means of furthering the work of the committee. It is suggested that more detailed data be compiled regarding engineering features involved, and that papers be contributed which will deal with the safeguarding of underground electrical installations.

There are few operating companies which have not started a more or less systematic campaign to safeguard their employees from accidental injuries and unhealthful surroundings, but a great deal is to be gained by experience. All may welcome the advice and coöperation of those who have been actively engaged in the work and have demonstrated the results obtainable by organized effort, and it is, therefore, exceedingly gratifying to note the ready assistance and hearty coöperation rendered by the companies that may be truly termed pioneers in the movement.

No solution of the safety problem is to be found in a strict standardization of methods, rules or devices, as the various mining, milling and metallurgical practices present too many conditions differing broadly in general principles, but some standard system embodying principles already proven to be efficient should be created, which may be modified or elaborated to meet specific requirements of each individual company, or to fit working conditions as they exist in the various mining camps throughout the country.

An industrial plant is primarily a business undertaking, therefore the economic side of a safety or welfare campaign is of importance. Many small operators have been deterred from undertaking the work by their belief that it involved an expense beyond their means, and one that could not be counterbalanced by greater efficiency or saving along other lines. This idea is generally due to a lack of knowledge of the actual costs of such work, and of the results secured by judicious selection of a method commensurate with the size and nature of the opera-

tion. The widespread adoption of safety measures by both large and small operators is conclusive argument in favor of economic results.

The cost need not be great, and should be, in a measure, proportional to the pay-roll. There are many ways in which a company will benefit both directly or indirectly by the work. The efficiency of the men is increased by their working under conditions where there is little danger of accident. Safety work with the proper enforcement of rules creates a higher degree of order, neatness and attentive application to work, thereby increasing the productive capacity of every man on the property, and lowering the operative costs. Legal fees, damage settlements, care of dependents, insurance premium rates, loss of time, and loss of experienced labor are all materially decreased, while a considerable burden is lifted from the local community in the number of dependents, cripples, widows and orphans. Improved sanitary conditions and a higher percentage of health among the men creates better artisans and more efficient labor.

Whatever means one may adopt to introduce safety measures, it is essential that certain fundamental principles be followed and the following suggestions may be helpful:

Secure the coöperation of every man and woman on the property.

Let all employees participate in the movement, thereby securing their interest and ready acceptance of safety rules.

Make it clear to all that the management is doing this work for the good of the workmen and their families.

From the outset, indicate plainly that the movement is to be made a success, and that it has the full support of the management.

Simplify rules, but create all that are necessary to guard against every hazardous practice.

Have no rules that are not enforced.

Whenever the judgment of the men has to be relied upon and strict ruling is impossible, have safety suggestions that will guide them to act properly and promptly.

Do not depend too much upon rules and regulations, but remove the danger wherever it is practical to do so.

Those administering safety orders must see that they are carried out. A boss ordering a dangerous roof taken down, or merely a nail driven flush, should not leave until his instructions are executed.

No possibility of conflict of authority should exist.

Constant vigilance must be maintained.

An educational campaign is absolutely essential.

No intoxication should be tolerated, and should be condemned by fellow workmen, as well as by the management.

Secure the interest of the men during their leisure hours.

Accurate records embodying classifications of injuries must be kept.

Every injury, no matter how slight, should be recorded, and medical attention or first-aid given by competent persons.

Any boss discovering an injury to any of his men which has not been reported, should inquire as to its cause, and reprimand failure to report such injury.

Medical examination of all men employed should be made.

Proper classification of injuries soon reveals a danger spot and the departments needing the greatest amount of attention.

Excessively long shifts cause fatigue, which leads to many accidents.

Eight-hour instead of 10-hr. shifts have been found by several companies to reduce the accidents occurring toward the end of a day's work.

Fooling and wrestling should be strictly prohibited, as many accidents are caused in this manner.

It is important to select the best grade of tools and general equipment.

Engage only sober, careful and reliable men.

Periodical bulletins, or booklets, published as a means of furthering the "safety-first" movement, arouse the interest of the men. Excellent examples of these are, the *Ingot*, published by the Raritan Copper Works; the *Anode*, published by the Anaconda Copper Mining Co.; *Employers' Magazine*, published by The Lehigh Valley Coal Co.

Organization

It is impossible to suggest any fixed plan of safety organization, but a review of those created by some of the larger companies will illustrate how the executive end of the work may be handled and an appropriate system adopted.

Edwin Higgins has furnished us with a description of several examples of safety organization existing among the mining companies operating on the Lake Superior Iron Ranges.

The first form of organization is illustrated by that created by a company which operates a number of large mines on the same range; briefly it is as follows:

An inspector is in charge of the safety department. His duty is to inspect all mines as frequently as possible, and submit reports and recommendations to the manager.

A committee composed of mine superintendents, head mining captain, master mechanic, assistant auditor, secretary of pension department, safety inspector and the manager, who acts as an ex-officio member, meets once a month; it confirms or rejects safety recommendations, and considers all accident reports. A committee of mine foremen, consisting of three members, each selected from a different mine, makes periodical trips in and about the mines. The inspector accompanies this committee and incorporates its recommendations in his report. At each mine there is a committee made up of workmen, with duties similar to those of the committee last-mentioned, except that its activities are confined to the mine from which the members are selected. The personnel of this committee is constantly changed, so that in time all employees are given a chance to criticize conditions.

A committee, acting more or less independently of the others, consists of three mine superintendents. It is the duty of this committee to investigate all fatal accidents.

The second form of organization is illustrated by:

A company operating large and small mines. The safety department is in charge of an inspector who, with the assistance of three experienced miners, inspects each mine at least twice a week. These committee men report to the inspector, who, in turn, reports to the superintendent.

Inspector makes monthly report in triplicate; one copy is returned to the inspector,

one to the head of the department, and the third is retained by the superintendent. The department copy is returned with improvement report, when improvements are made. All company bosses and first-aid men meet every 2 months to discuss accidents.

Another form of organization is illustrated by:

A company operating scattered mines which has an organization similar to the one just described. The safety committees consist of all foremen from a district, and act under the supervision of a general safety inspector.

The fourth operates in connection with one large mine:

An engineer has charge of the department of "efficiency and safety." Under him are three assistants. These assistants are given a certain feature of the work to oversee, and are required to report to the head of the department. At first daily inspections were made, but as conditions have improved, only two or three trips through the mine each week are now found necessary. There are daily meetings, at which all matters pertaining to efficiency and safety are discussed; these meetings are attended by the manager, superintendent, head of the efficiency and safety department, and the mine captains. By holding meetings in the morning it is possible to consider the reports of the shift bosses to the mine captain, and prompt attention can thus be given to all urgent matters.

The Copper Queen Consolidated Mining Co. was the first to undertake this work in the Southwest, and their organization and methods have been widely copied throughout Arizona and adjoining States. They began their organization by the appointment of a committee composed of three officials and six workmen, selected from the different mining divisions and departments. This committee formulated plans and rules, and developed the permanent organization, namely:

A central committee, composed of the superintendent of the mining department as chairman, the superintendent of mines, the master mechanic, a hoisting engineer, an electrician, the chairmen of each of the four workmen's committees, and the safety inspector, who is also the secretary. This committee acts as a legislative body, all matters which cannot be handled by members of the other committees being referred to the central committee for decision. Where heavy expenditures are necessary, the matter is referred to the general manager, with central committee recommendations.

There are four workmen's committees, three being from the mines, and one from the shops, power plants, yards, etc. These committees have from four to six members, whose duties are to investigate all accidents of a serious nature occurring within their jurisdiction. Their report and recommendation, if passed upon by a majority of the members, are submitted to the central committee; they also pass upon suggestions sent in by workmen in their departments. The members are appointed by the inspector, and are supposed to serve for 6 months, appointments being made 1 or 2 months apart, so as to retain men who are acquainted with its duties. The inspector coöperates with all committees, and renders as much assistance to them as possible.

In contrast to these well-organized departments, we find one large corporation, employing between 12,000 and 13,000 men, entrusting all its safety work to one man, who, unassisted by committees, has made a great reduction in the casualty list during 1914. This is an excellent example of what can be accomplished by individual responsibility, but only in exceptional circumstances is it recommended.

Methods

To secure results, the confidence and enthusiasm of every employee must be sought and maintained. It is necessary to adopt measures that

will continually call to the attention of all employees the importance of strictly complying with the rules and regulations, and being always vigilant of danger, and of the safety of others. Francis P. Sinn emphasizes the necessity of generalizing committee inspection work, as follows:

"Those most accustomed to certain dangers are, as a rule, the last to recognize those dangers. Any scheme for accident prevention which confines the inspection for safety of any department to men in that department will be defective on this account. It is usually the man unaccustomed to a particular line of work being done, and unfamiliar with the surroundings, who first sees the dangers at hand, and it is often he who has the best suggestions to offer to remedy the dangerous conditions. Whatever remedy is decided on, however, must fit into the particular conditions at hand, and must be entirely satisfactory to the man in charge of the work in question.

"It is at times hard to distinguish between a disregard of danger due to ignorance of its existence, and that due to a braggadocial spirit; the latter is more far reaching in effect, as it sets a bad example to others and is less easily remedied. An educational campaign must be followed, which will emphasize the safety-first idea and be beneficial to those at the top of the organization as well as those at the bottom.

"To assure one of the coöperation of the men, it is important to demonstrate plainly that the safety campaign is entirely for their good, and it is well to have them participate in the movement from the very beginning; at the same time, having it distinctly understood that the management inaugurated the idea, and will do everything in its power to make the campaign a successful one. Before commencing any work, or giving any publicity to the plan, have a careful survey made of the existing conditions, and avoidable dangers; then formulate the plans accordingly."

The organization created or the person detailed to attend to the safety work must fulfill all duties involved in a systematic and business-like manner. All suggestions should be acted upon, and those presenting them advised as to whether or not they are of a practical nature and are to be followed. Suggestions will thus be encouraged. Prizes sometimes induce men to take an interest in the work.

It is unjust to criticize or blame individuals for accidents due to methods or conditions which they have not been taught to regard as dangerous. On the other hand, publicity should be given to all accidents, as a matter of education. Illustrations showing how such accidents occurred, and how they could have been avoided, should be placed on bulletin boards and shown at public gatherings. Any violation of rules must be severely dealt with, and either carelessness or recklessness duly censured.

The Bethlehem Steel Co. has very successfully secured the interest and coöperation of its thousands of employees, in safety work, without any organization. The work is entirely in the hands of their safety engineer, Mr. Funda. They realized that there were many dangers due to unguarded machinery and the general layout of their plant, which it would be impossible to immediately alter or safeguard, and they feared that if suggestions were made by workmen's committees and not at once followed up, a bad impression would be created.

Mr. Funda has carefully studied the matter, he fully realizes where guards and changes are necessary, and gradually the plant is being safeguarded throughout. They claim that 90 per cent. of the accidents are due to careless practices, and the results of their campaign have more or less demonstrated the truth of this theory. They had but four fatal accidents in 1914, and a notable decrease in serious ones. No fair comparison can be made between minor accidents in 1913 and 1914 because the more accurate record kept during the latter year would give apparent increase, where in reality there is a decrease. The company expects soon to create an organization including workingmen's committees, believing that sufficient headway has now been gained.

Any safety campaign should be entered into enthusiastically, but not with an idea of immediate perfection. The strongest elements in such work are an accelerated interest and the natural development of a working system, involving safer and more efficient operative methods. Reckless expenditure of money for safety appliances and prizes, or a general hurrah campaign at the outset will result in a reaction; men's enthusiasm will soon lag, and the movement be defeated by its own explosiveness. However, conservatism should not be carried to excess, as general enthusiasm must be created as quickly as conditions will allow.

The human element is an important one, and the psychology of the case must be carefully considered. The point of view of the men may be entirely different from that of the management, and it is essential to realize this fact. In this regard, Mr. Sinn says:

> "There is a tendency, however, on the part of the workmen, and a perfectly natural one, to believe that they do not have to be told not to get hurt, as they are just as much interested in keeping alive and well as we are in having them so."

Every move in the educational work should be made with this point of view in mind, special emphasis should be given to the fact that we are all exposed to hundreds of dangers which, as busy men, we fail to remember, unless we are expressly reminded of them, and have the contributory causes and thoughtless practices clearly pointed out to us.

Where skilled labor exclusively is employed underground, the problem of education is not exceedingly difficult, yet even then the "old-timer" is likely to scoff at familiar dangers and disregard safety innovations. When foreign labor is employed, perplexing difficulties confront one; even requiring the men to sign receipts for rule books have in many cases nearly led to riots. The babel of tongues not only complicates the educational feature but it adds to the general confusion underground, and creates a natural nervousness among the workers, increasing the number of otherwise avoidable accidents. It is well to segregate the men according to race, as far as possible, and never allow a boss to have under him men to whom he cannot readily talk.

The number of accidents may be greatly reduced by having a central

employment agency; a physical examination of every man employed, and, at stated intervals while in the employ of the company, a rating based on such examination whereby men are given work for which they are better fitted; and a complete card index record of all employees, and of all accidents.

The Copper Queen Consolidated Mining Co. has adopted the following educational methods, in addition to its splendid system of first-aid exercises and the training furnished by committee work. Suggestion cards and letterheads are placed at all mines and departments, and may be used by any employee who wishes to send in a safety suggestion. The suggestions may be dropped in a box provided for that purpose or mailed to the inspector. They are collected from the box once a month. All suggestions are considered carefully by the central committee or the workmen's committee, and the sender advised of the action taken.

Whenever a fatal or serious accident occurs the inspector must be notified at once, and a report made by him of the accident. If the nature of the accident is such that photographs will prove of value, these are taken and filed with the report. Bulletins are published, telling where the blame should be fixed, and how the accident might have been prevented. Great care is used to secure all possible information, and to place the blame where it belongs, neither company, officials, nor workmen being spared.

A rescue station has been centrally located and is fully supplied with everything needed for safety, first-aid and rescue work. First-aid supplies are also placed in round, air-tight cans, 10 in. in diameter, made at the company's shops at a cost of $4 each; these are distributed through the workings and, except during a short time at first, little trouble from meddlers has been experienced.

Competitive field meets between first-aid teams are held under the auspices of the Red Cross, and according to the rules of the U. S. Bureau of Mines. Bulletin boards are at such places as the public library, reading room, Y. M. C. A., dispensary and outside the rescue station and employment office; on these are posted monthly accident reports, photographs illustrating the right and wrong way of doing certain work, for which men posed as if caught by falling of ground, etc. Safety mottoes printed on large cards have been posted throughout the mine workings.

Smokers are given, where the men are furnished with "safety-first" cigars; at these smokers lantern slides are shown, and moving pictures are given in the Y. M. C. A. by the committee. All employees and their families are invited to safety rallies. The lantern slides of underground scenes have proven of great interest to the ladies. The men are constantly inquiring when the next meeting is to be held, showing their interest in these rallies.

Upon the Michigan Iron Ranges, the U. S. Government Rescue and First-Aid Car, and first-aid instructions, have been of great assistance in securing the attention and coöperation of the men; this is true not only in Michigan, but wherever the Federal demonstrations have been given.

Some companies pay for all safety suggestions, and give buttons to employees who have served on the committee. Cash bonuses have been tried in various parts of the country, but probably nowhere has a better method been adopted, or greater results achieved, than at the United States Coal & Coke Co's. operations at Gary, W. Va. Howard N. Eavenson wrote the committee a letter, which was read at the Pittsburgh meeting as a part of the discussion of Mr. Higgin's paper, and which sets forth in the following words the premium system as practiced at Gary:

"For the past 4 years, the United States Coal & Coke Co. at Gary, W. Va., a subsidiary of the U. S. Steel Corporation, has been awarding premiums to mine foremen and assistant foremen for the prevention of accidents to its employees, which is arranged on a merit and demerit basis. The system adopted is as follows:

Qualifications

1. "No man shall be eligible for a premium for any month, in any position, who has not worked in that position every working day during the month excepting one, unless he shall have been promoted during the month from one position to another, and is eligible in both positions.

"*Explanation.*—It has been a custom for men in this section of the country not to work regularly. A number of accidents have occurred because of the regular foreman not working, and new men substituting. This qualification has therefore been inserted with the view of getting men to work regularly, and thereby assist in the prevention of accident.

2. "A man's work must be satisfactory to his immediate superior, and, if it is not satisfactory, his superior has the right to charge him with demerits to the extent of ten per month.

"*Explanation.*—This qualification is inserted as a means of discipline, as in a number of instances some of the assistant foremen do not take sufficient interest in the prevention of accidents to attend the weekly meetings of the officials for discussion and investigation of accidents which occur.

3. "This premium is not considered a part of the assistant foremen or foremen's wages, but is strictly in the nature of an award or a gratuity for faithful services, rendered to the company.

Distribution

1. "Each foreman or assistant foreman is charged with demerits for each man who is injured under his charge, each month, at the rate of 10 demerits for each minor, 20 demerits for each serious and 40 demerits for each fatal accident.

2. "Any foreman or assistant foreman who does not have any accidents under him during any month is given a credit of 5 merits, which will go toward reducing the number of demerits standing against him until all the demerits are wiped out, when he will not be given any further merits until he again receives demerits. No accident in which the victim loses less than 7 days will be considered.

"*Explanation.*—It is not considered advisable to allow a man to accumulate merits, as it would have a tendency after he had accumulated a large number of merits to cause him to be less careful.

3. "Any assistant foreman in whose section the company's mine inspector finds any dangerous practices or dangerous conditions which might cause accidents will be charged 5 demerits each visit he makes and finds such conditions. If he finds a section to be O. K., and no dangerous practices or conditions the assistant mine foreman will be given a credit of 5 merits.

"*Explanation.*—This provision is made as it is often the case that accidents occur for which the assistant foreman is not directly responsible; his place might be as safe as it is possible to make it, but through carelessness on the part of one of his workmen an accident might occur over which he would have no control. In order to aid an assistant foreman who has been so unfortunate as to have an accident of this kind to get back into good standing, it has been provided that 5 merits be given him if his place be kept in safe conditon.

"On the other hand, an assistant foreman might permit dangerous conditions and practices in his section and still be fortunate enough not to have an accident, though not due to any special care or attention on his part. It is, therefore, provided that such assistant foreman be given 5 demerits for the condition of his section.

4. "The foreman's account will be charged with all demerits and credited with all merits of the assistant foreman under him, excepting when demerits are given for neglect of duty or causes other than accidents.

5. "No person who has 10 or more demerits to his credit at the end of the month shall be entitled to any premium, but if he has less than 10 demerits, he shall receive a premium of $5, if an assistant foreman, and $10, if a foreman.

6. "Any mine foreman or assistant foreman, who for 6 consecutive months is entitled to the monthly premium of $10 or $5 under the present rules, will at the end of the sixth month receive a special premium of $15 or $10, and for each month thereafter so long as his record is up to the requirements under the present rules, but when his record does not come up to the requirements under the present rules, he will have to again make a clear record for another 6 months before he is again entitled to a special premium.

7. "The foremen and assistant foremen have it distinctly pointed out to them by their immediate superior what men or jobs are under their supervision.

8. "If a foreman or assistant foreman leaves the employ of the company and later reënters it, he assumes all demerits charged against him when he left the company."

Since the adoption of the premium system in May, 1909, the company has paid 2,275 premiums, amounting to $15,335. During this time they have increased the number of tons of coal produced per fatal accident inside over 400 per cent. Of course, this large increase cannot all be attributed directly to the premium system, but there is no doubt that it materially aided in accomplishing this result.

No assistant mine foreman is allowed to have under his charge more than 25 to 30 men, depending upon the way the places are scattered, in order to enable him to visit each working place at least every 3 hr. during the day. This gives the assistant mine foreman ample time, in case he sees any dangerous slate, a post that needs setting, or anything of a dangerous nature, not only to instruct the workmen to do it, but to stay with them until the post is set, or the dangerous conditions

are removed. Upon two particular occasions the company increased the number of men under the charge of each assistant mine foreman, with the result that in each instance an increase of accidents occurred.

The New Jersey Zinc Co. tried giving cash prizes at the end of 6 months to the shift boss who had the best record for freedom from serious accidents during that period, having a numerical rating for all accidents, closely following that used in reference to compensation insurance in the State of New Jersey. The element of chance entered into this plan to such a degree that they soon decided to distribute $10 monthly to every boss who attained a record better than an arbitrary standard, fixed at a rating of 1.2 disabilities per 1,000 shifts of labor worked in his gang during the month. It was further decided to distribute safety cigars at the end of each month to the members of the gang having the best record.

A modification of the bonus system has met with some degree of success at one or two iron mines, which require all shift bosses to be safety enthusiasts, and well versed in the most modern methods. For this service they have increased the wages given to shift bosses. However, this method lacks the incentive of a cash reward for a good record, and introduces a disagreeable element in the possibility of having to dismiss relatively good men. It is doubtful if such method would meet with any success, unless adopted by a company operating in a district where there were a large number of companies which had not adopted the plan and which paid a smaller wage to their own shift bosses.

Precautions that are of universal interest to coal operators may be revealed by studying the rule books printed by companies operating in different parts of the country, and under the laws of the various States.

Many of the dangers in mining cannot be prevented by mechanical devices, therefore it is of interest to note some of the causes of underground accidents, and the means sometimes adopted to lessen these dangers. The following summary of safety devices and regulations adopted by the H. C. Frick Coke Co. will illustrate a few of these points:

Rescue stations, and 33 rescue teams which practice quarterly.

General rules and regulations providing increased air at working faces and at intake above that required by the State mining law.

One hundred and eighty first-aid teams below and above ground.

Bore holes from surface to release any dangerous accumulations of explosive gas in gobs.

Shot firers and inspection of all places where shots have been fired to see that there is no fire or other danger.

Permissible explosives.

Blasting by battery.

A system of pipes for sprinkling dust in dry places.

Employment of steady, reliable and sober men only, for responsible positions.

The best and safest oils that can be procured for illuminating purposes in the mines and mine buildings, etc.

All mine buildings to be well ventilated and kept clean and neat.

Cans for storing oily waste, etc.

Open lights in all buildings prohibited.

Electric wiring to be well done and examined at least twice a year.

Proper fire apparatus and fire brigades.

Finger boards and arrows pointing way out of mine.

Mines visited and thoroughly inspected by company's mine inspector at least once in 60 days.

Frequent meetings at the plant of superintendents and bosses to exchange views and discuss conditions, etc.

The examination of safety-lamp mines on Sundays, holidays and lay-off days. The mines which have been idle for more than 2 consecutive days are examined before operations are resumed.

Religious and political opinions of workmen not to be interfered with.

Employees must have permission to be absent from duty.

Mine officials are to see that "turns" are properly and equally distributed.

Care to be exercised and economy in use of materials and supplies.

No discriminations on account of nationality or creed.

Bristol recording and "U" gages on fans to be kept in good condition.

Safety switches at bottom of all steep butts.

Masonry stoppings between all air courses.

Fencing off all abandoned places in mines.

Systematic timbering.

Use of post extractors.

Automatic uncoupling device on rope haulages where they can be used.

All shafts where men are hoisted are equipped with device to prevent overwinds.

All machinery fenced off and shop machine belts, etc., guarded.

Stables, pump rooms, haulage engine rooms and shaft bottoms underground are all of fireproof construction.

Printed pamphlets "Safety Regulations—Machinery" provided for the guarding of all machinery, and posting of danger signs, etc.

Additional shafts to aid ventilation in mines and provide escape way.

Connections between mines as an additional escape way.

Gates at surface which cannot be opened until cage is there.

Safety catches at cage landings.

Safety bars on cages. Introduction of concrete-lined shafts.

Safety cabs on electric larries. Steel rail throughout all mines.

Ascensional ventilation for mines. Testing for gas above roof in gobs.

Prohibiting work in any place in which gas is discovered.

Printed pamphlets in five different languages showing how to do work in safety and guard against accidents.

As mentioned above, this company has issued a pamphlet entitled "Safety First," which is printed in the different languages spoken by its employees and illustrated by reproductions of photographs, which clearly point out the dangers of not following the instructions as laid down. The proper way of performing various duties and the outcome of carelessness are strongly impressed upon one's mind by these underground photographs, for which men have posed as if performing their

work in a careless manner with injury or death resulting. Footnotes describe these pictures and important instructions are printed in large type.

Photographs especially illustrate the danger of men walking along haulage-ways and being crushed by trips; of failing to put clevis-blocks on rails when loading cars in dips; of bad joints in tracks; of riding in front of trip, or between cars; of not standing in a safe place when cars which have jumped the track are shunted on again; of pushing cars with hands on the corner; of not obeying signals and waiting for trip coming in an opposite direction; of coupling cars on the inside of a curve where one would be caught and crushed if trip began to move; of the dangers of shock from trolley wires, if men carry augers, picks or shovels over their shoulders, should trolley wire guard be broken.

One company operating in the West Virginia section requires a systematic standard amount of timbering, whether the top is considered dangerous or not, and extra timbering where unusually bad conditions prevail. By this requirement, they do not have to rely so much upon the judgment of the men as to whether or not the top is dangerous. This same company requires a clearance of not less than 2½ ft. on each side of the mine cars along all haulage-ways. In some cases, water is piped to all workings to dampen the roof and side and wet down the dust.

Safety Signs and Signals

It is necessary to have signs to warn men of dangers, and to point out places of safety, and means of exit. In view of the fact that many languages are spoken by the laborers of this country, and that the miner is naturally a great wanderer, it is highly important that some universal danger sign that needs no verbal explanation be adopted. Mr. Higgins suggests that the solid red circle be accepted as a universal sign, as it appears to be in every way adapted to underground use. He further suggests an arrow as a direction sign to exit along manway, etc., and a symbol of a ladder, to indicate a ladder-way or stairway. It is, however, impossible to cover all the requirements of danger signals by symbolic signs, but whatever wording is used should be as brief and explicit as possible. Too much variation in signboard instructions leads to confusion. It seems as though some distinction should be made between a danger sign and those denoting warnings or bearing instructions, as the red ball should instantly impress upon the mind the necessity for immediate caution. A few of the signs observed by Mr. Higgins in various parts of the country are described by him as follows:

Signboards for Points on the Surface

"Danger: Keep Away" (around the boundary of surface caves).
"Do Not Walk On These Tracks."
"Danger; Keep Away From Shaft Collar."

"No Smoking Allowed Around Shaft Collar."
"Lighted Candles Or Lamps Not Allowed On Main Cage."
"Look Up" (under trestle or tipple where rock, ore or coal is handled).
"Keep Out."
"Careless Workmen Not Wanted Here" (on office).
"No Admittance Without Permission From Office."
"Live Wire."
"High Voltage. Do Not Touch."
"Safety First."
"Riding On Ore Skip Positively Prohibited."
"Riding On Incline Forbidden."

Signboards for Shaft Stations

A placard showing complete code of hoisting signals.
"Do Not Ring To Move Cage Or Skip When Men Are Working In Shaft Or Sump."
"Not More Than — Men Permitted To Ride In Cage."
"Cage Doors Must Be Closed When Men Are Being Handled."
"No Loafing Around This Station."
"Throw All Rubbish Here" (at entrance to small opening off station).
"Miners Not Permitted To Ride On Motor."
"Look Where You Are Going—This Means You."
"Wrestling, Pushing Or Crowding Positively Prohibited."
"Safety First."
"2,000 Level" or "2,500 Level."
"Look Out For Motors" (Locomotives).

Signboards for Pump Stations

"Keep Oily Waste In Receptacle Provided."
"Do Not Oil The Timber—Oil The Machinery."
"Do Not Hang Candles Or Lamps On Timber."
"Keep This Place Clear Of All Rubbish."
"Beware—Live Wire."
"High Voltage—Do Not Touch."

Signboards for Mine Workings

"To 2nd Outlet Shaft" (with arrow or hand pointing).
"Ladder-way To 12th Level."
"To Main Shaft" (with arrow pointing).
"To Timber Shaft" (with arrow pointing).
"Dangerous Ground—Keep Away."
"Danger Overhead."
"Look Out For Trolley Wire" (at points where timber carrying the trolley wire is sagging, it is assumed that the wire has guard boards wherever men are at all likely to come in contact with it).
"Look Out For Motors."
"Danger—Dynamite Stored Here."
"Powder Magazine."
"Danger—Blasting Here."
"Danger Ahead—Old Workings."

"Ladder-way To Surface."
"No Smoking Allowed."
"Danger From Ore Trains—No Traveling In This Tunnel" (or Drift).
"No Candles Or Lamps Allowed In Magazine."
"No Candles Or Lamps Allowed In Stables."
At entrance to a gaseous mine, or district: "Open Lamps, Pipes, Cigars, Cigarettes Or Matches Must Not Be Taken Into This Mine" (or District).

The Tennessee Coal & Iron Co.,'s committee on the wording of "safety signs" is as follows:

The wording of signs should be as brief and as much to the point as possible The word "Danger" should be prominent on all signs put up at dangerous points. The word "Warning" should be prominent on all signs as to how an employee can best do work with safety.

The following signs are considered applicable to all works:

1. "Notice—Stop Machinery Before Oiling, Wiping Or Repairing."
2. "Elektrika."
3. "Danger" (a tag).
4. "Danger—Look Out For Cars."
5. "Danger—Keep Out."
6. "Danger—Will Not Clear Man."
7. "Danger—Keep From Under Loads Carried By Crane."
8. "Commit No Nuisance. No Loafing Allowed" (six different languages).
9. "Danger—You are warned against working without proper eye protectors, or with battered tools. Get proper tools and eye shields from your foreman."
10. "Warning—Employees working around engines, moving or revolving machinery shafting, etc., are warned of the danger, and are prohibited from wearing torn clothing, loose or unbuttoned jackets, blouses, shirts, long neckties, and loose sleeves. Always wear jackets tucked in the trousers or under the overall bib. Never forget to examine your clothing before commencing your work."
11. "All persons are forbidden to enter any cylinder of an engine until flywheel has been securely blocked, to prevent possibility of engine turning over."
12. "Danger—No Smoking Allowed."
13. "Danger—Do Not Move This Valve."
14. "Danger—Man in Boiler."
15. "Danger—Keep Off."
16. "Danger—Do Not Walk On Track Or Trestle."
17. "Engine Room Rules" (printed on cardboard and framed—copy attached).
18. "Boiler House Rules" (printed on cardboard—copy attached).
19. "What To Do In Case Of Electric Shock" (printed on cardboard and framed—copy attached).
20. "Danger—This is electrical apparatus and carries dangerous electrical current. All persons not especially authorized to work on the same, are hereby prohibited from doing so. Failure to comply with this warning may result in death" (five languages).

The following signs are applicable to all mines:

1. "Notice!—All persons are hereby warned not to travel on the slope, or ride on loaded or empty cars. Any one failing to comply with this notice will be discharged. You must travel the manway."

2. "Danger—Magazine."
3. "Notice! No Powder, Fuse, Caps Or Oil Allowed In The Wash House."

The following signs are applicable to the different works:

Ensley Works

1. "Notice! Men Must Take Fuses Out Before Beginning Work On Or Around Crane, And Return Same When Through."
2. "Danger! 6,600 Volts. Do Not Climb This Tower."
3. "Danger! This Platform Is Dangerous When Pouring Heat."
4. "Danger! Do Not Pass Between Wall And Track."
5. "Danger! Look Out For Larry Car."
6. "Danger! Employees Are Warned Against Crossing The Run-out Tables. Always Use Stairs."
7. "Danger! No Smoking Or Bare Lights Allowed In This Room."

Ore Mines

1. "Warning! No Traveling On This Slope. Must Use Manway."
2. "Danger! Do Not Go Inside Or Beyond This Notice Until Engine Is Stopped."
3. "Bell Signals" (copy attached).

Coal Mines

1. "Danger! Do Not Open Gate."
2. "Danger! Do Not Get On Cage Without O. K. From Cager. Ten Men Are A Load."
3. "Danger! Do Not Get On Cage From This Side."
4. "Escape-way To No. 6 Slope."
5. "Escape-way To No. 2 Slope."
6. "Escape-way To No. 1 Shaft."
7. "This Way To No. 1 Shaft."
8. "Escape-way To McArdle Slope."
9. "This Way Leading Along Fault To No. 6 Boilers."
10. "Warning! No Open Lights Allowed In This Stable."
11. "Danger! Keep Out! Gas!"
12. "Danger! Do Not Exceed 4 Miles Per Hour."
13. "Notice! No Loafing. For Signal Man Only."

Foundry Furnaces

1. "Do Not Pass Between Column And Track."
2. "Danger! Keep Out! 2,300 Volts."
3. "Notice! Danger! Do Not Touch."
4. "Danger! Keep Off Coal Bin."
5. "Danger! Keep Off Of Track."

Bessemer Rolling Mill

All signs reported in use at Bessemer Rolling Mill are general signs.

All signs should have the sanction of the department superintendent before being placed. Care should be exercised to see that all signs can be complied with. None should be placed unless they are enforced.

All signs should be made of enameled 18-gage steel. The groundwork should be white, with blue block letters and blue margin.

The size of letters in the words "DANGER" and "WARNING" should be 2 in. in height; balance of the letters 1 in.

We would suggest that all signs be made 8½ by 11 in. or some multiple of that size.

The minimum space between the letters shall be three-fourths of the width of the lines of the letters. The minimum space between words shall be three and one-half times the width of the lines forming the letters.

Safety Guards

Many of the machine guards and other safety devices used in power plants, and in factories of all kinds, are likewise serviceable throughout the mechanical and metallurgical departments of a mine. Wherever exposed gears, cutting tools, fly-wheels, revolving shafts, belts and grinding or emery wheels, are within the reach of those working nearby, there is danger of serious injury, unless proper guards are installed. The general standards recommended by the committee of safety engineers of the Workmen's Compensation Service Bureau descrrie the equipment approved by the insurance companies and State inspectors. The book published by this bureau will be found most helpful, when providing guards for machinery already installed. If new machinery is to be purchased, every effort should be made to eliminate dangerous construction, and the specifications should embody safety requirements. In most cases it is cheaper to design a machine for permanent safety than to annex guards after it is in place.

According to recommended standards, the following danger points are the usual ones needing protection:

All gears and all friction clutches within 7 ft. of floor or platform should be completely encased by metal or close-meshed wire screen.

Vertical shafts should be encased to height of 6 ft. from floor or platform.

Horizontal shafting and dead ends of shafts less than 6 ft. from the floor or platform should be encased or protected by railing.

Horizontal belts within 6 ft. of floor to be guarded on top and sides. Overhead belts within 7 ft. of floor to be guarded beneath and on both sides; those more than 7 ft. from floor to be guarded beneath.

Vertical and inclined belts should be either completely inclosed, or with a railing 3½ ft. high, and 5 in. from belt.

Belt shifter should be provided.

Belt pulleys within 7 ft. of floor should be encased or properly protected by railings.

Self-oiling bearings and shaftings provided.

All protruding or sunken parts on rapidly moving machinery are exceptionally dangerous.

Remove or guard revolving bolts or open keyways. Where keyways are essential, carefully fitted pieces of wood may be inserted.

Dead ends of shaft may be easily encased in pipe fittings. A cheap and novel means of encasing shafting is described in the "safety" supplement of the November issue of *The American Industries*. Ordinary three-ply mailing tubes the diameter of the shafting are cut to fit the length and split to slip over the shaft. These are fastened on by means of gummed paper, and should be shellaced to give added strength and durability. They may be wound spirally with tape, if greater strength is needed. These guards revolve with the shaft, but should a workman accidentally come in contact with the tube, it will cease revolving, thereby eliminating the danger of clothing being caught in the mechanism.

A personal knowledge of a number of large plants, and a set of photographs from The New Jersey Zinc Co., bring to mind the following appliances and brief details of their construction, viz.:

Place double-bar iron guardrails, 3½ ft. high, and provide with toeboards wherever there is an opening to a working place below:

Around all flywheels.

Around motors and dynamos, or least on side where men have to pass.

Around belts near floors.

Around permanent floor openings.

Around all open tanks.

Along the outer edge of all platforms, stair landings, bridges, open mezzanine floors, and open pits.

In front of hot refuse piles or slag heaps (with danger sign).

In front of doors or pathways near railroad tracks or other places where accidents occur from men stepping directly in front of moving cars, trips or cranes. (The necessity of walking around the end of such railings allows time to observe approaching danger.)

In front of engine blowers and grinding wheels.

Movable rail guards or wire-mesh gates installed around open elevator shafts, trap doors, etc. These should be automatically placed in position when openings are otherwise left unguarded.

Handrails in stairways, on all open sides; on one side of closed stairways less than 4 ft. wide; on both sides between 4 and 8 ft. in width and on both sides and in the middle for over 8 ft. wide.

Permanent ladders to have handrails at the top, and platforms at every 10 ft. in height.

Portable ladders provided with safety shoes (spikes for wooden or dirt floors and automatically adjusting flat shoes for concrete or stone floors).

Vacuum exhaust system for shavings, sawdust, grinding dust at all saws, lathes and emery wheels.

Locked gates leading to space back of electrical switchboards, and rubber mats near all exposed parts.

Safety belts for men working in hoppers.

Safety wrenches for opening side-bottom dump cars.

Safety lid opener for ore cars.

Safety grab hook for handling hot pokers.

Safety coupling tools for brakemen.

Goggles worn by men exposed to eye accidents.

Eye protection furnished where high heats have to be observed.

Hand warnings on tracks in front of moving cranes and machine carriages.

Safety catches on elevators and cages, and overwind guards on all hoists used by men.

Good light and ventilation add greatly to the safety of all working places.

Note.—The Ingersoll-Rand plant at Phillipsburgh, N. J., is ideal in this regard. Mr. Zook, the construction engineer, has designed the buildings so that no working rooms are more than 60 ft. wide, except in the machine shop, where sawtooth roofs with northern exposure, have been adopted. Steel-sashed windows have added greatly to the lighting conditions in the new packing building.

Electrical Equipment Underground

Electricity is today an important element in underground development, and owing to the conditions surrounding such installations, there are dangers which seldom enter

into other lines of electrical equipment. The presence of underground water, gas, explosive dust, and explosives used in mining, adds to the dangers of fire or shock. Owing to the constant changes made necessary, as underground workings are extended, it would be impossible to install installation that would be employed if the wiring was to be permanent.

At isolated properties, men with little knowledge of electricity are necessarily employed to equip the mine, and men with no knowledge or realization of the dangers associated with high potentials are in constant peril, and even experienced men are liable to accident due to poor insulation, low trolley wires, improper inspection and the cramped quarters in which they have to work.

Completely insulated circuits are far less dangerous than those that are grounded, as in the former case the current can be confined to its proper channel if adequately insulated. Trolley wires are especially dangerous, as there is always the possibility of men making a grounded circuit by either coming into contact with the wires themselves, or having tools accidentally touch them. Equipment may become charged if insulated from the ground. Improper ground connection is often caused by sanded tracks or excessive amounts of oil, in which case locomotives and cars may give to men standing on the ground as severe a shock as the trolley wires would.

Fires may be caused by short-circuits due to defective insulation, or combustible matter placed near fuses and other portions of the circuit where arcing may take place. Coal and wood may be ignited in this fashion. Explosions may be caused by the ignition of explosive gas and coal dust, if an electric spark of sufficient intensity passes through a dry atmosphere containing these elements. Explosives are frequently accidentally discharged either in the act of being transported upon electric haulage-ways, or by accidental detonation. It is always dangerous to use a grounded circuit for firing purposes, as the difference in potential between the explosive and the ground at a slight distance away may cause accidental detonation before battery connections are made, if an exposed surface of the wire should come into contact with the ground.

The underwriters' rules regarding electrical installations will be found of great assistance to any mine manager who has to deal with electricity for either power or lighting purposes.

Rules for Employees

From the various rule books at hand, the following are submitted for approval, correction, and additions. This list and grouping is the result of careful study, rearrangement, rewording and elimination, but those in charge of the safety work at the various plants will no doubt be able to make valuable suggestions owing to their daily familiarity with the practical conditions and unusual emergencies calling for special regulations.

It must be borne in mind that the rules are intended as a guide to others in formulating their own sets and consequently cannot provide for local dangers, while, on the other hand, it will include rules that can only be adopted by a few.

General Rules on Safety

1. Each and every employee is hereby instructed to report to his foreman any defective tools or appliances or any feature of his working conditions or surroundings which he may think affects the safety of himself or others. Do not work with tools or appliances of any kind that are defective.

2. If you are injured, no matter how little, you must report or get word to your foreman at once. A slight scratch often causes blood poison. (Report or have foreman report all injuries to the doctor.)

3. If you find anyone injured or sick on the property of the company, whether an employee or not, take him to the doctor.

4. Do not report for work when under the influence of liquor. Intoxicating liquors about the works are strictly prohibited.

5. Vigilance and watchfulness insure safety. To avoid danger adopt the safe course. An employee must not trust to the care exercised by another when his own safety is involved, but, on the other hand, always consider the safety of others.

6. When working above others great care must be taken not to drop any material without first giving warning to those below. When you are going to either work above or below other men, let those men know about it.

7. Never throw down any material without first looking to see that it will not hurt somebody.

8. Neatness and order are a safeguard against accident. Boards with nails protruding must not be left around the mine or premises.

Construction men must clean up all loose boards after them, but all men are requested to remove dangerous loose boards or planks they see and report to foreman.

9. Do not leave bars, tools, or material of any kind, leaning against a wall so that they are likely to fall and hurt some one.

10. All employees, except motormen, switchmen, trammers, are strictly forbidden to ride on moving cars.

(NOTE.—Designate those authorized to ride.)

11. Do not get on or off any elevator while it is in motion.

12. When working around machinery in motion do not wear gloves or clothing with loose ends or loose fitting coats or sleeves.

13. Do not reach or lean across moving belts or gears.

14. Do not set in motion any machinery without first seeing if anyone is in a position to be injured and whether all safety guards are in their proper place.

15. No one must undertake to put on or remove a belt while the wheels driving it are in motion, except men skilled in such work, and they must always use the proper appliances for the purpose.

16. Never undertake the removal of a belt which may have become wrapped around a shaft or pulley while such is in motion.

17. Keep from under loads suspended in the air.

18. Any workman removing a cover from any opening in floors, tunnels, ground or pit, or any man removing a brattice or stopping must see that it is properly replaced before leaving the place.

19. Do not start machinery or turn on electric connection without first seeing if anyone is in a position to be injured and whether all safety guards are in their proper place.

20. No one must handle explosives except men skilled in their use and properly authorized, and all precautions must be taken to prevent accidents from blasting. Explosives must not be left exposed or unguarded.

21. Do not use defective ladders, or stairways. Notify your foreman.

22. It is dangerous to permit molten matte or metal to come into contact with water, and you are warned to exercise great care to avoid such danger. Dangerous explosions may result from carelessness in this regard.

23. All employees shall use the regular roads and stairways for travel. They shall look out for moving cars on both the motor and railroad tracks. Keep off railway tracks except at regular crossings.

24. Any conduct which may be termed "fooling," such as wrestling, boxing or throwing pieces of loose material, is strictly forbidden.

25. Do not fool with compressed air and never blow compressed air on anyone.

26. When you complete a job never leave tools or materials overhead, and always replace safeguards.

27. Never rush from the light into a dark place or from the dark into a light place. Go slowly, use caution.

28. Do not touch electrical machinery, power or electric light wires, unless you are properly authorized to do so.

29. No person in a place of trust shall deputize another to do his work without the sanction of his superior officers.

30. No person in a place of trust shall absent himself from work without a legitimate excuse, or having first obtained permission from his superior officer, and when possible he must send notice.

Mining Department

1. Every working place shall be regularly inspected and employees who find anything in an unsafe condition shall report same to their foreman.

2. The general condition of the timbering in the mine shall be made safe and kept so.

3. A miner's first duty shall be to pick down the roof and walls, and not permit the shovelers or other employees to work under a place until it has been tested and made safe for them.

4. If an employee drops any material or tools down the shaft or deep working place, he shall immediately report same to the hoisting engineer, who will have the shaft inspected before continuing with the regular work.

5. In stopes worked by the square-set method, the working floors shall be securely lagged over, and the lagging shall be long enough to reach halfway across the caps or girts.

6. Every winze, raise or incline of steeper slope than 40° from the horizontal and deeper than 40 ft., through which men are obliged to travel, shall be provided with a ladder-way.

7. Suitable ladders or footways shall be provided to connect floors in stopes and other places requiring communication in the mine.

8. Ladder-ways shall be provided in all shafts in the course of sinking to within such distance from the bottom as will secure them from damage by blasting, and from the end of such ladder-way portable ladders shall be extended to the bottom of the shafts.

9. Broken or defective ladders shall be reported and repaired immediately.

10. Ladders shall not be removed from their usual places without orders from the shift boss.

11. Employees are forbidden to throw tools, steel or any other material down the manway, but shall lower them with a rope provided for that purpose.

12. Carmen shall exercise care when loading out of a chute, also when pushing cars, that their hands may not be crushed against the timber.

13. (Winzes or raises shall not be started in the direct line of a drift, but shall be offset from the drift.)

14. Every winze, or raise, now opening from below directly on any drifts or tunnels traveled by men shall be covered by a grizzly or by doors. The opening of such offset winzes shall be protected by a fence or guardrail not less than 3 ft. nor more than 4 ft. in height above the level of the drift.

15. Men shall learn the various exits and raises or winzes connecting the level on which they are employed, with other levels.

16. No employees except the motormen and switchmen shall be allowed to ride on a motor or motor train.

17. When men other than those employed on the motor are using the motor track, they shall notify the motormen accordingly, for the motor has the right-of-way.

18. Employees are forbidden to carry tools, steel or other material upon their shoulders in any workings where there are electric light or trolley lines. This is dangerous and may result in death.

19. Stationary lights shall be provided at all stations in the working shafts, and at night at all working places on the surface.

20. Employees shall not enter a drift, stope or other workings, where powder smoke, gas or bad air is such as would be injurious.

21. Employees shall use great care in keeping all underground ventilating doors closed.

22. No candle shall be left burning in a mine or any part of a mine when the person using the candle departs for the day.

23. All compartments of shafts used for the hoisting and lowering of men and ore shall be inspected at the beginning of each shift to see if there are any broken guides or other defects. Any defects, if found, shall be reported and repaired before further use of said compartments.

24. Men working in the shafts shall have a suitable covering to protect themselves from material falling down the shaft.

25. They shall give full information concerning the place and nature of their work, to all hoisting engineers on duty during the time they are employed in the shaft, so that the cage will not be let down on them.

They shall have their working platforms of sufficient size and strength to safely carry on their work.

26. In no case shall a cage, skip or bucket be lowered to the bottom of a shaft when men are working there. It must be stopped at least 15 ft. above the bottom until the signal to lower further has been given by one of the men at the bottom of the shaft. (This rule shall not apply to shafts less than 50 ft. in depth.)

27. No open hook shall be used with a bucket in hoisting, but only some approved form of safety hook or shackle hook.

All shafts or winzes shall be cleaned down below the bulkhead after each blasting.

28. The top of every shaft shall be protected by a substantial guardrail or chain.

29. At all shaft stations a gate or a guardrail shall be provided and kept in place across the shaft except when a cage is being used.

30. All chutes, manways, winzes, raises and other openings shall be covered by a substantial hatch, planking, or grizzly, or provided with guardrails or chains, and shall be kept in such a condition that men cannot walk or fall into them.

31. Employees shall not crowd or rush while getting on or off of cages.

32. When using the cage, all signals shall be given while standing on the cage and not upon the station.

33. To release the cage the signal shall be given while standing upon the station and not upon the cage.

34. Powder nippers shall be very cautious in the handling of explosives, both underground and on the surface. Before using the railroad or motor tracks they shall make sure that they have the right-of-way. They shall also use extra care in handling the powder on the cages and in all other places.

35. Employees shall be careful in the opening of boxes of powder.

36. All powder used in stopes, raises, and other places where it is necessary to hoist or lower it with a rope shall be carried in sacks provided for that purpose. A box may slip out of a rope and its use is, therefore, forbidden.

37. All blasting powder used shall be properly marked with the date of manu-

facture on each stick, and no powder shall be used after 12 months from the date of manufacture.

38. All explosives shall be stored in a magazine provided for that purpose alone; the magazine shall be electrically lighted and shall be placed far enough from the working shaft, tunnel or incline to insure its remaining intact in the event of the entire stock of explosives in the magazine being exploded. No powder or other explosives shall be stored in underground workings where men are employed. All explosives in excess of the amount required for 24 hr. work must be kept in said magazine, and such temporary supply shall not be kept at any place within the mine where its accidental discharge would cut off the escape of miners working therein.

39. Smoking in the magazine and while carrying powder is forbidden. The carrying of lighted candles or lamps in the magazine is also forbidden.

40. Oils or other combustible substances, or blasting caps, shall not be kept or stored in the same magazine with explosives.

41. Cap crimpers shall be furnished and must be used. Crimping with the teeth is dangerous and is forbidden.

42. Skewers shall be furnished for the use of all miners handling explosives and shall be used for making holes in the powder for the fuse. No candlesticks or other instruments shall be used for this purpose.

43. All employees are forbidden to use anything other than a wooden tamping stick in the loading or charging of a hole with blasting powder or other high explosive.

44. Tamping shall be done by pressure and not by strokes.

45. Miners shall never be alone when blasting, but must keep at least one man with them. They shall have a reserve light near a manway or some other convenient place in case their lights go out after "spitting" the fuse.

46. Before firing charges, warning must be given in every direction from which access may be had to the place where blasting is going on.

47. The number of reports shall be counted by the miners firing same, and misfire holes shall be reported to the timekeeper and posted on a blackboard made for that purpose. The shift bosses shall also notify their men working in places where misfires have occurred.

48. No employee shall be permitted to enter a place where a misfire has occurred until 30 min. after the time when the fuse was lighted.

49. Employees shall not attempt to extract explosives from a hole where a misfire has occurred, or in which all of the powder has not exploded. In such a case, a new charge shall be added before detonation.

50. Where boulders are being blasted by placing explosives on them and not in holes drilled for that purpose, employees shall see that the powder is well covered with clay, or some other suitable material, so that it cannot take fire.

51. Where boulders at the base or on the slope of a muck pile are so charged, the employees in charge of the work shall not permit any person to disturb or knock down any loose rock from above, as it might cause an explosion.

52. No employee other than the motormen and switchmen shall be allowed to ride on a motor or motor train.

53. Motormen shall ring their bell and slow up when coming to curves, switches, and doors.

54. They shall have an electric light on each end of their motor at all times.

55. They shall report any defects in their equipment to the electrician or to the foreman and have it repaired.

56. All cables used for hoisting purposes shall be of approved quality and manufacture.

57. All cables shall be daily inspected by some competent person appointed by

the mine superintendent, and any defects shall be immediately reported and the ropes changed if necessary.

58. Ropes shall be kept plainly marked for the benefit of the hoisting engineers.

59. All cages used for the hoisting or lowering of men in shafts deeper than 300 ft. shall be equipped with iron bonnets, with gates at least 5 ft. in height, with overhead bars of such arrangement as to give every man on the cage an easy and secure handhold. They shall also be equipped with safety catches of sufficient strength to hold the cage at its maximum load at any point in the shaft in case of the breaking of the hoisting cable.

60. Emergency chains shall be used from the cable to the cage in case of a breakage in the king bolt or clevis pin; and also between the upper and lower decks in case of a breakage of the connecting pins for these decks.

61. All cages, together with the king bolt and clevis bolt, shall be daily inspected by some competent person appointed by the mine superintendent. They shall be kept in first-class condition and all defects shall be reported and repaired before further use.

62. The safety catches on cages and crossheads shall be kept well oiled and in good working condition.

63. The cage men and top men shall keep a careful watch over the cages during their shift's work and immediately report all defects and have them repaired before continuing their work.

64. At no time shall more than (the number allowed depending upon size of cage) men be permitted to ride on the deck of a cage.

65. No person shall ride upon a cage loaded with tools, timber, powder or other material except for the purpose of assisting in passing these through the shaft.

66. No person shall ride on a cage loaded with rock or ore.

67. When hoisting or lowering tools, timbers, or other material in the shaft, the ends, if projecting above the top of the cage or bucket, shall be securely lashed to the cable or to the upper part of the cage; and tools, timbers or other material loaded erectly upon a cage shall be securely lashed before being hoisted or lowered.

Surface Department

1. If you remove the cover from a valve pit, sewer, or any manhole, see that the opening is guarded so that no one can fall into it. Holes caused by excavation, or any other cause, must be similarly guarded.

2. Always use cross bracing in excavating where there is any danger of a cave-in.

3. Do not remain near where blasting is being done.

4. Use care in placing ladders before using them. If there is any danger of the ladder slipping have someone hold it.

5. Never try to climb a ladder without the free use of both hands. If material is to be handled use a rope.

6. Never rush from the light into a dark place or from the dark into a light place. Go slow; use caution.

7. Do not attempt to jump on a moving car.

8. Always see that the brakes are set before starting to unload a car, and never leave a car set out without setting the brakes.

9. When opening dump cars, exercise great care so that the dumping lever will not hit you.

10. Do not walk alongside of cars when material is being thrown from them. It may strike you.

11. In unloading cars, material should be piled at least 4½ ft. from the rail. If it is impossible to do this, place a warning sign at each end of the pile. Any material that may fall upon or alongside the track must be cleaned up immediately.

12. Do not pile any material so high that it is liable to fall or cause another pile to fall.

13. Care should be taken in loading material on cars so that no portion will project over the sides, or fall off, while in transit.

14. Do not take hold of rope or cable above a block, as your fingers may be drawn into the block and injured.

15. Never leave a derrick boom so that it can be swayed by the wind.

Boiler-House Department

1. Report all defective conditions to superintendent of plant.

2. Two men should always work together when cleaning out boilers, it being the duty of one man to remain outside of the boiler all the time in a position to see the man working in the boiler and give assistance in case it is needed.

3. Never go into a boiler until you have locked the steam valve and closed all other valves.

4. Before entering a boiler put a torch or candle inside to determine the presence of gas or bad air.

5. A sign "DANGER, DO NOT MOVE," should be hung on the steam valve of the boiler when it is shut down.

6. Safety valves should be tested every shift.

7. Never tighten any bolts or nipples or do any calking while steam pressure is on the boiler.

8. In turning a boiler into the main steam line after being out of service, the steam pressure of the boiler should be brought to within 5 lb. of the main-line pressure and the valve next to the boiler should be opened first to allow any water in the connecting pipe to flow back into the boiler, after which the valve next to the main should be opened very slowly until the pressure is equal in boiler and steam line. When boilers are off for cleaning, the stop valves next to the main must be closed. No dependence is to be placed on any automatic valve alone. These latter valves must be examined each time boiler is out of service, so as to insure their working in emergencies, such as the bursting of a tube.

9. No one should enter a boiler without first making sure that all valves are provided with signs "DO NOT OPEN," or locked shut before any one is allowed to go inside boiler.

10. Where more than one boiler is connected solidly into one blow-off main, the boiler cleaned should be locked shut before any one is allowed to go inside boiler.

11. To prevent water tenders being injured by a glass bursting, water glasses should be provided with a No. 16 brass coil spring, about $\frac{1}{16}$ in. greater in diameter than glass and with about $\frac{5}{16}$ in. between separate turns so that the water level can be easily observed.

12. Combustion chambers must be kept free from accumulation of clinkers, ash and dust.

13. If the water forms a hard scale, the use of soda ash may be considered, or the use of good mineral oil, such as kerosene, may prove of benefit. About a $\frac{1}{2}$ gal. of each 100 H. P., should be placed in the boiler when empty; the addition of water causing the oil to follow the surface of the shell and loosen scale.

14. Each boiler plant should be equipped with the proper tools for firing, such as rakes, slice bars, steel poker, and a proper size of shovels (about No. 3). A proper place must be provided in which to keep all tools, and they must not be allowed to lie around promiscuously.

15. The boiler, when being taken out of service, should be cooled down as much as possible before being blown out; as the practice of blowing out hot water and filling

with cold water and blowing down, will cause the scale to bake on the tubes and shell plates from the heat remaining in the setting, as well as cause serious strains on shells.

16. Boilers should be blown at least twice every 24 hr. (more often with bad water) while in operation, and special care should be taken to see that the blow-off valves are kept from leaking. Feed valves must not be allowed to leak to such an extent that boilers have to be blown down to prevent high water.

17. Gages must be tested sufficiently often to insure corrections.

18. All boilers must be equipped with three gage cocks, and a water glass; except locomotive boilers, which only require gage cocks.

19. Bottom water valves must be provided for blowing out gage glasses.

20. Thermometers should be installed in feed lines of all plants and an effort made to attain the highest possible feed temperature. It might be stated here that each 10° heat added to the feed temperature means approximately 1 per cent. saving in coal, as well as increasing the capacity and reducing cost of cleaning and repairs to boiler.

21. In all horizontal return-tubular boilers the tubes must be headed at both ends.

22. In all water-tube boilers the tubes must extend through sheet about $\frac{5}{16}$ in. and flared to an angle of about 35° with the straight portion of tube well down against the tube sheet. This work should be done with a regular expanding tool.

Blast-Furnace Department

1. Men working in limestone, ore, or coke bins must notify the foreman of the unloading department as to the bin in which they will be working. When working in a bin always use a safety rope, fastened above so that it will not be cut in two by a passing train.

2. In loading cars, be careful not to get struck by pieces of ore or limestone bouncing off the car.

3. Do not walk out from behind a furnace on the charge floor without making sure that the track is clear.

4. When handling a hook or bar in the furnace, be careful not to strike the trolley box.

5. When pulling a short bar with the chain along the length of the furnace, get far enough away from the bar so as not to be dragged in, in case the crust breaks and the bar drops into the furnace.

6. When breaking blast-furnace crusts do not attach the chain to the locomotive.

7. Do not use too short a bar or too short a hook.

8. Do not use wet or moist bars around tap holes, spouts, or nose pieces.

9. Protect your eyes when digging away crusts from spouts and nose pieces.

10. Don't walk over settler roofs without first seeing that they are covered with sheet iron.

11. In tapping out a furnace at the back, see that the ground between the dams is dry.

12. Never use a key, gad, or bar, which is battered or burred on the striking end. Have the burr dressed off before using.

13. Don't throw moist "clean up" into matte ladles.

14. When helping on the slag machine, never stand near the molds where they enter the trip. Always stand behind the center of the trip.

15. Don't stand at the edge of the casting machine platform over the skip pit, or go into the pit when the skip is taking up a load of slag.

16. Whenever an arc light, or incandescent light, is broken or goes out, report it at once.

17. Motors must not be driven by any one except the motor driver or the foreman.

18. Motor drivers must not enter the concentrator tunnel with the charge train when the red-light danger signal is illuminated.

19. Motor drivers and helpers must stay with their trains until their relief comes.

20. Loud shouting is forbidden except for the purpose of preventing accidents.

Special Rules to be Observed by Employees Handling Explosives

1. Do not smoke.

2. Light no matches.

3. Handling and firing of dynamite is very dangerous both for you and others around you. Be careful.

4. Boxes of dynamite should be opened outside the powder house with wooden mallet and wedges.

Remove scale from powder cans very gently; do not tear open the can in any way.

5. Keep powder away from electric machinery.

6. Keep powder dry.

7. Don't allow dynamite, powder, electric caps or fuses to lie around loose in places where you are working.

8. Don't try to thaw dynamite. The dynamite used at this plant will not freeze.

9. Caps are very dangerous.

Do not bite them.

Do not drop them.

Do not step on them.

Do not carry them in your pocket.

Do not put them near dynamite.

10. How to make a dynamite cartridge.

Keep away from a fire, windward.

Do not smoke.

Cut off square the proper length of fuse (never shorter than 2 ft.).

Stick end of fuse into cap.

Crimp cap with crimper. (Never use a knife nor bite the cap.)

Seal joint with soap.

Unfold end of paper covering stick of dynamite.

Make hole in dynamite with round sliver of wood.

Lower cap into hole by fuse.

Fold paper around fuse and tie with a string.

11. Rules for blasting.

Tamp holes with wooden rammers only.

Ram powder down gently; do not pound.

Shake, or spring, or chamber, only one hole at a time.

Watch your shots.

Allow plenty of time for the rock to cool before putting in another charge.

In charging holes with black powder put in two cartridges, one with a fuse and one with an electric exploder.

Send warnings in all directions to get under shelter before blasting.

Do not spit more than five fuses at one time.

Count your shots. If a charge misses fire, do not touch it except under personal direction of the general foreman. A well-made cartridge never misses fire.

12. Rules for blasting machine.

Do not put a sharp bend in the wires.

Clean ends of wires before connecting together.

Twist tightly and wrap connection with tape.

Connect holes in series; one wire of the first hole is connected to one from the third hole, etc. This leaves a free wire from the first and last bore holes.

Connect these to the wires from the blasting machine immediately before blasting.

Rules for Coal Mining in General

1. Every person shall at all times obey the lawful demands or orders of his superior officer, or the person under whose charge he shall be.

2. When the boss tells you how to keep yourself safe, do as you are told.

3. No person in a place of trust shall deputize another to do his work without the sanction of his superior officers.

4. No person in a place of trust shall absent himself from work without a legitimate excuse, or having first obtained permission from his superior officer, and when possible he must send notice.

5. Every person who shall observe, or who shall come to the knowledge of any danger or dangerous condition in any mine, traveling way, air course, or defective door, brattice chuteguard, or mine appliance, which might cause a dangerous condition, shall immediately report it to the mine foreman, fire boss, or other person in charge, so that it may be repaired, but before leaving his place he shall put some plain warning across the entrance thereto to warn others against entering into danger, or tell his foreman or other official of any bad condition or dangers as soon as he knows of them.

6. Fire boss must examine all parts of the mine, especially where falls have occurred; he marks the date upon the rock, of the fall and upon the working face, also at a third point such as the side of a post.

Men must look for fire boss marks and also report the absence thereof immediately.

7. When you enter the mine, ascertain the conditions of your working place from the fire boss as soon as you arrive at the fire boss's station.

8. When you get to your working face, the first thing to do is to examine the roof of your place, and see that all loose rock, slate or coal is taken down and timbers set where needed.

9. Test the safety of the top and frequently as you mine the coal away beneath look for "slips."

10. Miner tests top by holding his hand against the top and striking with his pick at the same time—trust only to this method of feeling a movement; do not depend upon a "sound test."

11. If top cannot be reached by hand, miner uses two picks, one to feel for motion and the other with which to strike.

12. When top is found to be unsafe, post must be set at once.

13. Dangerous to go under unsupported drawslate, also to try to take it down by forcing pick handle between it and the top; if handle slips, man will fall under slate and probably be killed.

14. Thin coal under drawslate, which will not support it, but hides the slits, is exceedingly treacherous. Be watchful.

15. Set post and crosspieces to drawslate if coal is broken.

16. Set crossbars in working places that have been, or are likely soon to be, under great weight before you have mined it out.

17. Set plenty of timber in your working places.

18. Set the timber IN TIME—delay is dangerous.

19. Where you are not sure the top is safe, set "temporary" timber for safety while putting up "permanent" timber.

20. If the top is crushed or broken, do not leave it up—it is not safe.

21. It is not always safe to knock out posts, but generally safe to pull them out. When top is badly broken or has big slits in it, do not attempt to take out any props or crossbars. Rib bosses are employed to superintend this work, and see that a safe exit is provided.

22. Extra posts are set and rib boss tells miner which post to remove.

23. Timber men shall order all props, cap pieces, and all other timbers necessary, at least one day in advance of needing them. If he fails to receive such timbers, and finds the place unsafe, he shall vacate it until the necessary timbers have arrived.

24. Company's rules for timbering rooms are that posts should be in rows, 4 ft. apart and 4 ft. between posts; they should all be in line. A miner shall set no less but shall set more posts if top is not good.

25. While working on entries they must keep lagging ahead of crossbars. Crossbars should not be set close to top but room left for lagging.

26. Timber men must test entries and set posts where top is found to be unsafe.

27. Unsafe places must be fenced off, and standard danger signals set; both fence and signal are required.

28. Signs with pointing finger and the words, "This way out," show proper manways, and course for men to follow.

29. After setting timbers, bury all chips or small sticks, as they may become ignited and start a fire.

30. Do not put your hands on corners of cars to push or place them.

31. Keep YOUR track in good condition.

32. Travel on roads made for that purpose.

33. Do not pass DANGER signs.

34. Put a "drag" on rear of a car being hauled out of "dip" places.

35. Use a clevis-block in front of car wheel while loading, in all places going to the dip or to the rise, where cars would run away.

36. To make it more safe, use a tie-and-block in front of the car in all places going to the rise.

37. All forward motion of cars should be controlled by the use of sprags, and no one should place himself in front of cars or trip.

38. Do not carry tools, like augers or iron bars, on the shoulder on roads in which a trolley wire is hung.

39. Be careful of the passing trips in going back and forth to work in the mine. Should there be no manway, as is the case in a great many places, look out for the manholes in the slopes or motor roads. The manholes are white-washed.

40. Make yourself familiar with the manholes, so that you will know where every hole is. Do not use hauling roads if there is a manway.

41. Acquaint yourself with all the inlets and outlets of the mine in which you work. By doing this, you will be better able to find your way out if a fire or other accident should happen.

42. No miner or other working man shall be permitted to introduce into the mine any stranger or person, on any pretense, without the consent of the superintendent.

43. Always use the safety devices provided.

44. Ventilation at all times is one of the most important considerations. Air courses must be kept free of any impediment.

45. No miner shall enter any working room or place in which the air is taken to the face by means of a brattice, if this brattice is down or in defective condition, so as not to take the full current of air to the farthest portion of the room or place. If any brattice is found to be in defective condition, it must be reported to the mine foreman, or fire boss, at once, and the room or place shall not be worked until the brattice has been repaired.

46. All machinery, tools and other equipment should be examined at least once daily by duly authorized persons. Generally this duty is entrusted to foremen or to the highest man in charge of the work directly affected by the equipment to be examined.

47. All broken, damaged, or unsafe cars must be sent immediately to the shop,

and plainly marked "shop," so that there is no danger of their being shunted into operative use on the way.

48. Tracks, trolley wires, and switches must be kept in first-class condition at all times.

49. Coal should be properly undermined or overcut before blasting, according to the special requirements of each mine.

50. Old workings must not be entered unless due authority is given.

51. Any person who opens a door, removes a brattice or stopping, must see that it is properly closed or replaced before leaving the place.

52. Miners and all others are forbidden to knowingly or willingly deface or remove marks which may be made in any part of the workings for the guidance of the working operations. All employees are forbidden to displace, injure or damage in any way, the stoppings, props, tracks, machinery, or any other apparatus used in and about the mines.

53. Persons ascending or descending a slope, or riding in a car, will not be allowed to enter or leave the car while it is in motion.

54. The mine gives off explosive gas, and safety lamps only are used, or

In every working of a coal mine approaching any place where there is likely to be an accumulation of explosive gas, or in any working where there is imminent danger from explosive gas, no light, lamp, or fire other than a locked safety lamp shall be allowed or used.

55. Men are searched for matches and pipes upon entering and have their lamps examined, which is in compliance with the State law, offenders being arrested.

56. When safety lamps are in use they shall be used with the greatest care. Every person on receiving his lamp must examine it to see that it is securely locked; while at work he shall pay frequent attention to his lamp, and if it becomes unsafe from fire damp, oil, or gasoline spilled upon the glass or gauze, the gauze punctured, even with a small pin, the cracking of the glass, or any other cause whatever, he shall at once extinguish the light and immediately take it to the lamp house or relighting station and report it to the foreman or fire boss, or

Be careful not to injure your safety lamp.

57. A safety lamp shall at all times be kept in a perpendicular position and must be kept hung up when practicable, and at a safe distance from the swing of the pick, hammer, or other tools, or

Safety lamps must be locked, inspected and kept in upright position, and no matches shall be allowed in workings where lamps are used.

58. No person shall improperly use or damage any safety lamp, or attempt to blow out flame in any lamp except for purpose of testing a lamp before entering the mine; and no person, unless duly authorized, shall unlock or attempt to unlock or open any safety lamp, or have in his possession any key or contrivance for opening any safety lamp.

59. Employees will be charged with all breakage of safety lamps and for the loss of a lamp.

60. Do not drill holes to within 6 in. of the back of the mining. Do not drill out missed shots.

61. You should go to the powder house in the morning for your powder, and under no circumstances take out more powder than is required for one day's work.

62. Provide yourself with a safe container for carrying powder. Powder should not be carried in your pockets. A fiber powder case can be secured from the company's office. Such a powder case is safe, and has been used with great satisfaction to miners.

63. Do not mix giant and other powders in the same cartridge; it is dangerous.

64. Leave dummies enough in readiness to charge the hole, or holes, to the mouth.

65. Be very careful when filling the dummies that no coal is mixed in with the adobe clay or slate tamping.

66. Under no conditions should miners be allowed to use coal dust, or other combustible materials, for tamping.

67. When drilling rib shots, take care not to grip the holes too tight on the rib.

68. Do not get the holes too high, as they will get into the roof and make roof conditions bad.

69. The shot lighter shall see that all passages leading to the place where the shot is to be fired are properly guarded before lighting the shot; and when firing a shot in a place approaching a workings into which the shot might possibly hole through, it shall be his duty to give warning to the men in the workings.

70. Before firing a shot in a dusty place the floor, sides, face and roof must be thoroughly dampened for a distance of 20 ft. from the shot.

71. Do not fire shots with less than 100 ft. of "lead" wires.

72. Do not fire shots with damaged "lead" wires or bad connections.

73. Do not carry "electric caps," except in a locked case.

74. Do not fire a shot where there is any coal dust within 100 ft. of the shothole, or where the dust has not been thoroughly wetted for a distance of at least 100 ft. from the shothole.

75. Do not put more than 1½ lb. of explosive in any shothole.

76. Do not fire shots where there is not enough shelter from the blast.

77. Misfires must not be withdrawn, or the holes reopened, nor should the miner return to the place for 8 or more hr. if fuse is used, and not until current is disconnected and 5 or 10 min. have elapsed, if battery detonation is employed, or

Do not go back to a misfire shot before the lead wires are disconnected and 5 min. have passed.

78. If you have charged a shot and cannot fire it, fence off the place and put up a danger signal before leaving it.

79. Any person firing a shot must observe the above rules.

80. When charging his hole for a blast or shot, if the cartridge sticks he must remove it carefully and reduce its size, or enlarge the hole so it will enter easily. He must not ram or force the cartridge with a drill. When black powder is used, he must use a tamping rod tipped with copper. For other powder he must use a wooden tamping rod. Care must be taken to tamp the cartridge with sufficient depth of material. Slate or clay should be used in all cases. Coal dust must not be used. If his place be dry and gives off inflammable gas, all fine dust must be removed from the immediate vicinity of the shot, or the working face, roof and floor thoroughly dampened for 20 ft.

81. After each blast he shall exercise care in examining the roof and coal, and shall secure them safely before beginning work.

Tools

82. The use of iron needles and iron tamping bars not tipped with 5 in. of copper is hereby declared unlawful. Where dynamite is used a wooden tamping bar only must be used.

Car

83. No person shall ride upon a loaded cage, or car used for hoisting purposes, in any shaft or slope, except rope rider or couplers; nor shall any coal be hoisted out of any mine while persons are descending into such mine, notice of which shall be kept posted at said mines.

Care and Handling of Cars

84. Broken, damaged, or unsafe mine cars must be marked "Shop" by employees handling cars. It is part of the duties of top and bottom cagers, brakemen, switchers, engineers, motormen, loaders, drivers, and tipple men to so mark any defective cars coming under their observation and shunt the shortest way to the shop.

85. The brakemen on the bunker run will see that all cars marked "shop," or that are known to him to be defective, are switched on the shop repair track and not taken out until repaired.

Shot Lighter

86. The shot lighter shall be subject to the mine foreman, and fire all properly placed shots after first examining the working place as to the state of inflammable gas, or coal dust, and if any be found the shots shall not be fired until such danger is removed.

Switchmen

87. It is the duty of the switchmen to attend to switches and see that they are properly set, couple and uncouple the trip, brake or sprag the cars, keep a lookout ahead for danger, and signal the motorman or engineer.

88. Switchmen must not allow any person to ride on cars without permission of the manager or foreman.

Cagers

89. It shall be the duty of the top and bottom cager to take charge of the slope during working hours and to report promptly to the mine foreman any defect in track, rollers or signal wires; shop all defective cars, and in case of a wreck on the slope to send word to the mine foreman.

90. The top cager will make an inspection of the slope after any wreck, or at any time he has reason to believe there is anything wrong in the slope.

91. The cager will not allow anyone to ride on a loaded car, or when coal is being hoisted on the opposite side.

92. The cager will not allow anyone to get off or on a car while in motion, or after the signal has been given the engineer to hoist or lower.

Drivers

93. Be careful in getting on and off cars when they are in motion. Avoid getting off moving cars, if possible, near a prop or close rib.

94. Do not get between cars to uncouple them or release brakes. Keep your legs from between bumpers.

95. Do not ride between cars unless your work cannot be done any other way.

96. Be very careful in spragging cars, as drivers have been known to lose fingers by doing this carelessly.

97. In riding on front end of empty or loaded cars, keep your head low enough so that it will not be caught between the car and cross-timbers. Lives have been lost by not keeping the head low enough.

98. Do not make "flying switches."

99. Do not ride in front of loaded trip unless your work cannot be done any other way.

100. Keep your trips under control.

101. Promptly notify the mine officials of:

(*a*) Bad track.

(*b*) Slate, dirt, or posts piled along the side so that you cannot pass the trip in safety or keep it under control.

(*c*) Bad roof, when known to you.

102. Do not excite your horse or mule by rough or bad treatment.

103. The driver must properly care for mule while in his charge, feed and water it at dinner hour, and must not abuse it at any time, nor allow anyone else to do so.

104. The driver must report the condition of the mule and harness to the mine foreman in case of injury, sickness, loss of shoe, or defective harness.

105. The driver must shop all defective cars that come to his attention and report all defective track, chutes, and dangerous conditions on the entry or gangway.

Motormen and Locomotive Engineers

106. They shall at all times keep a lookout ahead and act promptly on seeing or receiving any signals.

107. They will be held responsible for the care and cleanliness of their machines and shall report any defects at once to their foreman.

108. Motormen must follow instructions received from the electrician in regard to the care of their machines.

Trackmen

109. Examine the roof carefully before commencing to lay switch or repair track, as you may be working at that place for several hours.

110. Do not carry spike or crowbars, picks or shovels on your shoulders when traveling in entries where trolley or naked wires are; you might get a shock.

111. Always be careful in repairing track near trolley or naked wires. Men have lost their lives by being careless in such places.

112. Always report on unsafe conditions that you may find in the mine to mine foreman; or, if possible, remedy the defects yourself.

113. Report to the foreman all defective track, switches, trolley wire, or appliances, and dangerous places.

114. Do not allow any person, except your switchman, mine foreman, trackman and inspector, to ride on the motor or locomotive, or on the cars, without the permission of the manager or foreman.

Trappers

115. Do not keep your door open any longer than to allow trips, mules or men to pass through it.

116. Your door was built to conduct the air to other parts of the mine, and should not be kept open longer than is absolutely necessary.

117. Do not get in the dark if you can avoid it, and do not go to sleep behind your door. Trappers have been seriously injured by this carelessness.

118. Do not change the cotton in your lamp while in the mine, not throw the unburned wick away. This has caused fires and serious loss of life in the past.

Pumpmen

119. It shall be their duty to keep the pump in their charge in good condition for work. All ordinary repairs to pumps and pipe must be made by the pumpman.

120. Serious damage or breakage of pumps, or failure to keep the water within a safe limit, must be reported immediately to the master mechanic and mine foreman.

121. Reports of insufficient supply of steam or power must be made daily until remedied.

122. Tracklayers, gangway timbermen, pillarmen, roperiders, bottom cager, top cager, pumpmen, are subject to the mine foreman and must execute their work as ordered from time to time by the foreman.

123. The above employees before absenting themselves from work must properly arrange with the mine foreman.

Master Mechanic

124. The master mechanic will have full and complete charge of all machinery in and around the mine, and must see that it is properly used, kept in good repair, and clean.

125. The master mechanic must inspect, or cause to be inspected all ropes and machinery used for hoisting or lowering employees out of or into the mines in each 24 hr.

126. The master mechanic will be chief of the mine fire department and will see that fire hydrants are placed and kept in repair, and a supply of hose is on hand for the protection of all mine buildings and mine slopes and pump rooms.

127. The master mechanic must keep the ventilating fan running as directed in writing. Such notice in writing must be posted in the fan house. In case of accident that will necessitate shutting the fan down, the mine foreman shall be immediately notified and the fan must, if possible, be kept turning over until word is received from the mine foreman to shut down.

128. The master mechanic will examine, or cause to be examined, the fan at the beginning of each shift and see that it is kept in perfect order. If it becomes necessary to repair the fan he must give timely notice to the mine foreman.

129. In case of a mine fire or explosion, he must report immediately at the fan and keep it running as usual and not allow anyone to stop or reverse it, or make any changes himself, without written instructions or direct telephone communication from the mine foreman or manager.

130. All men connected with the running and care of all engines, boilers, and other machinery, and employees in the machine and blacksmith shop, are subject to his supervision and orders.

Hoisting Engineers

131. Hoisting engineers must at all times pay strict attention to signals, and to the trip. When hoisting or holding a trip on the slope, under no circumstances shall an engineer leave his reverse and brake levers; and particular care must be exercised in hoisting and lowering man trips, which must never be run at a speed exceeding 600 ft. per minute.

132. Any defect in the engine or rope must be immediately reported to the master mechanic.

133. The engineer will consult with the mine foreman as to the speed trips are to be run, and report any defective track to the foreman.

134. A copy of the "Code Signals" must be hung up conspicuously in the engine house.

135. It shall be the duty of the engineer, subject to the control of the master mechanic, to see that the firemen perform their work properly, and that steam in sufficient quantity and pressure is furnished to do the work.

136. The engine house and engine must be kept neat and clean.

137. Firemen must at all times keep their boilers properly supplied with water and immediately report any accident to pump, injectors, or pipes furnishing same, and

any deficiency in the water supply. Should the water in the boiler get below the try cocks or gage, the fires must be pulled immediately and reported.

General Rules and Suggestions for Foremen and Superintendents as per Jones Interest in West Virginia Coal Mines

Coal Mines

1. All sights to be in center of entries and entries kept on sights.
2. Roadsmen laying track must fishplate every joint; and bond everywhere electric haulage is used or is expected to be used.
3. When a parting is laid, a bond of sufficient length to close the break caused by the switch must be put in.
4. When a parting is removed and a filler rail put in, it must be splice barred and bonded if electric haulage is used or expected to be used and the bond which was used when the parting was in, must be removed to a safe place for use again.
5. When roadsmen have finished a job on which they are working, they must pile up all extra ties and iron, gather up all spikes, splice bars and bolts, and put into kegs ready for removal to next job.
6. Insist on miners loading cars to their capacity.
7. Insist on miners loading all slack and machine dust.
8. Insist on miners keeping all slate, binders and other impurities out of the coal.
9. Insist on miners keeping road close to rib side of room, except in mines operating with road in the middle.
10. Insist on miners keeping themselves safe.
11. Insist on miners working full time when mine is in operation.
12. Allow no miners to load any unnecessary slate cars.
13. Insist on machine men cutting entries before rooms and cut close to bottom.
14. Insist on machine men removing exposed bits before allowing mule to haul the machine.
15. Insist on machine men and loaders having dull bits sent out, placing them on brake side of coal which check hangs.
16. Insist on the machine man keeping his section cut up or put somebody else in his place. This is for the purpose of keeping up the regular tonnage.
17. Any employee wishing to be absent from duty must first apply to and receive permission from his foreman.
18. Any workman offering money, liquor, or valuables of any kind to a foreman, boss, or clerk will be discharged; and any foreman, boss, or clerk accepting money, liquor, or valuables of any kind from workmen will be discharged.
19. Entries must absolutely be not over 10 ft. wide.
20. Entries must be kept on sights.
21. Slate must be trimmed off sides and loaded before yardage is taken up.
22. Entrymen must not fill in road with bug dust.
23. Entries must get all the cars they can load; must be cut as soon as cleaned up.
24. One cut or more can be taken out of room necks as the entries advance, providing it does not delay the entry.
25. Air must be kept to the faces.
26. Rooms shall not be driven over their prescribed distance unless to recover some lost coal.
27. Great care must be taken in posting rooms, and if in danger of falling they should be reposted or road removed.
28. Allow no slate to be piled on rib or blind side of rooms.
29. Try to induce the miners to block off the coal in front before shooting and take less powder.

30. No room should be allowed to get behind as it prevents uniform shooting of ribs and receives all the weight from other rooms.

31. Posts should be set in break-throughs to support rooms.

32. No loaded or empty cars should be permitted to stand idle at any point.

33. All rooms to be measured up before rib drawing is begun.

34. All break-throughs to be finished before entries are driven more than three cuts ahead.

35. Have short slate posts in entries.

36. Room sights shall be carried in the middle of track and shall be sighted each cut.

37. Motormen and drivers upon receiving empty cars should always examine brakes to make sure they are in good order, and if there is any defect or the wagon is not safe to load, mark the wagon and return it empty to the outside.

38. Safety or derailing switches must be installed in such parts of the mine as in the opinion of the mine foreman may be necessary to prevent runaway cars from injuring employees.

39. Puncher runners cutting in sections shall have two safety lamps and keep them out of the dust as much as possible when cutting.

40. All safety lamps to be examined by a person designated by the mine foreman, other than the one cleaning and filling them.

41. Where water lines are used they shall be kept within 60 ft. of the face and not closer than 20 ft., and in extending them the valve shall always be carried on end of line next to the face.

42. It shall be the duty of the mine foreman to see that haulage ropes, sheaves, cages, etc., are examined according to law.

43. In all shaft mines when cage has not been running for some time, or after supplies have been lowered, an empty cage shall be run before men shall be allowed to ascend or descend.

44. In shaft mines men shall not be hauled on a cage when anything else is on the other cage.

45. In shooting entries about to hole through, no shot shall be fired until the shooter himself warns the men on the other side.

46. Danger boards shall comply with State mine law.

47. No place shall be considered as fenced off unless a danger board is fastened to fence in such manner that a man cannot cross without seeing the board. Laying a board on the bottom does not comply with this rule.

48. All miners must use the kind of powder prescribed by mine foreman for this mine.

49. Loaders must not drill holes for the purpose of blasting further in any direction than the undercutting or mining.

50. Motormen must not run with trolley pole ahead except where it is impractical to turn it. Where it is absolutely necessary to run with pole ahead, trip must be kept under control so no injury can occur to motorman through breaking of pole in case it should fly from wire.

51. Motormen must report at end of each shift exact condition of their motors, an account of work performed during shift, and all delays interfering with the progress of his work.

52. Motormen must keep sufficient supply of sand in sand boxes of motors for a round trip.

53. Motormen must examine their motors frequently and see that all connections, contacts, and gears are in perfect working order.

54. Motormen must thoroughly understand the mechanism of their motors and keep them in good running order.

55. Motormen must oil their motors regularly.

56. When anything is out of order that motormen cannot readily fix, motor must be shopped.

57. Motormen must not overload their motors.

58. Motormen must not throw the reverse lever to change the direction of the current when motor is moving in the opposite direction, except in case of emergency.

59. No shooting of any kind must be done unless the explosive is placed in a hole properly drilled and tamped, except under the immediate supervision of the mine foreman.

Discussion

Wallace McKeehan, Douglas, Ariz. (communication to the Secretary*).—In going over this report, which I have done very carefully, I find that the summary as composed deals with the various problems almost entirely from the employer's standpoint and that practically no consideration is given to the employee's point of view.

An experience of 3½ years as safety inspector, and almost 30 years as workman and official, has convinced me that any plan for the betterment of working conditions which will achieve any degree of success must give the workmen their share of responsibility and allow them full participation. The company may finance and direct the movement, but its ultimate success depends upon the support given by the workmen. The coöperation of the men doing the actual work is so essential that all writers insist that this must be secured, but they do not agree as to how this can best be done.

I am convinced that the employer who, in starting an employees' welfare department, puts the proposition up to his men as being strictly a good business deal for both, will secure results. While he who prates of humanitarian motives, of doing all this for the good of the workmen and their families, at the same time using a paternal tone, will only cause the men to become suspicious. American workmen want no petting. All they ask is a fair deal all around.

Accident prevention and welfare work in general cannot be forced upon English-speaking employees, and the management that tries it will lose out; but a straight-forward, look-you-in-the-eye proposition, with all the cards on the table, will appeal to the men. Once they are convinced that they have an even break, they will sit in the game and raise your bid every time.

Too many welfare organizations are placed almost entirely in the hands of officials and bosses; especially are they in the majority upon the committees. This is not as it should be. In my opinion the workmen should constitute the majority upon all committees. If this is done, rarely will they abuse the privilege. Our experience would go to prove

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* Received Feb. 1, 1917.

that workmen take more interest in accident prevention matters than the officials or bosses, and are quite quick to censure their fellows for infractions of rules. In proof of this I would state that of the many safety suggestion cards received by this department, only one was sent by a boss.

This department[1] receives many inquiries as to the best method of initiating and conducting a safety and welfare organization. The reply invariably sent is, "Do your part first; make all working conditions as safe, sanitary, and convenient as it is possible for you to do. After you have shown the men that your intentions are of the best, you are in a position where you can ask for their coöperation."

The work that can be done by companies engaged in mining, milling and metallurgical operations is, comparatively speaking, slight as regards the number of accidental injuries prevented, but the work done creates a good impression among the men.

While no fixed method can be advanced for the formation and conduct of an employees' welfare organization, if the coöperation of the workmen has been secured, the other problems may be quite easily worked out. All employees are naturally suspicious of the good intentions of their employers. One of the first things that the manager must do is to issue an order to all those in charge of operations, that safety must be the first consideration in the conduct of all work, and insist that this order be carried out to the letter. Most managers dislike to do this, because they fear it will interfere with the output. Such is not the case, however, as safe working conditions make for greater efficiency. All officials and sub-bosses must be made to understand that unsafe conditions, methods, and practices will not be permitted. A good way to secure their coöperation is the paying of a bonus for a reduction in accidents; also a bonus for keeping their respective places clean, orderly and sanitary; this last to be cared for by a monthly inspection, preferably by a committee of employees. The Copper Queen Co. pays its mine bosses a bonus of $30 each. This is paid quarterly and is based on the least number of accidents per thousand shifts worked, ten bosses participating.

One thing that has caused the workmen to view with suspicion the formation of a welfare organization is that some companies have started the work with a great hurrah and then have let the movement quietly die. Although this practice has about died out, it will prove a severe handicap to future operations along welfare lines by these particular companies, to say nothing of its effect upon others.

The present tendency among safety inspectors is to have as few and as simple rules as possible. No rule should be allowed in the book that cannot be enforced. All rules should be couched in the terms and

[1] Mr. McKeehan is Safety Inspector with the Copper Queen Consolidated Mining Co.

phraseology used by the workmen. For the above reasons, the rules committee should include a number of good practical workmen. There are so many good rule books now in use that the rules committee can select sound practical rules that will cover the operations of practically all departments. These can easily be changed or amended to suit local conditions.

Strict observance of all rules must be insisted upon. To secure this, some form of discipline will be necessary. Just how this matter will be handled should be left to the workmen's safety committees. With our organization, a man who breaks a safety rule is reprimanded for the first offense; given a layoff of from 1 to 15 days for the second, and discharged for the third.

Both men and bosses must have a thorough knowledge of all rules. This is often a difficult thing to obtain, for with a large employing company a personal examination cannot be made of each individual. One way in which the rules can be given wide publicity is to have all the rules covering a certain class of work printed on a card and to have this card placed at or nearby where this operation is being carried out. Rules for carmen and loaders should be tacked up at chutes; for motormen and swampers, at oil houses, sidings, skip pockets, and where these men stop for lunch. Miners on stope work should find the card nailed on timbers. Rules covering machine work, the handling of explosives, in fact all important rules, may be placed at points where the men congregate for lunch. This will be found to be a better plan than to depend entirely upon rule books. The books, however, should always be issued and the employee's receipt taken.

A classified accident record of all injuries, even the slightest, must be kept. This should be so arranged as to cover all operations in every department. This will have greater value if the report sheet used by the bosses is so arranged as to call for full details as to the occurrence of an accident. Information regarding slight, minor, and no-loss-of-time accidents rarely reaches the inspector, except from the boss's reports. Injuries causing no loss of time should always be reported. These constitute about 40 per cent. of all injuries, and are the ones most likely to become infected, the men believing that they are not of sufficient importance for medical attention. With our company, all hurts, no matter how slight, must be reported. Failure to report will cause the employee to be discharged.

All reports of accidents should be absolutely impartial. This is very necessary if an inspection and report is made by the inspector or safety committeemen. Particular pains must be taken to fix the responsibility and the blame placed where it belongs. In no circumstances shall the fact that the company, or one of its officials, was at fault be ignored. If the injured, or any of his fellows, was to blame, he should be censured.

In case of a fatal or serious accident, photographs should be taken of the place immediately, if possible posing a fellow-workman to show how the accident occurred. These photographs are of considerable value when used in connection with a bulletin describing the accident, as they serve to fix the occurrence and nature of the accident more firmly in the minds of the workmen.

Live, snappy, up-to-date bulletins, if simply worded and using the familiar trade terms and phrases, illustrated by photographs taken on the ground, will have considerable value in calling the workmen's attention to safety matters. What is more, the men will read and discuss them with their fellows. Bulletins to be effective should be written by one who is acquainted with local conditions. Photographs of local scenes are the only ones that have any value. If workmen who are popular with their fellows are shown in the pictures, it will cause more interest to be taken in the bulletins. The common bulletins sent out by safety organizations to all parts of the country, printed on poor paper and with cheap illustrations, if any, have just about as much effect upon the men as would an advertisement calling for recruits to the National Guard.

Writers upon safety matters are laying considerable stress on the necessity for educating the workmen to protect themselves. This is all very good as far as it goes, but it doesn't go far enough. Bosses and officials need training along these lines as well as the men and should be the first to receive it. Bosses and officials can be given this instruction at weekly and monthly meetings, where lantern slides showing local conditions, methods, practices, etc., may be thrown on the screen, illustrating the talk made by the inspector or some one in authority.

New workmen can best secure this instruction when first they go to work. They should always be placed with an old experienced employee whose business it is to see that they are fully informed as to the dangers of their new work and as to how they should proceed to protect themselves. This may cost a little more at the start but will be found to provide a more efficient workman and will be cheaper in the end than compensation payments. The tendency among up-to-date safety organizations is to secure better training for the new recruit and this method will become universal just as soon as its value is thoroughly understood.

The education of the older employees is a far more complex problem than most people suppose. The workman, quite naturally, thinks he knows enough to care for himself, and being from Missouri, has to be shown. The one who is to do the showing must have had practical experience in the work and be one in whom the men have confidence. They also put a great deal of reliance in the recommendations of their own safety committee.

When it is possible to do so, all instruction should be given to the men when they are at work. They do not like having their leisure time

taken for this purpose and will resent it if this is done frequently. If it is necessary to instruct them when they are off duty, it can best be done by giving them some form of entertainment, to which they can bring their families, and using a small portion of the time for an illustrated talk on accident prevention. A well-conducted bulletin board, with safety mottoes, rules, etc., posted near the working places, will do effective work in training the men to watch for unsafe conditions and in protecting themselves.

One thing that should be given more attention than it ordinarily receives, is that of selecting men for a particular kind of work; in other words, fitting the job to the man, not the man to the job. It is the common practice to put a man on a job; if he doesn't make good, discharge him. Yet this same man, if given a chance at something else, perhaps tried at several different kinds of work, will more than likely make good. If a man can be placed at work that he likes, he will be a more efficient, safer, and better workman.

In this connection I would call attention to the indiscriminate discharge of workmen by foremen and bosses. I do not believe that any man in direct charge of workmen should have the right to discharge them, and I say this after many years spent in handling men. The practice is a relic of the dark ages and should be abolished. The trouble with the present system is that the workman holds his place subject to the whims and fancies of his boss, and if discharged, has no recourse. In these days the education, training and shaping of the untrained man into an efficient, safe and reliable workman is done at a considerable cost, and the employer should not have this investment jeopardized by the sole judgment of any one man. The man in charge should have the right to lay off a man, at the same time filing a complaint against him, but his final discharge should only take place after the workman's side of the matter has been heard. The discharge of a workman should be handled through the employment office, and after the case has been passed upon by a committee.

The large corporations, realizing the value of the trained worker, are taking steps to provide a better system of disciplining the men, also to see that the men get a hearing before being discharged. It is certainly disheartening to the leaders in a safety organization to find that men who have been trained in accident prevention, mine rescue, first aid, etc., are being discharged and forced to seek work in other localities.

Accident prevention is a good business for the employee, employer and the community, and if carried to its ultimate success, will pay substantial dividends to all, but the greater returns will be received by the employee, and deservedly.

If the safety campaign is extended to cover the betterment of conditions under which the employees work and live, it will create a better

feeling between both parties and will do a great deal to eliminate some of the problems that are confronting capital and labor today.

E. Maltby Shipp.—It was the intention of the Committee on Safety and Sanitation to collect information and useful data from the companies having well-organized safety departments and print this in pamphlet form for distribution among the small companies that have no way of employing safety engineers nor have at their disposal these data. Consequently, I took these rules and the general data, especially the rules, went over them a great many times and tried to cut out whatever duplications appeared.

Mr. Eavenson, the Chairman of the Committee, obtained a great deal of information from the coal companies. Mr. Charles W. Goodale sent in a great many booklets and rules and all issues of the *Anode* and other publications published in the Butte district, and this report is a boiled-down skeleton of all of the information at hand. Mr. Sidney Jennings called attention to a duplication as follows: "The principal duplication here might be stated to be between safety signs, and safety rules."

It was thought at the last meeting we had that a great many people might refer to the pages on safety signs to get an idea of the one they wanted to have printed. If they were making up a list of signs they would not be likely to turn to the rules, as they would think the signs had been covered, and *vice versa* in reference to the rules. If they were getting up a set of rules they would not turn to the signs to get out the data from there for rules, consequently in a great many cases the signs and the rules are practically the same for the same conditions and in making rules for general purposes I think that there is a possibility of cutting down, and not putting in the general rules anything that is covered in the special rules.

But again, it is hard to do that, and was not the object of this paper to do that, because we wanted to get the data before a large number of the committee, and also before the members of the Institute who had been doing this work in the field and might criticise from their practical experience.

Mr. Eavenson makes the following suggestions in his letter of comment:

"In the bottom paragraph on page 253, you instance the lack of knowledge of the actual cost of the safety work. So far as I know, the only actual costs that have been published are those of the United States Steel Corporation. These figures were exhibited at the Panama-Pacific Exposition in San Francisco, and have also been published in one of their industrial bulletins, and I can see no reason at all why they should not be repeated in your report as they are the actual figures and show a large

money saving aside from every other consideration resulting from their expenditures for safety work.

"In addition to this, the Associated Companies, who are now insuring some of the mining companies in States having certain kinds of compensation laws, are experiencing this same saving, and in a paper read about a month ago before the Coal Mining Institute of Kentucky, some instances of saving were given, and I believe it would be advisable to incorporate these with your report.

"The advantages listed in the first paragraph of page 254, could be strengthened a little by the morale produced among the employees as the result of safety work, as there is no question that the thinking men soon realize that their employer has their best interests at heart and appreciate the interest that is being shown in this way. We have experienced this in a number of cases in our own work here.

"In the list of Fundamental Principles on the bottom of page 254, I think it would be better to emphasize as fully as possible that an educational campaign is absolutely essential. In fact, I believe that the experience of everyone doing work of this kind is that three-quarters of the problem is an educational one and that the question of safeguards of various kinds, while important, is a comparatively small part of a safety campaign. This same point is emphasized further in your discussion of the human element on page 258, as there is no question that the ordinary man, as a general thing, does not realize many of the dangers daily encountered.

"Another important point which is not emphasized as thoroughly as it might be, under the head of The Organization, is the practice of having committees of workmen appointed at regular intervals to make an inspection of the mine or plant and make any recommendations necessary for safety measures. This point was brought out in my paper presented at the February, 1915, meeting and both ourselves and the H. C. Frick Coke Co. have found that the appointment of such committees results in considerable good.

"In Rule 37, page 280, the words 'or brakes' should be used after 'sprags' as in most of the up-to-date coal mines sprags are no longer used.

"Relative to the use of explosives, it would be advisable to insert that none but permissible explosives should be used in mines generating gas or in those using safety lamps.

"On page 282, Rules 70 and 74 conflict. In Rule 80, on the same page, it should be specified that the gage of augers should be examined frequently in order that the hole may be bored amply large for the cartridge. Rule 88, on page 283, is not in accordance with the best practice, as this is to prohibit everyone riding on trips excepting on the regular man trips, used for taking men into the mines.

"It would have been advisable, in my opinion, to have omitted the General Rules and Suggestions for Foremen and Superintendents, as per Jones' Interest in West Virginia Coal Mines, as most of this matter is included in the rules already written."

Now those Fundamental Principles that he spoke of were just a general review placed in the front of the paper and the educational feature is better emphasized on pages 255 and 256 but could be even more featured.

B. F. Tillson, Franklin Furnace, N. J.—In discussing this report of the Committee on Safety and Sanitation, I am glad to express my high appreciation of the great amount of work that has been done in collecting so much information from various sources. I believe that the Secretary is to be congratulated for his success in accomplishing this work.

I think it might be of value in addition to this report to state conditions that have since been changed, as apply to the method used by the New Jersey Zinc Co. for encouragement of interest in a safety campaign. On page 262 of this report the first paragraph mentions a bonus system which had been in vogue and which helped to greatly reduce the accident rating. That system has been still further elaborated as follows:

During the past year all shift bosses in the mine who have not had any accidents which caused a disability of a workman received a bonus of $5 monthly; semiannually, all those shift bosses whose record for the 6 months showed that all the number of disabilities (a disability being an accident which required the loss of 1 or more days of time) would permit a rating under one and one-quarter disability per 10,000 hr. of labor worked, would receive a bonus of $20.

Furthermore, at the end of the year, an added bonus was paid on a basis of the amount of time lost owing to accidents to men in a shift-boss's gang, and the rating used there was a matter of 0.4 per cent. of the amount of time which was worked by the laborer in each shift-boss's gang during the year. In other words, 4 days of time lost in a shift-boss's gang rated per 1,000 shifts of labor worked was a bonus record and all shift bosses who had an average less than this received an annual prize of $20.

This system has worked out with considerable advantage to us. In May, 1913, we started in this bonus system which we have changed slightly from time to time in order to equalize the distribution of prizes so as to get the maximum from the bosses. The average for the first 4 months of the year was a rating of two and a quarter shifts of disabilities per 10,000 hr. of labor worked. This rating has been gradually dropping from year to year until we find that our average for this past year was three-quarters of a disability per 10,000 hr. of labor worked. It proves that a stimulation of the interest of the shift bosses underground in the prevention of accidents in order to fatten their payroll is probably the

most efficacious means you can employ for gaining their sincere coöperation with all the workmen and their help in the latter's education. In our case we have found it most satisfactory.

H. N. Eavenson, Gary, W. Va. (communication to the Secretary*).—The United States Coal and Coke Co. at Gary, W. Va., has, as a safety measure, installed electric lights in all of the working places in its mines. The voltage carried at the substations ranges from 275 to 290 volts, and at first carbon-filament lamps of this voltage were used. These gave good satisfaction, but in a short time after their general adoption, their manufacture was discontinued, and Mazda lamps were then installed. On account of the extreme delicacy of the filament, the life of these high-voltage lamps was very short, and the use of 100-volt lamps, three in series, was begun.

Along the headings lights were installed at intervals of about 100 ft., a lamp always being placed at each switch.

On headings these lamps are connected to the trolley wire, to each other, and the return, by No. 14 wire. In the rooms and other working places, No. 14 duplex rubber-covered wire is used, a length of wire sufficient for the projected length of the room being wound upon a reel which is hung upon a post near the face. At the mouth of the room, each end of this wire is connected to a pull-out plug, which is connected in turn to the trolley wire and the return.

The lamps are connected in series by weather-proof sockets to a special cable ahead of the reel and are carried forward and suspended on posts as the face advances, the reel being moved correspondingly; 40-watt lamps are used, the two outer ones being shaded by 30° metal reflectors, the center one being bare. Experience has shown that this amount of illumination is sufficient for a place as wide as 40 ft. Wires are carried on insulators on an inside row of posts. In all, over 10,000 lamps have been installed.

As usual, considerable objection was made to the installation of these lights, but after more than a year's trial, no one would willingly revert to the old conditions. The increased amount of light available increases the efficiency of the workman, as well as his safety.

C. W. Goodale, Butte, Mont. (member of the committee)—(communication to the Secretary†).—In the discussion of papers presented at the New York meeting of the Institute in February, 1915, and at the Arizona meeting of September, 1916, I referred to the activities of the Bureau of Safety of the Anaconda Copper Mining Co., and I am now submitting a few thoughts by members of the staff of that Bureau.

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* Received Mar. 7, 1917.
† Received Apr. 13, 1917.

Torgus H. Oaas, Safety Engineer, Butte, Mont.—The report of the Committee on Safety and Sanitation has described different organizations and methods of promoting safety work. The effectiveness of these methods would aid the beginner in his campaign for safety in choosing the kind of organization and methods best adapted for his field. This can be obtained only by a discussion by those who have long promoted this kind of work, and have met with successful results. As the field is new, the different organizations promoting safety work have not established a common basis of rating for their fatal, serious and slight accidents, so that it is now difficult to get a comparative basis from which to judge the merits of the different systems.

C. W. Goodale.—The U. S. Bureau of Mines adopted, at the beginning, as a basis for its yearly summaries, the number of accidents per 1,000 men employed, and it is now (quoting from its report for 1915, *Technical Paper* 168) making the following "classification of injuries:"

"Many of the States now have compensation laws, and in order to conform with their classifications, the bureau's classification of serious and slight injuries for 1915 is on a 14-day (2-week) basis instead of 20 days, as in previous years. The new classification of injuries includes three types, as follows:

1. Fatal
2. Serious (time lost, more than 14 days)
   (*a*) Permanent disability
      Total
      Partial
   (*b*) Others
3. Slight (time lost, 1 to 14 days, inclusive)

"*Permanent Total Disability*—Loss of both legs or arms, one leg and one arm, total loss of eyesight, paralysis or other condition permanently incapacitating workman from doing any work of a gainful occupation.

"*Permanent Partial Disability.*—Loss of one foot, leg, hand, eye, one or more fingers, one or more toes, any dislocation where ligaments are severed, or any other injury known in surgery to be permanent partial disability."

For our own purpose, that is, to enable our superintendents and foremen to follow the progress of our safety campaign, and the part which each one takes in the movement, we have adopted the shift or day's work as a basis, considering each shift as a risk, but in order to avoid too many decimals, we show the number of accidents per 10,000 shifts. The difficulty in using as a basis the number of accidents per 1,000 men employed will be seen when attention is called to the fact that on Sundays and holidays work is not entirely suspended, as in the case of a factory, with only watchmen on duty. These days are taken for repairs in shafts and on equipment, and this work requires many men. To arrive at the exact average number of men employed for a year or for any period would therefore call for a good deal of arithmetic.

Referring to the classification of injuries, the Bureau of Safety of the Anaconda Copper Mining Co. has insisted that all accidents, however

slight, shall be reported, as it may happen that a trivial injury will develop infection, and it wants a complete history of every accident.

During the year 1916, the accident record covering underground and surface operations was as follows:

Kind of Accident	No. of Accidents	Per Cent.	Accidents per 10,000 Shifts
Fatal[1]	46	0.76	0.12
Serious, time lost more than 14 days	301	4.97	0.79
Slight, time lost 1 to 14 days	2,913	48.09	7.64
Trivial, no time lost	2,797	46.18	7.34
	6,057	100.00	15.89

[1] 21 of the 46 fatalities were due to a fire in the Pennsylvania mine.

T. H. Oaas.—Our experience in the mining department of the A. C. M. Co.'s Bureau of Safety has proved that the strictest disciplinary measures must be enforced if accidents are to be reduced. The payment of a cash bonus to the mine foreman having the least number of fatal and serious accidents, in proportion to the shifts worked, has been tried with some measure of success. A set of rules covering all dangerous practices has been established with an attached penalty which must be inflicted impartially on all offenders. This not only affects the miner, but also the shift boss and foremen who neglect to instruct their men in the necessary precautions before they undertake any work.

You may be sure whatever respect or interest the foreman and shift bosses have for the safety of their workmen is reflected in the manner in which they do their work. The shift boss, or foreman, who says to his men "How's she going?" and does not look for dangerous slabs, poor floors, unguarded chutes, misfires, etc., in the working places, is not the leader of that crew of men, but only an extra timekeeper. That type of shift boss or foreman is the one who is a detriment to the safety work of his company. These men in charge must caution their men about that dangerous slab, or that unguarded chute, and they must see that these unsafe conditions are remedied at once. Should they pass over these conditions, their men will do likewise. Some shift bosses have compelled their men to work under loose roofs and sides, fearing the loss of a car or more of ore for the time taken to secure the loose roof and sides. This is a very common and mistaken practice, for safe methods have proved that they are necessary to efficiency. The A. C. M. Co. has handled this problem by discharging shift bosses and foremen who fail to look after the safety of their men, and obey the safety rules of the company. By insisting on safe practices and appliances, and by strict disciplinary measures backed up by the operating department it has convinced the majority of the men of its earnestness in safety work.

During the year 1916 there was a reduction of 35.9 per cent. in the fatal and serious accidents happening at the mines of the company employing about 11,000 men, by comparison with the year 1915. This was due largely to the establishment of a 7-day layoff as a penalty for the first offense of disobeying the safety rules, 2 weeks layoff for the second offense, and discharge for the third offense. This method seems logically correct, and has been successful both in reducing accidents, and securing the coöperation of some of the old-timers who were the hardest men to convince that they should be more careful in their work.

A monthly bulletin, *The Anode*, is printed and distributed to all employees. One feature of this bulletin is the publication of the accident records of the different mines and shops, and this stimulates interest on the part of both foremen and men. Two prizes of $15 each are offered monthly, for the two best articles on some feature of safety work connected with the operations upon which the author is engaged. One of these prizes is awarded to an employee of the metal mining department, and the other to an employee of the reduction works, or other departments outside of Butte. It also publishes pictures, taken in the mines, of bad and good practices, and these guide the beginner into the right channel. Bulletin boards at the mines covered with bulletins of local interest also tend to keep before the men that important watchword "safety first"; which slogan the majority of men are now approving, realizing that it decreases the misery and hardships caused by needless accidents.

To further promote the interest of the shift bosses and foremen, there is a meeting each month of each of the four mine safety organizations composed of the shift bosses and foremen of the various mines. These meetings are presided over by the general superintendent of mines, who reads and discusses every accident report of the previous month concerning accidents which happened to the men working under the shift bosses and foremen present. The reports of the two mine safety inspectors as to the condition of the working places in the mines visited by them is also read. New safety devices and infringements of safety rules are discussed. This gradually awakens the interest of the mine foremen from a lukewarm condition until it causes them to become most zealous advocates of the safety-first movement.

Sanitation in the mines is promoted. All mines are provided with sanitary toilet cars, sanitary drinking tanks, and some of the mines have installed sanitary drinking fountains. Stopes, cross-cuts, and drifts are provided with water so that men may wet down the dust.

Instruction in first-aid and mine rescue work is an added effort to bring home to the men the fact that the company is seeking their fullest coöperation in safety work. All employees who are desirous of learning first-aid and mine rescue work are taught by an able instructor who has

charge of such work. Two rescue stations are located centrally so as to accommodate the mines for any emergency as quickly as possible. There are two modern automobile ambulances to carry injured men from the mines to the hospitals.

The safe and fireproof installation of electric fans, motors, trolley wires, electric wires, and power lines, is a further factor in safety to which too much attention cannot be paid. They are a constant source of dangerous fires. Experience has shown that one fire may cause more fatalities than will result from any other cause in a whole year.

The following additional signs are suggested for shaft stations:

"Danger. Do not stand near the shaft."

"Danger. Hands Off." (Placed at each shaft bell.)

"Call Bell" or "Buzzer" "1,700, 5–2."

"All hands are requested to keep this plant safe and clean."

"Men are prohibited from eating their lunches on this station."

Added signs for mine workings:

"Danger. Do not use this manway."

"Do not dump here."

"Men working on wire. Do not close switch."

"All lighted candles must be placed in candle sticks or sconces."

The following additional safety rules are suggested, and the reasons given after some of the rules:

1. A miner's first duty on entering his working place shall be to clean and fix his floors so as to give a safe retreat, and then bar down the roof and sides before working under them.

2. In all manways that are vertical, or only slightly inclined, platforms should be constructed every 30 ft., and where these platforms occur substantial guard rails should exist between them and the timber slide.

3. All chutes and working raises shall be covered with 10 by 10 grizzlies. Openings between grizzlies must not be more than 10 in., and they must be properly blocked. They shall also be guarded with railings.

4. At all stations of hoisting compartments where skips are used the stations must be equipped with gates to keep the falling rocks, caused by the overloading of skips, from flying into the station.

5. When explosives are brought to the shaft on the surface to be lowered into the mine, hoisting rock by skips must cease at once.

6. Powder must be lowered in powder cars that are covered to prevent falling rocks in the shaft from striking the powder.

7. When powder is taken off at the station all power to the trolley lines must be shut off and powder trammed to the magazine by hand.

8. Powder magazines must be kept clean.

9. A box, boat, or steel car especially built for hoisting and lowering steel must be used.

10. In side dumping a mine car, a support must be used to prevent the car from tipping backward.

Many accidents, to the extent of broken limbs, caused the enforcement of this rule.

11. When lowering and hoisting timbers in manways, a chain equipped with a hook to be driven into the timber and keep the chain from slipping, must be used attached to the rope.

12. All trolley wires must be boxed, no matter how high above the rail. The boxing must extend 3 in. below the wire.

14. Mine cars, whether loaded or empty, moved by a motor, must at all times be coupled so that the cars are under direct control of the motor man.

15. All timbered sets must be backlagged next to the roof.

16. Cages and skips must be inspected daily, and at the time of inspection all cages in the several compartments must be at the surface.

17. In rill stopes safety chains must be used instead of ropes, because the ropes are quickly cut to pieces by rolling rocks and blasting.

18. Men are forbidden to take short cuts through dangerous workings.

19. Men are forbidden to take wine, beer, or other intoxicating liquors underground.

20. Strangers, and visitors, shall not be allowed underground unless accompanied by some employee designated to accompany them.

21. When timber is being lowered into the mine on multiple-deck cages, and successive deck loads are to be unloaded at different levels, the timber for top levels should be unloaded first, and the remaining decks in consecutive order on levels below.

22. The use of carbide lamps is recommended for spitting fuses in wet drifts, and in shaft sinking.

23. All motor seats must be fixed on the motor car, and they must be equipped with back rests.

There is generally not much clearance underground between the motor and the ground or timber, and there is danger of the motorman sitting out over the motor seat and being squeezed between the motor and mine timber when the seat is not equipped with back rest.

24. "Firebugs" shall be sent through the working places after each shift.

25. Traveling about unlighted stopes or drifts without a light is prohibited. When underground, men should always carry matches and candle, or lamp.

Some of these rules were adopted after a fatal or serious accident had occurred because of the dangerous practices we now prohibit, although before the occurrence of the accident such practices were not considered dangerous. Past experience has thus proven the necessity for the adoption and strict enforcement of the rules.

A. S. Richardson, Safety Engineer, Butte, Mont.—In the report, mention is made of the use of printed books of safety rules for distribution to workmen to inform them of the regulations that have been made to prevent accidents, and by which they are to be governed. In some cases men are required to pass an examination to show their knowledge of the contents of these books of safety rules, but in other cases no such effort is made. At a large mine, or plant, the task of undertaking this examination of new men entails a lot of work, and since it is likely that the information, once gained, is quickly forgotten, it is doubtful whether the purpose for which the rule books are distributed is not better attained by the use of bulletin boards, or other publicity measures, which renew interest periodically.

The establishment of any safety rule should also be accompanied by the establishment of a definite penalty to be imposed in case of violation. Making safety rules that can be broken with impunity is worse than useless, yet none the less common.

Bulletin boards, to be effective, must be placed where men have time to stop and look at them, and must present information that is constantly new and interesting. Bulletin boards presenting a large quantity of stale material, and placed near a timekeeper's office, past which the men always pass when in a hurry, have little value. Bulletin boards located near a drinking fountain, or some similar place, where men may be when they have a minute to spare, and which present information that they can look over in a short space of time, have great value, particularly if the information has local interest, and is changed at reasonably short intervals.

Where foreigners are employed, it is desirable that printed bulletins should appear in their native language. Pictures are, however, universally and quickly intelligible, so that they are decidedly the most valuable material to use.

Attention may be called to the importance of showing by comparative data what each shop, mine, or department is doing, relatively, in the work of reducing accidents. Such comparisons, when justly made, do much to stimulate a spirit of emulation that is of the greatest value.

Unfortunately for the success of the safety-first movement, there is as yet no universally accepted rule for calculating accident rates.

The solution of the problem of arriving at a universally applicable method of determining accident rates is, we believe, much simplified by getting down to fundamentals. Primarily there are two things to be considered: the accident, and the risk of accident. There is as yet no common basis for classifying accidents, but a classification based upon "time lost" and "disability" is being rapidly accepted. The risk of accident under given conditions seems most exactly to be measured by the time a man is employed under those conditions, and so it would only be

logical to adopt some unit of time of employment, or multiple of units, as the basis upon which to calculate accident rates. In the work of the A. C. M. Co.'s Bureau of Safety, the number of accidents of each class per 10,000 shifts, or 10,000 8-hr. working days, are used as accident rates. There have been no objections to the use of this basis of calculation from any of the operating officials; the information has been readily obtainable; and we confess that efforts to supply others with information as to accident rates based upon the average number of men employed were not satisfactory.

As a part of safety work related to metal mines, the provision of sanitary toilet cars, and properly protected cars for drinking water, for use underground, is necessary. In the Butte mines the special steel toilet cars that are used are equipped with hinged lids which keep the cars tightly sealed when not in use, and the cars themselves are partly filled with a disinfecting fluid. At the end of each shift all toilet cars are taken to the surface, emptied through a valve in the bottom of the car, sluiced out, and prepared for further use. Iced drinking water is sent into the mines in closed cars which are securely locked so that nobody can take out ice, or touch the water, and which have the taps protected so that nobody can touch them with their hands or mouths. Open kegs for use in the stopes are being taken out of service as rapidly as possible, and only closed tanks will be used.

Mine ventilation, and the prevention of dust in mine air, are matters whose importance does not seem to have been fully recognized by the metal mining industry in this country. In deep mines, the hot, humid, air is debilitating, affecting the general health of the miner, decreasing his desire and ability to protect himself from injury by accidents, and greatly reducing the amount of work he is able to do. "Miner's Con" is well recognized, but systematic efforts to do away with the conditions that induce it have been much neglected. A far-sighted and humane policy in dealing with these matters is much needed, and undoubtedly all effort made would be well repaid.

In connection with mine ventilation, stress should be laid upon the very great importance of making all housings and surroundings of mine ventilation fans absolutely fireproof. Electrical connections and apparatus used to drive the fans are notorious incendiary agencies, particularly when under the control of unskilled or careless operators. Probably no one thing can endanger so many lives in a metal mine as a fire starting near a ventilation fan which may quickly blow the smoke through the mine workings.

First-aid and mine rescue work is rapidly becoming an important feature of the safety-first campaign, but at many metal mines the work is still limited to the training and practice of a first-aid or mine rescue team which shall represent the mine at contests, and which hardly returns a

maximum of service for the work and expense involved. To provide prompt first-aid assistance for all men who may be injured in a mine which has 50 or more scattered working places is a difficult problem. It cannot be solved merely by training a first-aid team of six men, for it is more than likely that but few of these men will be working when the accident occurs, and that it will take some time to find them. Also, it is quite likely that if the first-aid man has not been training for some time he will be badly out of practice. Effective service requires systematic training and practice of a sufficiently large number of men, and properly planned arrangements so that both first-aid men and material may be quickly available at any locality in the mine.

In conclusion, emphasis should be laid upon the one thing that more than all else is necessary for the success of a safety-first campaign; namely, the necessity for the active support of the management. It is, we believe, an unfortunate fact that some companies have safety-first departments that were created as a sop to progressive opinion, but that these departments accomplish little or nothing toward the purpose for which they were created, merely because they are not supported in their efforts by those active in the management or control of operations. A reduction in the accident rate at any mine or plant can be accomplished only as a result of changes, either in the care used by the men in doing the work, or in the equipment and methods used. Necessarily this means changes in the operating departments, and, generally, outside influences trying to produce such changes will not be welcomed if they can be ignored. The accomplishment of safety-first measures, therefore, demands such active support by the management as will compel sincere coöperation by the operating departments, or else delegation of full authority to the safety department. For complete success there can be no halfway measures.

James L. Price, Great Falls, Mont.—The part a foreman plays in a safety campaign, and how to secure his sustained interest:

It is generally agreed that there is no set rule by means of which best results can be obtained in an accident-prevention campaign. Every industry, each individual plant, must learn to solve the various safety problems that present themselves in some way best adapted to the particular organization in question. Some claim to have obtained good results through the functions of so-called safety committees; others achieve gratifying results in accident-prevention work by placing the onus on their foremen; still others rely upon the combination of both to minimize industrial hazards, while the rest are continually devising methods altogether their own.

After 3 years of intensive study of the subject, we feel that in our particular work, the metallurgical branch of zinc and copper mining,

placing the responsibility upon the foreman appears to be the most efficacious method in accident-prevention work. Undoubtedly, every move and step that has for its aim accident prevention, if not too far-fetched, or absurd, is bound to do some good, even if it does seem to the onlooker "muddling" and to a large extent a waste of effort and time. Since we all, however, are desirous of obtaining in our accident prevention campaign a maximum of efficiency of the lasting kind, we must frown upon any and all methods that savor of the haphazard and the slipshod. Sooner or later such methods are sure to prove harmful for the very simple reason that we acquire a false sense of security, and when disaster of any magnitude overtakes us, we are not prepared to cope with it.

At the fifth Congress of the National Safety Council, A. H. Young, then Supervisor of Labor and Safety of the Illinois Steel Co. at South Chicago, and now Director of The American Museum of Safety, presented a brief paper in which he contended that the surest method to prevent accidents is to hold the foreman in whose department accidents happen to strict accountability. Through an unfortunate association of ideas, most of the 2,000 members of the Council present took the expression of "strict accountability" in the same spirit in which the Romans must have taken Antony's: "Brutus is an honorable man." . . . Yet the more thought one devotes to safety as applicable to a plant, the stronger grows the belief that the solution of the problem is held by the foreman and by no one else. To be sure, as adjuncts in the safety campaign, it does no harm to have a workmen's safety organization; safety committees made up of the rank and file have a certain, if questionable, value, but to prevent the most accidents with the least expenditure of effort and money you have to come to your foreman, to the man who hires and fires; every other means must either coördinate with this method or become subordinate to it.

From the foreman's point of view, however, such reasoning is unjust. He, the foreman, does not want to be held responsible for the accidents that may happen in his department. He has, for so many years, mentally evaded the responsibility that moral and physical evasion comes as if it were second nature to him. Any ill-considered, tactless attempt on the part of the employer to change this attitude of the foreman is bound to bring about friction, misunderstanding, frequently quite silly, in some cases open rebellion. For so many years, the foreman has been in the habit of placing the responsibility on the man who met with the accident, that he cannot, will not, see things now in a different light.

We are all familiar with the report that the average foreman hands in to the management after an accident in his department. In 90 cases out of 100, the cause of the accident, according to the theory advanced by the foreman, is: "His own (meaning victim's) carelessness," "No one

to blame but himself," "Purely accidental," and so on. The foreman makes no mention, probably the idea never occurs to him, as to whether anything might have been wrong with the method that was used at the time the accident happened, or who is responsible for the method in vogue. When cornered and pinned down he will cut the knot with: "he (the man) should have been more careful." In this manner he clears his conscience and is perfectly satisfied that he has done his duty toward his employer and to his subordinates.

Now let us see if we cannot analyze to some extent the reasons that prompt us to assume an entirely different line of thought. Let us look into the reasons that justify us in placing the responsibility on the foreman, and furthermore, why he should bear this responsibility.

First, let us put ourselves in the place of the foreman and let us reason as we want him to reason:

"I am, then, the foreman. An accident has happened to one of my men. Within 10 min. after the accident occurred there will be some one out to investigate the causes and to properly fix the blame. Witnesses will be summoned and questioned, the details of the operation will be gone into, the condition that prevailed at the time of the accident will be analyzed. I know that if a possible remedy, or means of prevention, suggests itself to the investigator, such will be embodied in the report to the management. I know it will be useless for me to say that the man was hurt because of his carelessness, or through the carelessness of a fellow workman. I know that the management will call me in and begin to ask questions which, I somehow feel, reflect upon my ability to handle men. I know that if the report to the management proves that the accident could have been prevented, and would have been prevented by the exercise of closer supervision, or by giving the man more explicit instruction, I am due for a week's vacation without pay. I know that if accidents, even such as I honestly and sincerely consider to be unavoidable, continue to take place in my department, the management will become prejudiced against me, and will begin, before long, to look around for another foreman. In other words, I know that if I care to remain on the job, I shall have to pay more attention to the way my men are doing things. Then, of course, there is another way of looking at this safety-first campaign. At first I thought that it was a fad. I felt that while not altogether a joke, the interest that the management displayed was more or less superficial and perfunctory. I had an idea that if ever things should come to a showdown—production first or safety first—safety first would come in last. I can see now how mistaken I was. Of recent months I have been told explicitly, insistently, firmly, that the safety-first idea is not a side issue and must not be considered as such either by myself or my men. I am almost beginning to suspect that from the management's point of view safety first comes ahead of production.

It is an unheard of state of affairs. It is incredible, and that is the reason it took me so long to grasp the significance of this new phase in the life of the plant. But if such is the case, then there is a bright side to it. If I make it a point to prevent accidents in my department, I think I can accomplish it just as well as I did other things. Then, because I am painstaking, because I tell my men what I want done and how I want it done, because when there is any doubt in my mind as to whether I made myself clear to my men, I show them how to do things safely and efficiently, because I am on the job 60 min. an hour and make every minute count, I actually succeed in keeping accidents in my department down.

"Months go by. No one of my crew gets hurt. No one of my men caused an accident to men in other departments. I feel the management is aware of the work I am doing. The management sees that I have fallen into line and am doing the things other foremen declared were impossible. My monthly check begins to carry with it a little prize, not much to be sure, still it is something tangible, it is proof of recognition. The management thinks well of my efforts and that little extra money is just a sign of appreciation. . . . Who knows, but some day there may be an opening to a somewhat bigger job, with some more pay and still better prospects. . . . What then am I going to do about accident prevention?

"I know what I am going to do.

"I shall see that each man who is hired by me is thoroughly informed what he is to expect from me if he is caught taking a chance. I shall make it very plain to him that I will not tolerate for an instant in my department any one who disregards safety rules. It will be an open secret among my men that the careless, thoughtless, indifferent man will be gotten rid of at the very first opportunity . . ."

Thus the foreman. Now we place ourselves in the position of the workman who is employed by the foreman whose line of thought we attempted to reproduce:

"I, the workman, know that there are orders out from higher up to cut out accidents at this plant. I know that Bill, John, Bob, Tom, Dick, Steve, have been let out at some time or other, because one did not wear goggles while babitting, the other for throwing a plank from off the roof without first looking to see whether there was some one below, the third for using a mushroomed chisel bar, the fourth for leaving a manhole open and failing to put the guard up, the fifth for coming out to the works under the influence of liquor, the sixth for not reporting a dangerous condition he was aware of . . . I know that if my job is worth keeping I must not jeopardize my foreman's bread and butter, for 'they' put it up to him, and he puts it up squarely to me. I know that if I am ever caught indulging in an unsafe practice I am going to be

laid off or discharged. As I think it over, I can be as careful as they want me to."

So much for the workman.

If our analysis is correct, if it is understood by the foreman and by his men that the foreman is looked to for a minimum of accidents just as much as for a maximum of production, it is fairly safe to say that preventable accidents will not happen with dismaying frequency and discouraging persistency.

The question presents itself immediately: How can we induce the foreman to look at the accident-prevention problem from our point of view? How are we to teach him to consider our problem, his problem? How are we to convince the foreman that the matter of accident prevention is up to him? Shall it be through instilling in him a fear of punishment? Or shall we hold out to him the hope of reward? Which of the two offers the stronger incentive? Or perhaps, shall we use as a propelling motive both of these interest-developing stimulants? These are the all-important questions in connection with accident-prevention work that managements are now called upon to decide. When an accident takes place and after having analyzed the surface and underlying causes we can fasten the blame on to the foreman, what are we going to do about it? Shall we lay the foreman off? Shall we discipline him in some other manner? On the other hand, let us assume that a foreman has no accidents at all, let us say, for the sake of argument, that as far as accidents are concerned this foreman's record is absolutely clear. Are we to pay him a bonus? Or raise his salary? Or mark him for promotion? Or, a judicious application of both theories would work out well. That is to say, combine fear of punishment with hope of reward, or rather hold out to him the contrast between the two. For it would seem that if when a foreman has accidents he is either demoted, or discharged or if, on the other hand, when he takes an interest in safety, eliminates unsafe practices, insists upon the exercise of care, keeps his tools and machinery in good condition, his pay is increased and he stands in line for promotion, the contrast between the two extremes is made stronger, and so is the incentive to make good.

The Hancock Jig in the Concentration of Lead Ores

BY HAROLD RABLING,* B. M. E., BONNE TERRE, MO.

(St. Louis Meeting, October, 1917)

THE following notes are taken from results obtained on a standard 25-ft. Hancock jig[1] tested during regular operation in the Bonne Terre mill of the St. Joseph Lead Co. The object of the tests was to determine the conditions for most effective work, and the nature of products that could be made.

The material treated was sized between 9-mm. and 2-mm. round-hole screens, and was constant throughout the experiments. Three products were made, concentrates from the first three hutches, middlings from the fourth and fifth, and tailing from the sixth. The middlings were crushed in one pair of Allis-Chalmers Style B, 30 by 14-in. rolls, and elevated with the original ore to the 2-mm. screens, oversize of which formed the jig feed.

While this arrangement was very simple, it gave rise to undesirable crowding of the circuit. Table 1 shows tonnage and assay values of the jig feed and products at the beginning of the experiments, and brings out the excessively large amount of middlings, comprising 63.3 per cent. of the total material fed to the jig. The rolls, being heavily overloaded, did very little crushing, a screen analysis of the product showing that only 27 per cent., or 128 tons, was crushed fine enough to pass to the tables. The remaining 347 tons returned to the jig with 400 tons of original ore. At times, this overcrowding became much worse, and the tonnage in circuit grew to enormous proportions. It became imperative

TABLE 1.—*Work of Hancock Jig at Beginning of Tests*

Material	Tons per 24 Hr.	Assay, Per Cent. Lead	Tons Lead per 24 Hr.
Feed	750.0	4.06	30.500
Concentrates	16.5	70.00	11.550
Middlings	475.0	3.50	16.625
Tailing	258.5	0.90	2.325

* Testing Engineer, St. Joseph Lead Co.

[1] For description of Hancock Jig see *Trans.* (1913), **46**, 213.

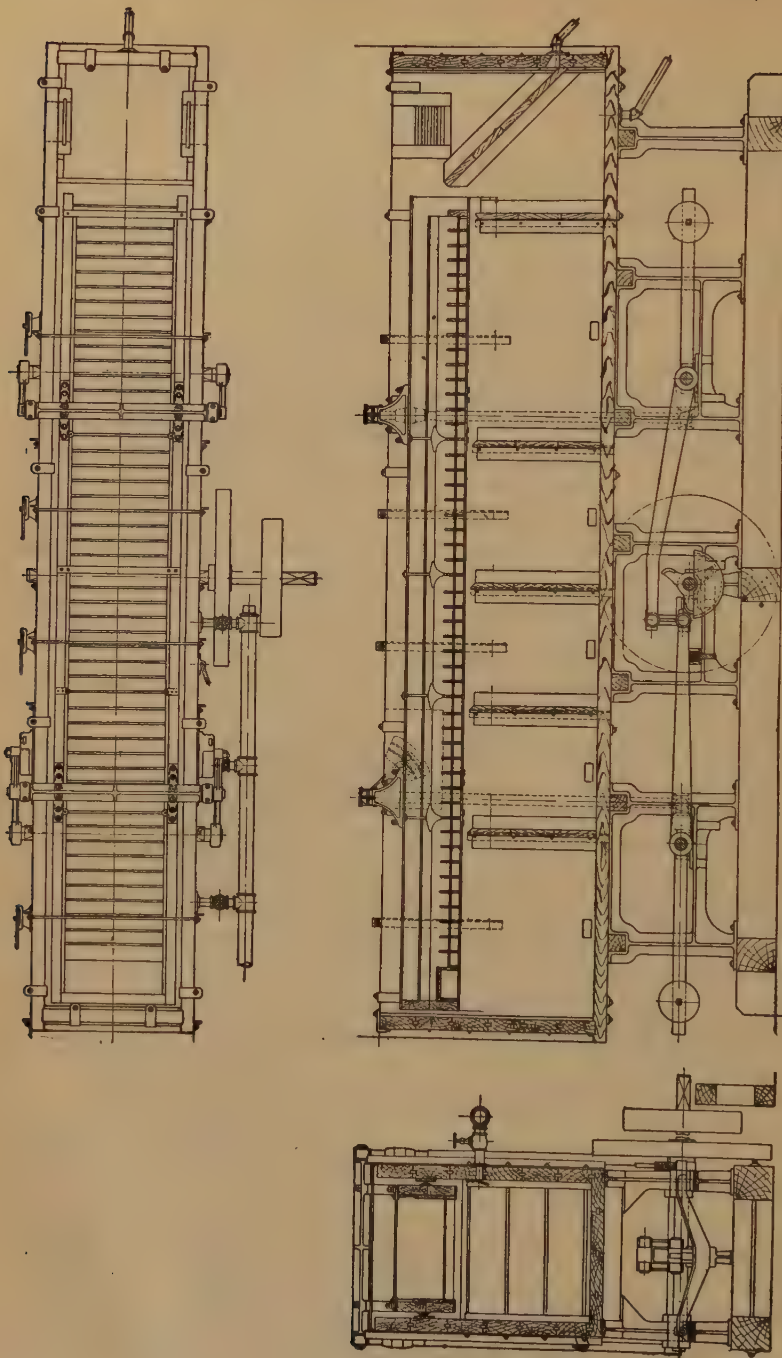

FIG. 1.—THE HANCOCK JIG.

either to decrease the amount of middlings, or to provide more crushing capacity, and series of experiments were begun along both lines. This paper deals only with the attempt to improve the work of the jig as then equipped.

Before commencing the experiments, a combined sizing and sorting test was made on an average sample of the jig feed. Tyler standard screens were used for the sizing test and all sorting was done by hand. Table 2 shows the result of this test, and Fig. 2 and 3 represent the same thing graphically. The horizontal divisions in the diagrams represent the percentages of material on each size, and the vertical divisions the percentages of free galena, middling, and free gangue as obtained by

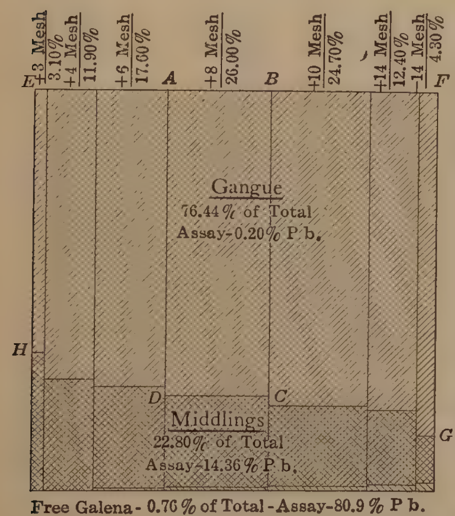

FIG. 2.—SCREEN ANALYSIS OF JIG FEED AND PERCENTAGES OF FREE GALENA, MIDDLINGS, AND FREE GANGUE IN EACH SIZE.

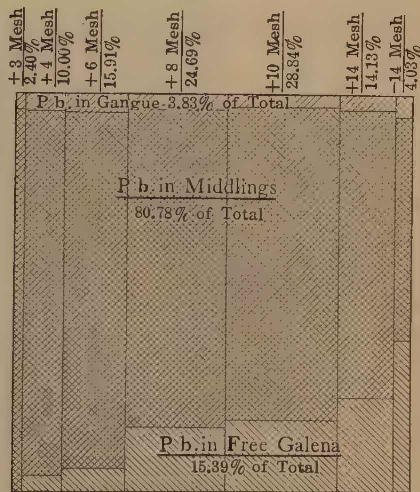

FIG. 3.—SCREEN ANALYSIS OF LEAD IN JIG FEED AND PERCENTAGES OF TOTAL LEAD IN EACH SIZE AS FREE OR INCLUDED GALENA.

sorting. Areas, therefore, represent percentages of the total weight of sample. For instance, in Fig. 2 the length *AB* is 26 per cent. of the total length of the diagram, and represents the amount of material between 6 and 8-mesh. The length *AD* is 76.65 per cent. of the total height of the diagram and represents the percentage of the 6 to 8-mesh material, which occurs as free gangue. The area *ABCD*, therefore, shows the percentage of the total sample occurring as free gangue between 6 and 8-mesh and is equal to 19.927 per cent. of the total area of the diagram, while the area *EFGH* shows the total amount of free gangue in the sample, or 76.44 per cent.

The results obtained by this test led to the belief that it was possible greatly to reduce the tonnage of hutched product without affecting the

TABLE 2.—*Sizing and Sorting Test on Jig Feed*

Mesh	Per Cent. Weight	Assay, Per Cent. Lead	Per Cent. of Total Lead Content	Per Cent. of Weight on Each Mesh			Per Cent. of Total Weight			Assay, Per Cent. Lead			Per Cent. of Lead on Each Mesh			Per Cent. of Total Lead Content		
				Free Galena	Midd.	Free Gangue	Free Galena	Midd.	Free Gangue	Free Galena	Midd.	Free Gangue	Free Galena	Midd.	Free Gangue	Free Galena	Midd.	Free Gangue
On 3...	3.1	3.14	2.40	....	34.75	65.25	....	1.078	2.022	....	8.64	0.22	....	95.38	4.62	....	2.290	0.111
On 4...	11.9	3.41	10.00	0.16	28.00	71.84	0.019	3.330	8.551	86.8	11.22	0.18	4.06	92.14	3.80	0.407	9.213	0.380
On 6...	17.6	3.66	15.91	0.26	25.86	73.88	0.046	4.550	13.004	86.7	12.62	0.24	6.14	89.02	4.84	0.976	14.165	0.769
On 8...	26.0	3.85	24.69	0.78	22.57	76.65	0.203	5.870	19.927	80.3	13.71	0.17	16.34	80.27	3.39	4.030	19.819	0.836
On 10...	24.7	4.73	28.84	1.02	20.69	78.29	0.252	5.110	19.338	82.0	18.09	0.20	17.80	78.90	3.30	5.140	22.745	0.955
On 14...	12.4	4.62	14.13	1.32	18.81	79.87	0.164	2.335	9.901	81.6	17.84	0.22	23.40	72.79	3.81	3.300	10.295	0.538
Through 14.....	4.3	3.80	4.03	1.80	12.20	86.00	0.077	0.525	3.698	80.5	17.45	0.26	38.15	56.00	5.85	1.537	2.257	0.237
Total....	100.0	4.06	100.00	0.76	22.80	76.44	0.761	22.798	76.441	80.9	14.36	0.20	15.39	80.78	3.83	15.390	80.784	3.826

grade of tailings. Table 3, worked out from the results of the sorting, shows the work that might be expected from a theoretically perfect concentrating machine, and differs considerably from the record of the actual work obtained (Table 1). An unnecessarily large amount of free gangue seemed to be passing through the jig screens, and a sorting test on the jig products (Table 4) confirmed this, the fifth hutch especially containing almost as much free gangue as the tailing.

In the investigations following the above work, special attention was given to the possibility of decreasing the amount of gangue being hutched with the middlings, and a summary of results obtained and conclusions arrived at, is given here.

TABLE 3.—*Theoretically Perfect Work Obtainable on Material Fed to Jig*

Material	Tons per 24 Hr.	Assay, Per Cent. Lead	Tons Lead per 24 Hr.
Feed	750.00	4.06	30.500
Concentrates	5.70	80.90	4.753
Middlings	171.00	14.36	24.600
Tailings	573.30	0.20	1.147

TABLE 4.—*Summary of Sorting Test on Jig Products*

Material	Assay, Per Cent. Pb	Per Cent. of Total Weight			Assay, Per Cent. Lead			Per Cent. of Pb Cont.		
		Free Galena	Midd.	Free Gangue	Free Galena	Midd.	Free Gangue	Free Galena	Midd.	Free Gangue
Concentrates	74.40	64.4	35.6	....	82.3	60.40	....	71.1	28.9	
4th hutch	6.78	....	31.2	68.8	....	20.70	0.48	....	95.4	4.6
5th hutch	1.53	....	16.7	83.3	....	8.33	0.17	....	90.3	9.7
Tailing	0.88	....	15.5	84.5	....	4.91	0.14	....	84.4	15.6

Operation of the Jig

The following factors may be taken as having most effect on the work of the Hancock jig:

1. The speed of the jig.
2. Length of vertical stroke.
3. Length of horizontal stroke.
4. Depth of bed.
5. The size of the grains forming the bed.
6. The specific gravity of the grains forming the bed.

1. *The speed of the jig* was varied between wide limits—from 170 to 210 strokes per minute. The high speeds gave increased tonnage of hutch product, and were objectionable from a mechanical standpoint. At the slow speed of 170, difficulty was experienced in keeping the bed moving freely, it having a tendency to pack, and lose its quicksand effect.

A speed of 190 to 195 strokes per minute was finally decided on as being the most desirable.

2. *The length of vertical stroke*, by affecting the pulsion and suction velocities affects the tonnage of hutched product, increased length causing more material to be hutched, and *vice versa*. It varies between 3/8 and 3/4 in., the adjustment being made by the operator according to the lead content of the material treated.

3. *The horizontal stroke* should be as long as possible, in order to have, above the bedding, a thin layer of ore. In treating large tonnages, the long stroke is particularly desirable, the improved separation in a thin layer more than compensating for the shorter time the material is under treatment. A minimum horizontal stroke of 3/4 in. is used at Bonne Terre, and it is probable that greater capacity could be had by making the stroke much longer.

4. *The depth of bed on the Hancock jig* is determined by the height of the slats above the screen. No experiments were made with variations of this distance, all jigs being operated with 3 in. of bed depth.

5 and 6. *The size and specific gravity* of the grains forming the bed are determined entirely by the size of the jig screens. No artificial bedding has been supplied at any time, as experiments have shown that with a carefully designed system of screens any desired bed can be produced and held, relying on the ore to supply all the material necessary. The nature of the bed being the most important factor influencing the work of the jig, the main part of the investigation was taken up by the effort to work out a system of screens that would give satisfactory work under average conditions, and would be flexible enough to take care of extreme variations in the grade and nature of the material treated.

The system that was finally evolved is given in Table 5, and while it would probably only be applicable to the Bonne Terre ore, the principles of its evolution may be found useful in the solution of similar problems elsewhere.

Table 5.—*System of Screens Used on Hancock Jig*

4-mm. round hole to rib	3	3 ribs
5-mm. round hole to rib	15	12 ribs
7-mm. round hole to rib	16	1 rib
6-mm. round hole to rib	22	6 ribs
7-mm. round hole to rib	30	8 ribs
8-mm. round hole to rib	31	1 rib
6-mm. round hole to rib	41	10 ribs
9-mm. round hole to rib	43	2 ribs
5-mm. round hole to rib	end	3 ribs

Close examination of the jigs brings out the following points:

1. Stratification of the material into layers of varying richness takes

place almost immediately after it reaches the tray. This is helped by the strong pulsion effect on the first hutch, where suction is minimized by the fineness of the bed of free galena, which lies in a heavy mass on the screen.

2. In order to remove all the material containing lead, both as free galena and middling, a bed is required that gradually decreases in specific gravity, with increasing size of grains. This was found to be most effectively obtained by arranging the tray with an occasional row of large holes, to take out the bedding in steps.

3. The bed may be likened to a screen of varying size holes, this size depending on the size of grains. Any section should be just long enough to give a good "screening efficiency" on such material as will pass through it. For instance, a bed composed of grains of specific gravity 4.0 will hutch a product assaying about 40 per cent. lead. If it be found that all the 40 per cent. material present will pass through in a length of 2 ft., there is a loss of efficiency by extending this particular bed any farther

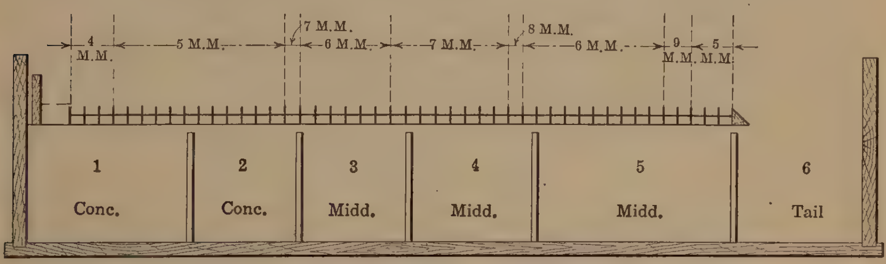

FIG. 4.—SYSTEM OF SCREENS USED ON HANCOCK JIG.

than 2 ft. A row of large holes at this point will allow the heavier particles to pass through, leaving a bed of lighter specific gravity for the next section of tray. A bed which changes rapidly in character will therefore allow all parts of the jig to do equal work, and give maximum efficiency.

4. There are two distinct operations to be considered. First, the making of clean concentrates, and second, the making of clean tailing. These are justly considered apart, and of the two the latter is by far the more difficult. The specific gravity of galena is 7.40 and of gangue 2.80, and while the separation of these two minerals is easy, the material received on the jig is made up of grains ranging from one extreme to the other. By sorting the products of the jig (Table 4) it is found that concentrates are made up of grains of free galena of grade 82.3 per cent. lead, and middlings of 60.4 per cent. lead. Specific gravities are 7.00 and 5.20 respectively; a difference of 1.80 tailings are made up of middlings of grade 4.91 per cent. or specific gravity 2.95, and gangue of grade 0.14 per cent., or specific gravity 2.82, a difference of 0.13. These figures

illustrate the wider range of specific gravity permissible in concentrates, and therefore the relative ease of separation of the high-grade material.

5. The small amount of free galena present, as shown in Table 2, also helps to simplify its separation, and it would appear to be good practice to hutch the free galena in as short a distance as possible, reserving the greater length of the tray for the more difficult work of separating middling from tailing. An attempt to do this was very successful, and concentrates are now made on only two hutches, occupying 80 in. of screen length. The larger part is caught in the first hutch, the second acting as a safeguard to prevent free galena being carried over into the middling hutches. As a general rule, the tonnage of product from the second hutch is very small.

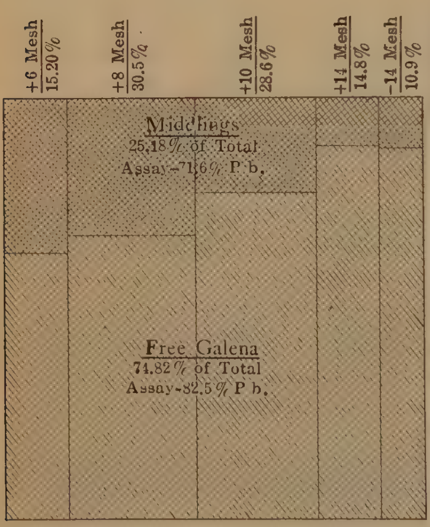

FIG. 5.—SCREEN ANALYSIS OF FIRST HUTCH PRODUCT.

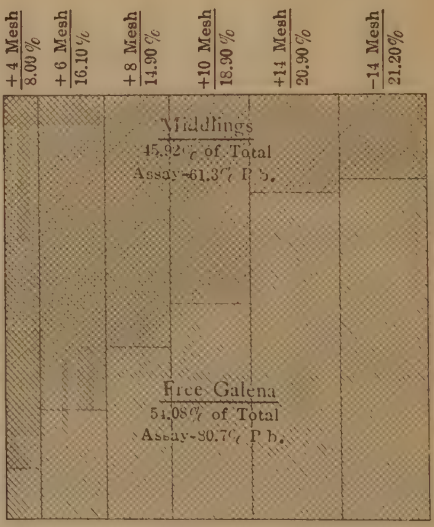

FIG. 6.—SCREEN ANALYSIS OF SECOND HUTCH PRODUCT.

6. The separation on the final section of the tray, or the part over the fifth hutch, must necessarily be rough, owing to the low grade of the middling remaining, and the small difference in specific gravity between it and the free gangue. The bed consists of large-size grains of low-grade middling, and the heaviest tonnage of hutched product is made here. At the end of the fifth hutch, two ribs of holes, large enough to take out all the remaining bedding, are used.

7. Variations in grade of the feed introduce complications that must be allowed for in the screen layout. An excess of bedding material, due to higher-grade ore than usual, causes an accumulation of heavy grains that cannot pass through the screens, while low-grade ore will allow the bed to become too light, and the hutched product will contain too much gangue.

By arranging the screen to take out the bedding in steps, as mentioned before, such an accumulation cannot extend very far, but extreme varia-

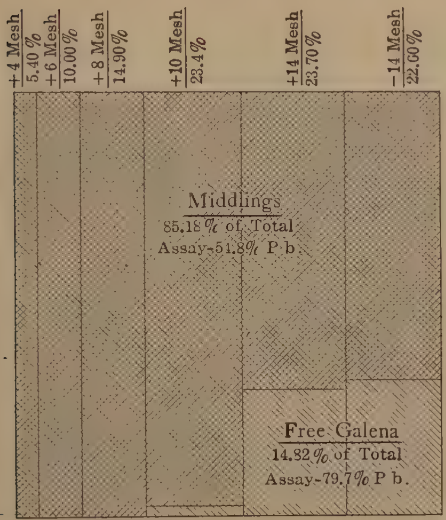

FIG. 7.—SCREEN ANALYSIS OF THIRD HUTCH PRODUCT.

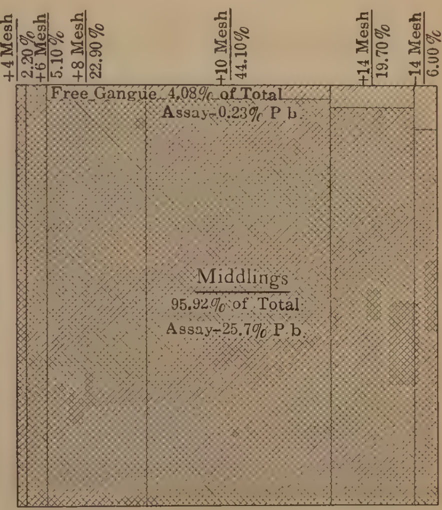

FIG. 8.—SCREEN ANALYSIS OF FOURTH HUTCH PRODUCT.

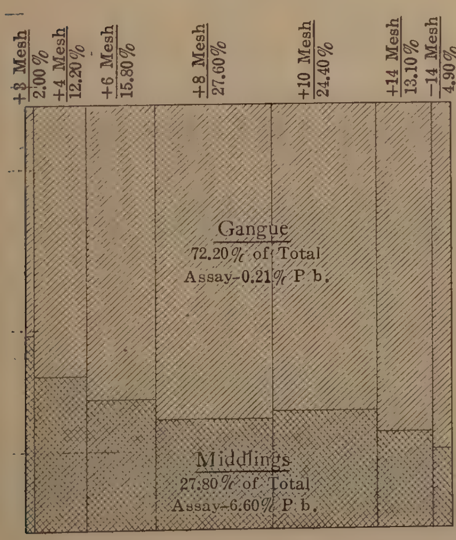

FIG. 9.—SCREEN ANALYSIS OF FIFTH HUTCH PRODUCT.

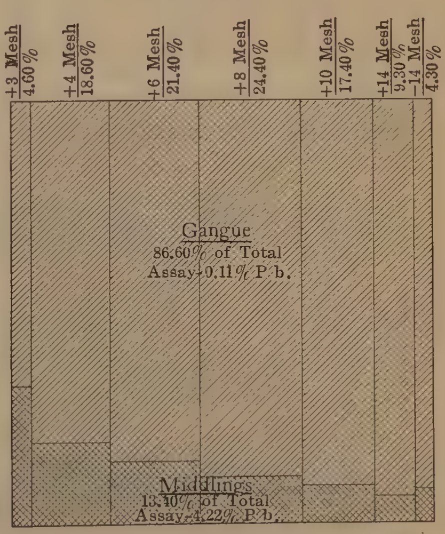

FIG. 10.—SCREEN ANALYSIS OF JIG TAILING.

tions are best taken care of by the operator, who shovels off excess bed, and plugs the large holes in the screen when bedding material is scarce.

8. Wire cloth and punched plate were both tried for the jig screens,

TABLES 6–11.—*Sizing and Sorting Test on Jig Products*

Mesh	Per Cent. Weight	Assay, Per Cent. Lead	Per Cent. of Total Lead Content	Per Cent. of Weight on Each Mesh		Per Cent. of Total Weight		Assay, Per Cent. Lead		Per Cent. of Lead on Each Mesh		Per Cent. of Total Lead Content	
				Free Galena	Midd.	Free Galena	Midd.	Free Galena	Midd.	Free Galena	Midd.	Free Galena	Midd.
6. 1st Hutch													
On 3	......	......	......	......	......	......	......	......	......	......	......	......	......
On 4	......	......	......	......	......	......	......	......	......	......	......	......	......
On 6	15.2	80.3	15.29	63.0	37.0	9.58	5.62	82.1	76.9	64.50	35.50	9.87	5.42
On 8	30.5	78.6	30.08	67.2	32.8	20.50	10.00	82.7	70.4	70.65	29.35	21.25	8.83
On 10	28.6	79.7	28.60	77.2	22.8	22.10	6.50	81.7	71.3	79.62	20.38	22.78	5.82
On 14	14.8	81.2	15.09	88.3	11.7	13.07	1.73	83.1	66.7	90.42	9.58	13.65	1.44
Through 14	10.9	79.8	10.94	87.8	12.2	9.57	1.33	82.0	65.1	90.00	10.00	9.85	1.09
Total	100.0	79.7	100.00	74.8	25.2	74.82	25.18	82.5	71.6	77.40	22.60	77.40	22.60
7. 2d Hutch													
On 3	......	......	......	......	......	......	......	......	......	......	......	......	......
On 4	8.0	68.2	7.60	12.0	88.0	0.96	7.04	79.0	66.8	13.90	86.10	1.06	6.54
On 6	16.1	68.9	15.45	26.0	74.0	4.18	11.92	80.2	65.1	30.15	69.85	4.63	10.82
On 8	14.9	71.3	14.80	41.0	59.0	6.11	8.79	80.6	65.0	46.30	53.70	6.86	7.94
On 10	18.9	69.5	18.30	51.3	48.7	9.70	9.20	81.7	56.8	60.20	39.80	11.02	7.28
On 14	20.9	74.7	21.75	77.2	22.8	16.15	4.75	80.9	53.4	83.75	16.25	18.22	3.53
Through 14	21.2	74.7	22.10	80.1	19.9	16.98	4.22	80.0	53.2	85.85	14.15	18.98	3.12
Total	100.0	71.8	100.00	54.1	45.9	54.08	45.92	80.7	61.3	60.77	39.23	60.77	39.23
8. 3d Hutch													
On 3	......	......	......	......	......	......	......	......	......	......	......	......	......
On 4	5.4	59.9	5.54	......	100.0	......	5.40	......	59.9	......	100.00	......	5.54
On 6	10.0	58.4	9.98	......	100.0	......	10.00	......	58.4	......	100.00	......	9.98
On 8	14.9	59.8	15.25	......	100.0	......	14.90	......	59.8	......	100.00	......	15.25
On 10	23.4	59.2	23.68	2.3	97.7	0.54	22.86	80.1	58.7	3.12	96.88	0.74	22.94
On 14	23.7	63.3	25.65	29.8	70.2	7.06	16.64	79.3	56.5	37.30	62.70	9.57	16.08
Through 14	22.6	51.4	19.90	32.0	68.0	7.22	15.38	79.8	38.2	49.60	50.40	9.88	10.02
Total	100.0	58.5	100.00	14.8	85.2	14.82	85.18	79.7	54.8	20.19	79.81	20.19	79.81

9. 4th Hutch

Mesh	Per Cent. Weight	Assay, Per Cent. Lead	Per Cent. of Total Lead Content	Per Cent. of Weight on Each Mesh		Per Cent. of Total Weight		Assay, Per Cent. Lead		Per Cent. of Lead on Each Mesh		Per Cent. of Total Lead Content	
				Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue
On 3	....	....	....	....				....	....	....	....	....	....
On 4	2.2	37.0	3.30	100.0	....	2.20	....	37.0	....	100.00	....	3.30	....
On 6	5.1	29.1	6.01	100.0	....	5.10	....	29.1	....	100.00	....	6.01	....
On 8	22.9	27.1	25.14	96.2	3.8	22.03	0.87	28.2	0.18	99.975	0.025	25.14	0.006
On 10	44.1	22.6	40.22	96.6	3.4	42.59	1.51	23.4	0.20	99.966	0.034	40.21	0.014
On 14	19.7	25.2	20.13	94.7	5.3	18.66	1.04	26.6	0.24	99.950	0.050	20.12	0.010
Through 14	6.0	21.4	5.20	89.0	11.0	5.34	0.66	24.0	0.36	99.815	0.185	5.19	0.009
Total	100.0	24.7	100.00	95.9	4.1	95.92	4.08	25.7	0.23	99.961	0.039	99.96	0.039

10. 5th Hutch

Mesh	Per Cent. Weight	Assay, Per Cent. Lead	Per Cent. of Total Lead Content	Per Cent. of Weight on Each Mesh		Per Cent. of Total Weight		Assay, Per Cent. Lead		Per Cent. of Lead on Each Mesh		Per Cent. of Total Lead Content	
				Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue
On 3	2.0	1.15	1.16	46.0	54.0	0.92	1.08	2.32	0.15	93.05	6.95	1.08	0.08
On 4	12.2	1.63	9.95	36.1	63.9	4.40	7.80	4.07	0.24	90.49	9.51	9.01	0.94
On 6	15.8	1.56	12.38	30.5	69.5	4.80	11.00	4.66	0.20	91.07	8.93	11.27	1.11
On 8	27.6	2.20	30.47	26.0	74.0	7.15	20.45	7.96	0.18	93.94	6.06	28.62	1.85
On 10	24.4	2.36	28.90	27.4	72.6	6.67	17.73	7.97	0.25	92.37	7.63	26.68	2.22
On 14	13.1	2.04	13.42	22.7	77.3	2.96	10.14	8.25	0.22	91.66	8.34	12.30	1.12
Through 14	4.9	1.67	3.72	18.4	81.6	0.90	4.00	7.42	0.18	90.28	9.72	3.35	0.37
Total	100.0	2.00	100.00	27.8	72.2	27.80	72.20	6.60	0.21	92.31	7.69	92.31	7.69

11. Tailing

Mesh	Per Cent. Weight	Assay, Per Cent. Lead	Per Cent. of Total Lead Content	Per Cent. of Weight on Each Mesh		Per Cent. of Total Weight		Assay, Per Cent. Lead		Per Cent. of Lead on Each Mesh		Per Cent. of Total Lead Content	
				Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue	Midd.	Free Gangue
On 3	4.6	1.04	7.26	32.8	67.2	1.510	3.090	2.92	0.13	91.67	8.33	6.655	0.605
On 4	18.6	0.67	18.90	19.3	80.7	3.585	15.015	3.07	0.10	88.00	12.00	16.630	2.270
On 6	21.4	0.60	20.70	14.8	85.2	3.165	18.235	3.84	0.08	89.20	10.80	18.490	2.210
On 8	24.4	0.63	23.23	10.9	89.1	2.660	21.740	5.12	0.08	88.67	11.33	20.625	2.605
On 10	17.4	0.63	16.74	8.8	91.2	1.530	15.870	6.60	0.06	91.40	8.60	15.300	1.440
On 14	9.3	0.69	9.70	6.6	93.4	0.615	8.685	6.24	0.30	59.30	40.70	5.750	3.950
Through 14	4.3	0.53	3.47	7.8	92.2	0.335	3.965	4.60	0.20	65.50	34.50	2.280	1.190
Total	100.0	0.66	100.00	13.4	86.6	13.400	86.600	4.22	0.11	85.73	14.27	85.730	14.270

with a final decision in favor of the latter. While it is sometimes claimed that wire cloth gives less trouble by blinding, it was found in these tests that there was very little difference in this respect, while punched plate had the advantage of being more easily cleaned.

The nature of the products obtained with the screen system given in Table 5, are shown in detail in Tables 6 to 11 and Fig. 5 to 10. The first two hutches produce good concentrate, the third and fourth a rich middling, and the fifth a low-grade middling containing a large proportion of free gangue. Analysis of the tailing shows some middling escaping. The lead in this occurs as very small pieces sticking to the large pieces of gangue, or included in them, and owing to its low specific gravity it is doubtful whether this low-grade middling could be saved without recrushing the whole tailing. Table 12 shows results of a tonnage test on the jig, equipped with the screen system given in Table 5.

TABLE 12.—*Work of Hancock Jig Under Present Conditions*

Material	Tons per 24 Hr.	Assay, Per Cent. Lead	Tons Lead per 24 Hr.
Feed	505.0	4.74	23.95
Concentrates	7.7	75.00	5.79
Middlings	195.0	8.16	15.91
Tailing	302.3	0.74	2.25

The rolls have a feed of 195 tons per 24 hr., of which 58 per cent., or 113 tons, is crushed fine enough to pass through 2-mm. screens. The remaining 82 tons returns to the jig with 423 tons of original ore. The lighter load on the jig enables it to do better separation, and a cleaner concentrate and lower tailing are possible.

Subsequently the tonnage of ore treated in the mill was raised from 1,800 to 2,200 tons per day without extra jig equipment, and with lower tailings than previously, the gain being almost entirely due to the more efficient work done by the jigs.

DISCUSSION

A. P. WATT, Mine La Motte, Mo.—The information in Mr. Rabling's paper is of great value to users of the Hancock jig. In the Lead Belt, these jigs treat a larger tonnage than any other type of concentrating machine, and produce tailing of a larger size. This indicates the necessity for logical operation of the Hancock jig. The jigs receive at least 75 per cent. of the total original feed, the other 25 per cent. going to tables and flotation machines. Jig operation is therefore of paramount importance on account of the large tonnage treated, and the great volume of coarse tailing, which ranges from 10 to 2 mm.

The tonnage of tailing made by jigs will vary with the efficiency of the crushing machinery used for re-crushing the jig middling. The use of ball-mills will diminish the tonnage of jig tailing. The ratio of jig tailing to jig feed may vary from 50 to 80, depending on the means used for crushing the middling. The loss of mineral by jigs, theoretically 4 per cent. but sometimes actually 16 per cent., is three to four times the combined loss of tables and flotation plants. This again shows the importance of proper jig operation.

Mr. Rabling's figures are of special interest to the operators in the Lead Belt, because the tonnage of jig middling that may get into circulation with poor supervision sometimes reaches enormous proportions. At times, under poor operating conditions, the returned feed may equal the tonnage of original feed.

Furthermore, if the jig bed does not receive proper attention, the fifth hutch product may assay lower than the tailing. When this occurs, the jig is not acting as a jig but simply as a screen, the fine material being sifted through, while the larger sizes are tailed over; owing to their size and the presence of included grains, they assay higher in lead than the fifth hutch product. Under poor operating conditions, the fifth hutch may contain 90 per cent. clean gangue.

Concentration Practice in Southeast Missouri

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(St. Louis Meeting, October, 1917)

CONTENTS

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Foreword

The problem of concentrating the disseminated lead ore of southeast Missouri is extremely simple. The economic mineral is galena and the gangue is dolomite. The ore assays from 4 to 6 per cent. lead, 80 per cent. of which is recovered in a 70 per cent. concentrate. The ore is crushed through 10 mm. and sized on 2 mm.; the oversize is jigged, while the undersize is tabled, and the slime treated by flotation. The district is the largest lead producer in the world, the output of metallic lead for the year 1915 being 183,906 tons. Over 20,000 tons of ore a day is being treated.

The management of the different properties is progressive and always open to suggestion. The fact, however, must not be lost sight of, that the problem in the district is peculiar, as the average ore contains but 80 to 120 lb. of a metal normally selling at about 4c. per lb. The low price of the metal limits the treatment system to one of low cost. An elaborate treatment is neither logical nor advisable. An engineer first visiting the district may consider that the method of treatment could be improved

and, in some cases, such may be the case. On the other hand, the method of treatment followed is largely one of evolution and, consequently, probably not far from correct. However, the ultimate method of treatment of the ore is a problem that has not yet been solved. The methods of concentration used at the different properties are practically identical, differing not in principle, but only in detail. This is to be expected, because of the similarity of the ore in the different properties.

Location of District

The disseminated lead district of southeast Missouri—locally known as the "Lead Belt"—is 70 miles south of St. Louis. It is reached by the

Fig. 1.—General map showing location of the southeast Missouri lead district in relation to St. Louis and other cities.

Mississippi River and Bonne Terre R. R., which runs over the rails of the St. Louis, Iron Mountain & Southern R. R. from St. Louis to Riverside. From Riverside to Doe Run, which is the end of the line, the Bonne Terre road operates over its own tracks. This railroad, which is owned by the St. Joseph Lead Co., attends to both the freight and passenger business of the district. The Illinois Southern passes through the district at Flat

River and Elvins, terminating at Bismarck on the Iron Mountain R. R. This road gives access to the Mississippi River to the east at Ste. Genevieve, and thus permits tapping the coal fields of Illinois. The Lead Belt is wholly within St. Francois County, the principal towns being Bonne Terre, Flat River, Elvins, Desloge and Leadwood. These towns are all within the Lead Belt, while Farmington, the county seat, is 9 miles to the southeast. The location of the district in relation to St. Louis and other cities is shown in Fig. 1.

Geology and Mining

There is no vein formation in the district, the ore occurring in large bodies, roughly horizontal. The ore is not shattered nor brecciated, but as a general thing very compact, and large orebodies gradually fade out into nonprofitable dolomite.

The ore occurs at a depth of 300 to 400 ft. and varies from 30 to 100 ft. in thickness. It is mined by underground methods, no open-cut work being done in the district around Flat River, but such a method is in use at Mine La Motte. H. A. Guess[1] has fully described the mining methods in use in the district, and as the geology will be made the subject of a separate paper, this subject will not be further discussed.

Topography and Climate of the District

The country is rolling, the average elevation being about 800 or 900 ft. The topography is not sufficiently steep to permit the operation of a gravity-type mill. The winters are mild, snow seldom remaining on the ground for more than 3 or 4 days at a time, the temperature seldom falling below zero. The summers are hot, as one would suspect who is familiar with St. Louis in the summer time, the temperature during the days frequently standing around 100°. The average rain fall is 40 in. per year.

Lead Production of the District

Tabulated statistics are usually not interesting reading, but in order to appreciate the size and importance of this district, it is logical to compare the production of southeast Missouri with certain other lead-producing districts. The following figures are taken from *Mineral Industry*, 1915, and show the production of lead by different nations for the year 1913—the latest complete figures available:

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[1] *Trans.* (1914), **48**, 33.

THE WORLD'S LEAD PRODUCTION

	Tons
Australia	116,000
Austria	22,312
Belgium	35,750
Canada	17,089
France	28,000
Germany*	181,100
Greece	18,309
Hungary	1,790
Italy	21,674
Japan	3,600
Mexico	55,530
Russia	1,000
Spain	203,000
Sweden	1,235
United Kingdom (domestic ore 1912)	17,706
United States	396,034
Southeast Missouri**	147,558

* This figure includes 83,781 tons imported.
** Private communication from C. E. Siebenthal.

Note that the production of southeast Missouri is eclipsed by one foreign country—Spain—and that southeast Missouri alone produced more than Australia.

For further comparison, consider the large lead-producing districts of the United States. These figures are for 1915, and are refinery statistics taken from *Mineral Industry*, 1915.

Southeast Missouri, as usual, was in the lead with a production of 183,906 tons[2]. Idaho was next with 160,680 tons, followed by Utah with 106,105 tons and Colorado with a production of 32,352 tons. Southeast Missouri alone produced more lead than the State of Idaho.

Southeast Missouri—by which term is generally meant the Flat River district—is one little known to the average engineer. This is due to the fact that there are only large companies operating in the district and that they have no reason for advertising themselves. Consequently, few engineers visit the Flat River district. When mention is made of lead mines in Missouri, the average person thinks of Joplin, because southeast Missouri, to many people, appears to be an unknown quantity. Joplin is situated on the other side of the State and the lead production of Joplin does not compare with that of the Lead Belt, as Table 1 clearly shows:

[2] Private communication from C. E. Siebenthal.

TABLE 1.—*Lead Production of Flat River and Joplin Districts*

Year	Flat River District, Tons	Joplin District, Tons
1913	147,558	28,558
1914	168,439	24,173
1915	183,906*	26,534

* Private communication from C. E. Siebenthal.

The disseminated lead district of southeast Missouri is the largest pure lead camp in the world, the profit from mining being due solely to lead, silver not being of commercial importance. This one district alone produces 33 per cent. of the entire lead output of the United States.

Operating Companies in Southeast Missouri

Southeast Missouri contains two distinct lead-producing districts: the Flat River or "Lead Belt," and another district which lies 25 miles southeast of the Lead Belt in the neighborhood of Fredericktown, in Madison County. The Lead Belt is much the larger and more important district, the daily tonnage milled being approximately 20,000 tons, while the mills in the Fredericktown region are handling about 2000 to 3000 tons daily, although the capacity in the near future will probably be greatly increased. A local map of the Lead Belt is shown in Fig. 2.

In the Lead Belt the following companies are operating: St. Joseph Lead Co., Federal Lead Co., St. Louis Smelting & Refining Co., Desloge Consolidated Lead Co., Baker Lead Co. and the Boston-Elvins Lead Co.

In 1913, the St. Joseph Lead Co. absorbed the Doe Run Lead Co., which operated its property at Rivermines. The St. Joseph Lead Co. has a 2400-ton mill at Bonne Terre, a 2000-ton mill at Leadwood and a 4200-ton mill at Rivermines. This company is the largest operator in the district.

The Federal Lead Co., which is affiliated with the American Smelting & Refining Co., operates two mills—the No. 3 mill at Flat River, which has a capacity of 5000 tons daily, and the No. 4 mill at Elvins, with a capacity of 3000 tons daily.

The St. Louis Smelting & Refining Co., locally known as the "National," because of its affiliation with the National Lead Co., operates a 2500-ton mill at St. Francois. This plant is being redesigned to handle 4000 tons daily.

The Desloge Consolidated Lead Co. operates one mill of 1500 tons capacity at Desloge.

The Baker Lead Co., at Leadwood, and the Boston-Elvins Lead Co., at Elvins, operate no mills, but ship their ore for treatment to the St. Louis Smelting & Refining Co.'s plant.

In the Fredericktown district, there are three lead-producing companies. The Federal Lead Co., near Mine La Motte station, operates

the Catherine property, which has a capacity of 600 tons daily. The Missouri Cobalt Co., at Fredericktown, operates the plant formerly known as the North American Lead Co. The old plant has been largely scrapped and is being replaced by a modern one which will produce lead,

FIG. 2.—LOCAL MAP OF THE LEAD BELT, SHOWING LOCATION OF PLANTS OF OPERATING COMPANIES.

∘ = Shafts; S = St. Joseph Lead Co.; F = Federal Lead Co.; DR = Doe Run Lead Co.; N = St. Louis S. & R. Co.; D = Desloge Consolidated Lead Co.; B = Baker Lead Co.; BE = Boston-Elvins Lead Co.

copper, cobalt and nickel. The mill capacity will probably be 300 tons daily. This plant is not yet operating. The Missouri Metals Corporation is operating the classic old Mine La Motte property, at Mine La

Motte. The property at present is producing lead and copper, but in the near future cobalt and nickel will also be produced. To the south of Fredericktown, the Einstein mine is being operated as a tungsten property.

The Fredericktown district is served by the Belmont Branch of the St. Louis, Iron Mountain & Southern R. R., which operates from Bismarck, on the main line, to Cape Girardeau and Belmont on the Mississippi River.

The galena-dolomite ores of the Catherine and Mine La Motte properties are similar to those of the Lead Belt and are given similar treatment. The copper-nickel-cobalt-iron-lead ores of the Missouri Cobalt Co. and at Mine La Motte demand a very different treatment. The proper method of handling these complex ores has not as yet been satisfactorily evolved.

The discussion in this paper will be limited to the concentration practice as related to treatment of the galena-dolomite ore of southeast Missouri, with special reference to the practice of the Lead Belt. The practice of treating the galena-dolomite ore of the Fredericktown district is similar to that used in the Lead Belt. Occasional reference, however, will be made to practice in the Fredericktown district which is radically different or which calls for comment. The methods used for treating the other ores of the Fredericktown district will be touched upon in a later portion of the paper.

History of the District

There is much of historic interest in reference to the lead deposits of southeast Missouri. M. de la Cadillac, or Lamotte, as he signed his name, arrived in Mobile Bay in 1713, with a commission as Governor of Louisiana. In 1715, he embarked upon an expedition into Illinois in search of silver, and it is thought that it was on this trip that he discovered the lead deposits at the headwaters of the St. Francois River, which even to this date bear his name—Mine La Motte. The date of the discovery is not definitely agreed upon by historians, but the time was about 1715 or 1720. In 1717, the charter of the "Company of the West" was registered in France, being established by the celebrated adventurer John Law.[3] This company, doubtless, controlled Mine La Motte—the story of which company has been so well recorded in *The Mississippi Bubble.* From that date, mining in this region has been almost continuous, but the work done in the early days was largely limited to surface diggings. During the Civil War, the furnaces at Mine La Motte were destroyed by the United States troops, but were soon afterward rebuilt, and from that date the work has been prosecuted on an increasing scale.

The year 1867 marks the turning point in the history of southeast

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[3] Pulsifer: *Notes for a History of Lead,* 92, 93.

Missouri. Previous to that date, mining had been practiced on a small scale, being limited to gophering in the surface deposits. In 1867, Charles Parsons came to Missouri and began making history for the region. At that time he occupied the position of superintendent for the St. Joseph Lead Co., and acted in either that capacity or that of manager until his death in 1910. He was actively engaged in the district for 43 years.[4] When Parsons came to the district, the yearly production of metallic lead was 100 tons. He secured the only diamond drill in the country—one imported from France—and began drilling. The drill was not a great mechanical success, but it did permit of developing the ground, at least to a depth of 200 ft. However, even after developing the disseminated ore, the company had no money for mill construction and the nearest railroad was 10 miles away. This meant that he had to freight in and out all supplies over poor roads, at the same time trying to develop a low-grade property. But he refused to acknowledge failure, continued to develop the property, and later succeeded in erecting a small mill where he began experimenting to determine the best means of concentrating the ore. This work was started about 1870, hand jigs, as well as all other types and kinds of machinery, being used. About 1880, the plant then operating was treating 200 tons a day in a hodge-podge mill, which has been aptly referred to as a "museum of ore dressing." H. A. Wheeler informs me that this was the beginning of the first actual milling in the district. The pay roll was frequently 6 months in arrears, and the property existed in this fashion for several years. In 1880, however, with the help of the Desloge Lead Co., a narrow-gage railroad 13 miles long was laid to connect with the Iron Mountain at Summit. The laying of the railroad was the beginning of success, and hundreds of teams, which formerly had been engaged in freighting, were laid off. Progress was rapid and the milling problem was rapidly worked out. The 200-ton test mill was burned in 1883 and a new mill was erected at Bonne Terre in 4 months, which building is yet standing. This was the first large mill in the district.

This mill had many ingenious and noteworthy features; all the concentration machinery was on one floor, there was plenty of light and the arrangement of the machinery was very simple, being patterned after Joplin practice, but the Parsons instead of the Cooley jig was used. An improved side-bump Rittinger table was installed which gave excellent results, but had only a small capacity. This mill had a rated capacity of 500 tons in 24 hr., and was really the beginning of large-scale milling in the Lead Belt. It was operated for 15 years without any decided changes except in length of building in order to increase the capacity. The capacity was finally increased to 800 tons.

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[4] Charles B. Parsons: *Engineering & Mining Journal* (Feb. 5, 1910), **89**, 309.

The flow sheet in use in this mill was described by H. S. Munroe.[5]

For the year ending May 1, 1887, the ore yielded 5.65 per cent. lead, the loss in the tailing was 2.13 per cent.—or 27.4 per cent.—of the total, and the loss was largely due to slimed galena. This work may be considered very satisfactory and compares favorably with the results obtained from the modern mill before the introduction of flotation. The cost of milling was 36.4 c. per ton, made up as follows: Labor, 13.4 c.; repairs, 10 c.; supplies, 3.5 c. and coal, 9.5 c., making a total of 36.4 c. The results obtained in this mill are interesting, both as regards the costs attained and the metallurgical recovery, but of course the machinery was not crowded as is now the case, thus permitting much better operating conditions.

This same building is now standing and operating at Bonne Terre, but the flow sheet has been much changed. One of the first changes made was to replace the crusher feeders—12 men per shift—by a belt feeder. Later, the 12 small crushers were replaced by a central crushing plant using gyratory crushers, but the Cornish rolls still remain and give excellent results as regards cost. After the death of Parsons, the mill was much changed, the Parsons jigs being replaced by Hancocks and the Rittinger tables by Wilfleys. This plant is now treating 2400 tons daily.

About 1875 a mill was erected at Mine La Motte, which used the German system of concentration, and was a most excellent mill in many respects, and where the metallurgical work was very satisfactory. The ore was elevated to the crusher which was situated at the top of the mill. The ore then passed by gravity to the rolls and then to the line of screens. The oversize of 15 mm. was returned to the rolls, the undersize was sized on 10-mm., 7½-mm., 5-mm., 3-mm., 2-mm. and 1-mm. screens, each oversize being treated on a separate Harz jig. The undersize of 1 mm. was classified and treated on Evans convex buddles which were used as roughers, the concentrate being cleaned on Frue vanners. This mill was noted for its close sizing, typical of German practice. I am informed by H. A. Wheeler that the feed to this plant averaged from 12 to 18 per cent. lead, and that the tailing contained from 2 to 2½ per cent. lead, and that the mill had a capacity of 250 tons in 24 hr. It operated until about 1902, when it was torn down and replaced by two succeeding mills operating with the Joplin type of treatment, using the Cooley jigs. These mills were soon abandoned, and in 1907 a mill was erected by the Traylor Engineering & Mfg. Co. for the owners. This plant had a rated capacity of 500 tons in 24 hr. It is still operating, but the flow sheet has been greatly changed, the rolls being the only original piece of machinery left in the mill, which is now treating from 1200 to 1500 tons daily.

The Desloge mill at Bonne Terre was built about 1880 and had a

[5] *Trans.* (1888–89), **17**, 659.

capacity of 200 tons. It used the German sizing system. This plant was burned about 1888 and was never rebuilt. This mill was described by John M. Desloge. The ore yielded by concentration about 7 per cent. of non-argentiferous galena and 1 per cent. of cobalt-nickel-bearing pyrite. The concentrate contained 70 per cent. lead, while the so-called "sulphide" contained equal parts of galena, dolomite and pyrite.

About 1878 or 1880, a 100-ton mill was erected at the Einstein mine at the "Silver Mines" south of Fredericktown. This was a well-designed plant and used the German system of close sizing. Harz jigs and convex buddles were used, the mill giving good metallurgical results. Trouble with the mine caused the plant to shut down after about a year's operation.

The Doe Run Lead Co. built a mill at Doe Run in 1889. It was an exact duplicate of the Parsons mill at Bonne Terre, the capacity being 1000 tons per 24 hr. It operated up to about 7 years ago.

The Donnelly, Leadington and Central mills in the Lead Belt were constructed about 1894. Each of the two former ones had a capacity of 100 tons. They were built by Fraser & Chalmers and were standard "shop" mills. The Central had a capacity of from 200 to 300 tons, but in 1896 it was treating 600 tons—at least that tonnage could be put through the mill—with excellent results, the operating costs being about 24 c. per ton. This mill is fully described in Richards' *Ore Dressing*, Vol. IV. The Central was one of the famous old-time mills of the district.

About 1897, the Desloge Consolidated Lead Co. erected a 500-ton mill at Desloge. This mill has been changed considerably, and is now handling from 1500 to 2000 tons daily. It is of wooden construction and still retains the original Cornish rolls.

An 800-ton mill was built by the Federal Lead Co. about 1900. At this plant the ore was crushed through 3 mm. and treated on Bartlett tables, no jigs being used in the original plans. Later jigs were installed and initial concentration took place at a coarser size. The plant has not been in operation for several years, having been recently dismantled.

Fraser & Chalmers, in 1900, built a 250-ton mill for the Columbia Lead Co. It had three jig sizes and originally had convex Evans buddles, but later they were replaced by Frue vanners and reciprocating tables. This mill ran intermittently for 6 years until purchased by Doe Run Lead Co., which company ran it a year before abandoning it. About the same time, the Catherine mill was built near Mine La Motte, in the Fredericktown district. The Catherine is still operating, treating about 600 tons a day. A Joplin-type mill was built 2 years later at the Rattlesnake

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[6] John M. Desloge: In discussion of paper by H. E. Armitage on Concentration of Low-grade Ores. *Trans.* (1889), **18**, 262.

Diggings at Mine La Motte. It operated for a couple of years and was then abandoned.

The Hoffman mill of the St. Joseph Lead Co. was erected about 13 years ago at Leadwood. This mill is of wooden construction. It is still operating, and has a capacity of approximately 2000 tons daily.

The St. Louis Smelting & Refining Co. put their mill into operation in 1900. This mill was the first one in the district of really modern construction, being built of steel, and was at that time the largest one in the district. It was designed to handle 1200 tons a day and used 175 Harz jigs and 24 circular slime tables.[7] In the crushing plant, seven jaw crushers, fed by hand, were used. They were soon replaced by gyratories. This plant operated along approved lines until 1907, when it installed the first Hancock jig ever used in the district. At this time the jig middling was ground in Huntington mills, the sand being treated on tables, the slime on vanners. This mill has, of course, been much changed —the Huntington mills and vanners having long since been discarded. Hancocks are still operating and the mill is handling 2500 tons daily.

A steel and concrete mill, plant No. 3, was built by the Federal Lead Co. in 1907. This was the largest mill in the district at that time, having a capacity of 2600 tons daily. It had a separate outside crushing plant where the ore was crushed dry to pass 8 mm. It then passed through a sampling plant to the mill storage bins. This arrangement was a big advance over anything practiced in the district. Three screen sizes were used, the oversize being treated on Harz jigs while the undersize of the 2-mm. screen was classified, some spigots being jigged, others being tabled. Soon afterward the Harz were replaced by Hancock jigs. This change greatly simplified the flow sheet and increased the capacity from 2600 to 4000 tons a day. The new flow sheet is shown in Fig. 3. This plant is still operating, treating 5000 tons a day, and it still maintains the distinction of being the largest mill in the district.

The steel-concrete mill of the Doe Run Lead Co. is situated at Rivermines. This mill, which began operating in 1910, followed the usual practice of the district of that day and installed Hancock jigs and tables. A description of this mill was published by A. H. Fay[8] and a later description has been given by C. T. Rice.[9] The flow sheet in use in the mill at the time it was constructed is shown in Fig. 4. This mill is still operating, having a capacity of 4200 tons a day.

The most recently constructed mill in the Lead Belt is the No. 4 mill

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[7] Lead Mines of Southeastern Missouri: *Engineering & Mining Journal* (July 28, 1900), **70**, 92.

[8] Operations of the Doe Run Lead Co.: *Engineering & Mining Journal* (March 19, 1910), **89**, 610.

[9] Milling in Southeastern Missouri: *Engineering & Mining Journal* (June 28, 1913), **95**, 1283.

of the Federal Lead Co. This plant is the last word in mill design in the district, the construction being entirely of steel and concrete. The mill has a capacity of 3000 tons and will be described in more detail later in the article.

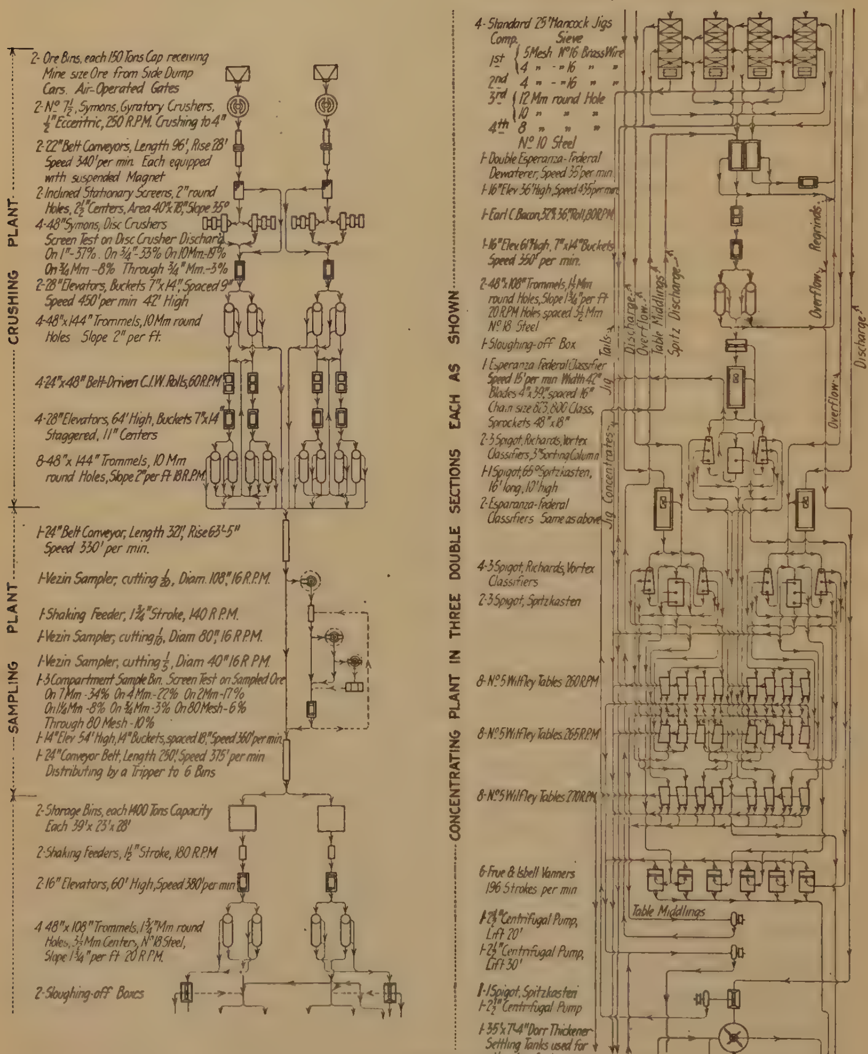


FIG. 3.—FLOW SHEET USED IN NO. 3 MILL OF THE FEDERAL LEAD CO. AFTER REPLACING HARZ BY HANCOCK JIGS.

As previously pointed out, there are seven mills now operating in the field, the practice in all of them being similar so far as principle is concerned, the flow sheets differing only in details.

Present Method of Treating the Ore

The usual method of treating the ore is as follows: It is given an initial crushing in gyratory breakers, later finished in rolls to pass the first screen, which is 9- or 10-mm. round-hole punched plate. Water is first

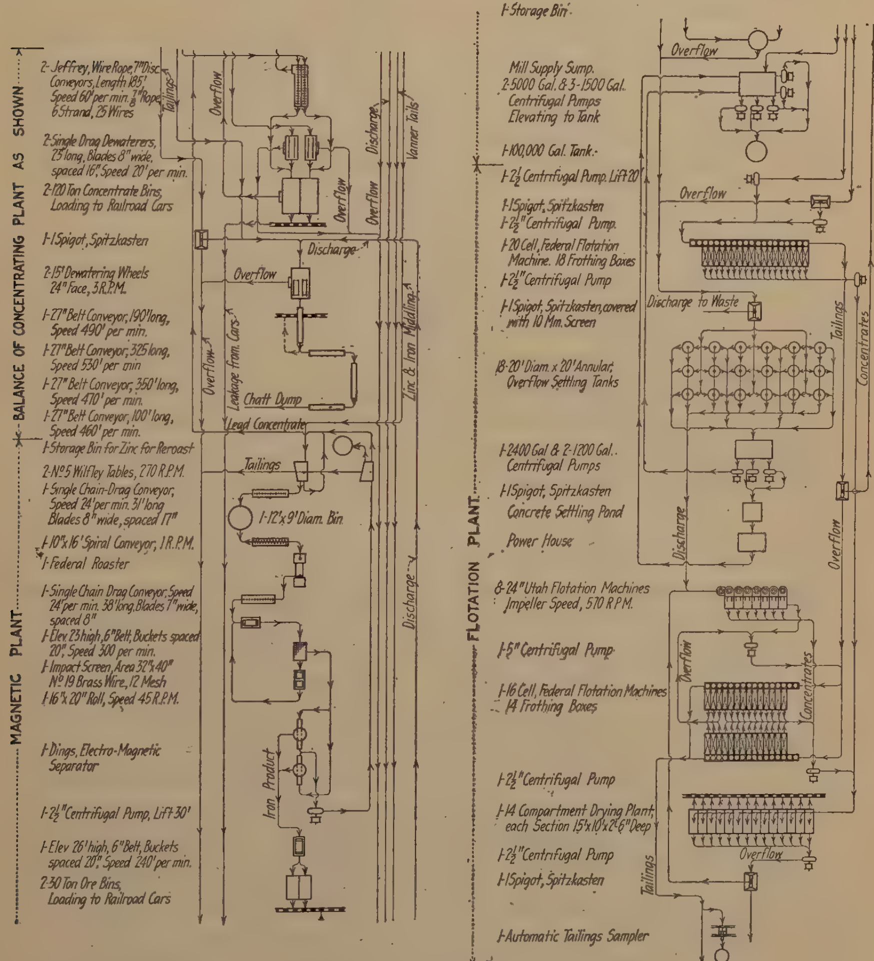


Fig. 3.—Continued.

added to the undersize of the first limiting screen, the pulp passing to the second limiting screen. This size varies to some extent, the usual size, however, being from 1½ to 2 mm. Punched plate with round holes is used entirely on the second limiting screen, woven wire not being used in the district for screening.

The oversize of the 2-mm. screen passes to Hancock jigs which yield concentrate, middling and tailing. The concentrate is either a finished

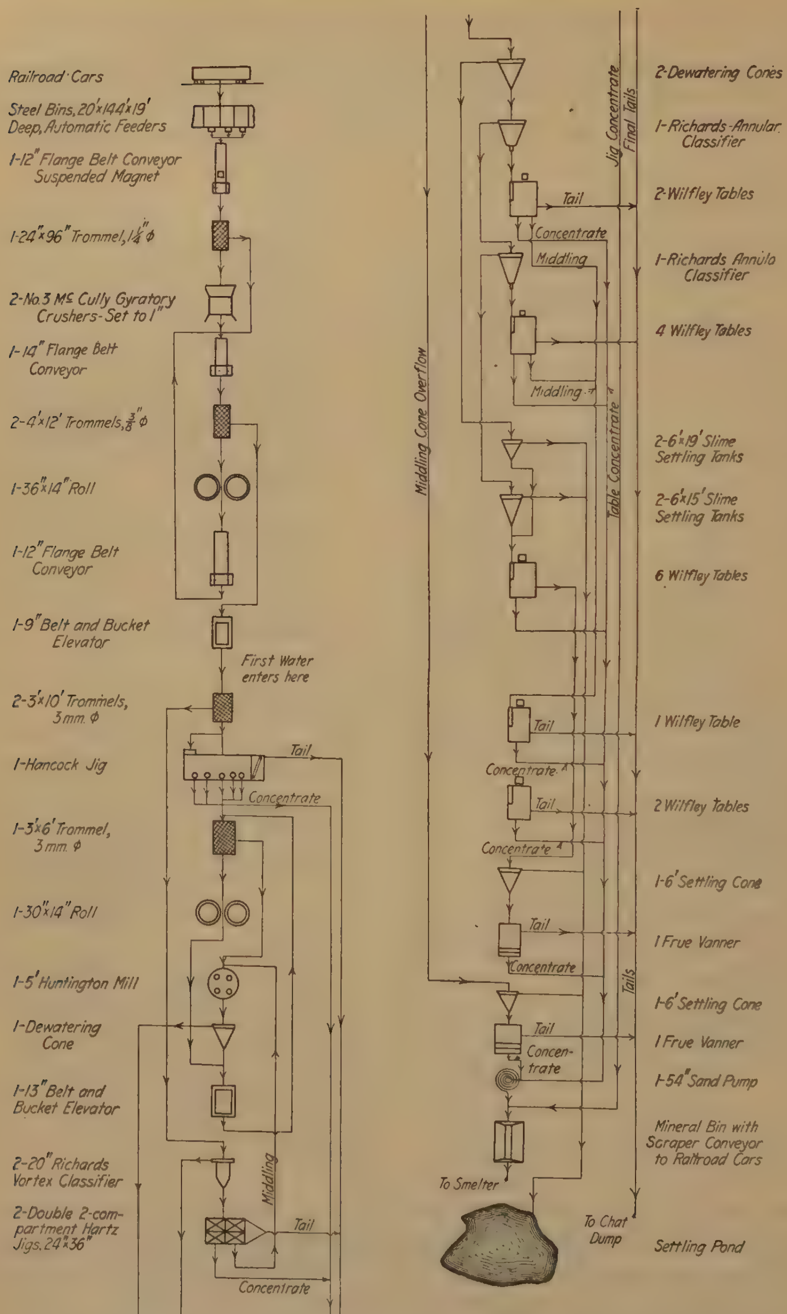


FIG. 4—FLOW SHEET IN USE IN 1910 IN THE ELVINS MILL OF THE DOE RUN LEAD CO.

product or else it passes to a re-treating plant. The tailing, in general, is an end product, although one plant re-treats the general tailing before stacking. The middling is generally crushed in high-speed rolls, but much work is being done in the district in an endeavor to replace them with regrinding mills; Allis-Chalmers, Hardinge, Marathon and Marcy mills being now tested for the purpose of middling regrinding. It is desirable to crush the jig middling through a 1½- or 2-mm. trommel. Formerly the reground middling was joined with the original feed, but at present the usual practice is to keep them separate. The discharge from the middling rolls, or regrinding mills, is sent to the middling elevator, which discharges to the middling screen. This is generally a trommel covered with 1½- or 2-mm. punched plate, the oversize of which may be sent to a middling jig or to the jig treating the original feed.

There are two undersize products to treat on the tables—the original undersize and the middling undersize. They may be combined, but generally they are treated separately and the latter course is the logical one to follow. The original undersize is comparatively small in amount, but contains much galena and little water. The contrary is true of the middling undersize: here the tonnage is large but of low grade and this product contains all of the circulating water in the middling system.

The method of treatment given each of the two products smaller than 2 mm. is practically identical. The product may be classified, each spigot going to tables, or it may be deslimed in any of a large number of desliming machines. In this latter case, which is the more common, the deslimed spigot is treated on Butchart tables. In any event, whether the table feed is classified or simply deslimed, the sand tables make a final tailing, a concentrate that may be final or re-treated and a middling. This middling is circulated, retabled or reground and generally joins the main stream of middling at the middling elevator. The table middling contains considerable pyrite which builds up in the system, and this "iron," as it is locally called, is now being ground through 80 mesh in ball mills and tabled, the tailing going to flotation.

The overflow of the desliming apparatus or the classifiers may pass directly to the flotation plant, or else be first sent to the slime-table department for treatment before going to flotation. If the latter course be taken, the overflow of the desliming cones is settled, the spigot is tabled, yielding a concentrate and a tailing. The concentrate is a finished product and the tailing is combined with the overflow of the settling tanks in the slime-table department and sent to flotation.

At the flotation plant the slime is settled in Dorr tanks, the overflow being sent back to the mill-water system. The spigot product, thickened to 20 per cent. solids, goes to the flotation machines, of which both agitation and pneumatic types are used. Creosote is the usual frothing agent employed. Theflotation concentrate is not re-treated, while the tailing

generally receives further treatment. Re-treatment of the tailing generally consists of sending the primary tailing to pneumatic machines, which make a final tailing and a concentrate that may be final or may be cleaned later. The general flotation concentrate is settled in Dorr tanks, the spigot product is treated by Oliver filters, the cake from which contains 50 per cent. lead and 14 per cent. moisture. It is loaded into box cars and shipped to the smelter, but separate from the jig and table concentrate.

The general mill tailing is stacked by conveyor belts, as the topography of the ground does not permit of disposal of the tailing by gravity alone. The tailing from the flotation plant is either pumped to the chat pile to be used as flushing water, or else sent to a slime-settling pond. For operating and metallurgical reasons, the mill concentrate is kept separate and distinct from the flotation concentrate.

With this preliminary statement regarding the milling practice, we are now in a position to discuss its different phases in detail, but before proceeding with the methods used for treatment of the disseminated lead ores of the district, it will be well to first make a study of the ore to be treated, in order that we may understand the reason for some of the apparently radical steps employed in the district.

Character of Ore Treated

The ore treated in the district is galena in dolomite; the galena being practically free of silver, the ore containing but 0.05 to 0.10 oz. per ton. The dolomite, which is light gray in color, has a specific gravity of 2.8 to 2.85. The ore contains on an average from 4 to 6 per cent. lead. Much ore, however, containing less than 4 per cent. lead is milled, and the average ore contains much nearer 4 per cent. than it does 6 per cent. of lead, but the more fortunate companies have 6 per cent. ore to treat. Some pyrite is present in the ore, as well as some chalcopyrite and sphalerite. The iron in some cases may be considerably less than 1 per cent., and in some cases it may be more than the lead. Although the zinc is generally small in amount, ores from certain parts of the district may contain as high as 2 per cent. of it, the silver appearing to follow the zinc. In general, lead is the only economical mineral recovered, although two mills are attempting to save a zinc concentrate. Small amounts of cobalt and nickel are present in the ore, as well as copper. Some of the iron middling contains as high as 5 per cent. copper and has been shipped, but the cobalt and nickel are not of economic value. A little silica is present in the ore, but not enough to cause undue wear on the crushing machinery. Chlorite and calcite are also present, but the ore for all practical purposes may be said to be galena in dolomite, with some pyrite present as an accessory mineral.

Chemical analyses are given in Table 2 of two samples of ore: Sample "A" is considered good ore in the district, while sample "B" would be considered very rich.

TABLE 2.—*Chemical Analyses of the Disseminated Lead Ore of Southeast Missouri*

	Sample "A," Per Cent.		Sample "B," Per Cent.
Lead	4.32		6.06
Silica	4.83		7.40
Iron oxide	6.64		6.10
Alumina	1.16		5.50
Lime	30.80		26.70
Magnesia	17.96		13.60
Sulphur	0.97		1.70
Zinc	0.50		Trace
Nickel and cobalt	Trace		Trace
Copper	0.03		0.50
Carbon dioxide by difference	32.79	By difference and undetermined	33.57
	100.00		

The ore is tough and compact and there are no cleavage planes in it, although there are a few shale seams. These, however, affect the crushing but little and are of no importance in the finer sizes. No weathering or oxidation has occurred. Not all the rock shot down in the mine is ore, and some can be rejected. Although the ore in general is disseminated, some of the rock is absolutely barren, while at the other extreme some high-grade material can be shipped direct. However, the amount of this rich ore is of minor importance and can be neglected. The bulk of the actual lead-bearing rock is probably limited to 40 per cent. of the total tonnage, and the presence of the large amount of barren dolomite permits of making a tailing at 10-mm. size.

The lead-bearing dolomite contains the lead in particles generally smaller than ¼ in. and much of it in particles smaller than ⅛ in., and much of the galena is present in smaller particles of all sizes, some being practically microscopic. On the other hand, there is much galena present in size much larger than ¼ in., but this is reduced in crushing, so the larger size of the mineral is of no special significance so far as the concentration problem is concerned.

In order that a better understanding may be had of the ore of the district, specimens have been polished and photographed. The photographs are actual size and show typical ore specimens, varying in assay from very low grade to rich ore. The reproductions are shown in Fig. 5–10. The galena is present as dark spots, the occasional white patches being due to reflected light from the cleavage faces of the galena.

Description of Polished Specimens

No. 5. This sample is of very low-grade ore. The lead is very finely disseminated and, if it were not freed during crushing, it is probable that much of it would be lost as included mineral in the jig tailing. There is much ore, however, that contains the lead

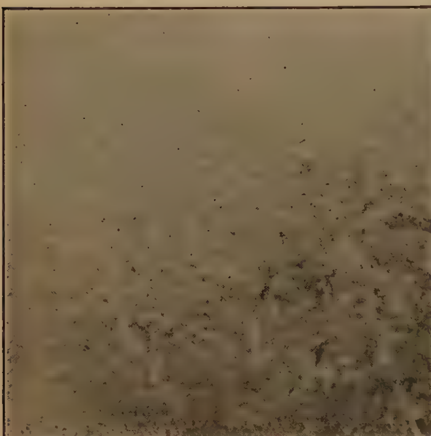


FIG. 5.

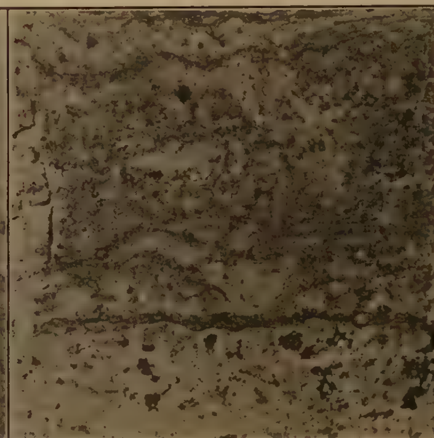


FIG. 6.

in even smaller particles than this specimen shows, closely approximating, in fact, the appearance of some of the Western copper "porphyries."

No. 6. This specimen shows slightly coarser crystallization of the lead, and small bands of shale and chlorite are noticeable.

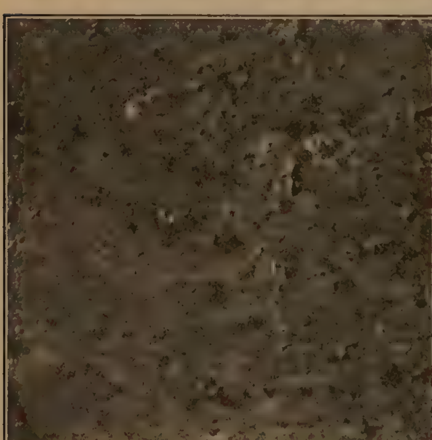


FIG. 7.

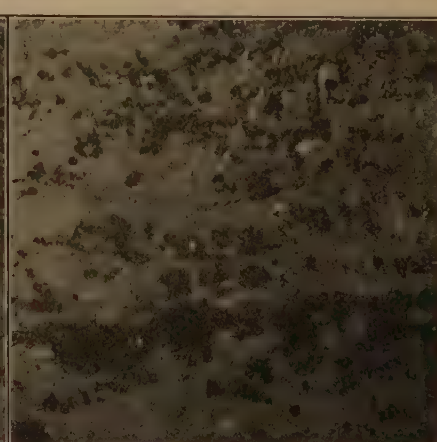


FIG. 8.

No. 7. Coarser crystallization and the porous character of this specimen is noticeable in this sample.

No. 8. Considerable silica and some pyrite is present in this specimen and the presence of thin bands of shale can be noted.

No. 9. In this specimen the galena is present in still larger patches, and the minute shale stringers are very noticeable.

No. 10. The galena is very coarse in this specimen. There is ore, however, which contains galena very much coarser, and as a limit the ore may contain as much as 50 per cent. lead or even be pure galena in pieces weighing several pounds. There is much ore milled that contains lead much coarser than any of the accompanying specimens show. The large coarse galena, however, breaks up during crushing.

Although some large, coarse galena occurs in the ore, bull jigging is not possible, as in general when 4 per cent. ore is crushed through 1¼-in. round hole, the first free lead of economic importance is about 9 mm. in size.

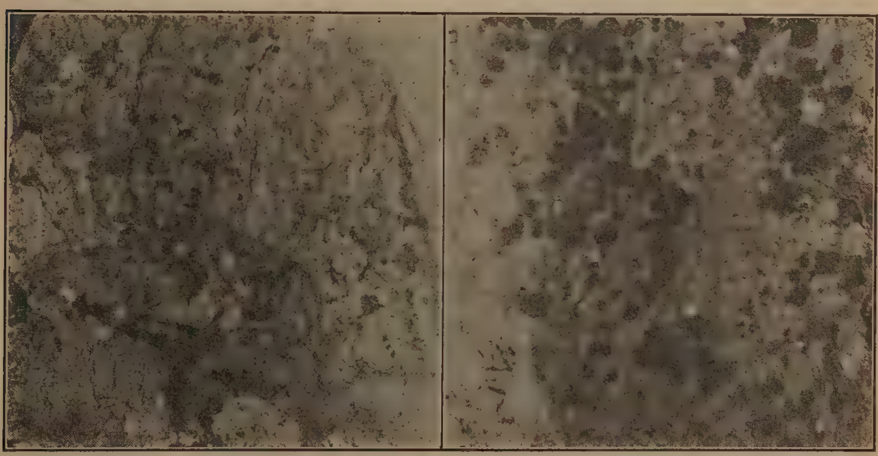


FIG. 9. FIG. 10.

In general, crushing through 3 mm. will liberate from 90 to 95 per cent. of the total lead, and crushing through 2 mm. will serve to free practically all of the lead that can be done economically. At 0.208 mm.—65 mesh—practically 100 per cent. of the lead is free. These figures show that extremely fine crushing is not necessary to free the lead in these ores. At 10 mm. a jig tailing can be made assaying 0.7 to 0.8 per cent. lead.

The Concentration Problem

The concentration problem of the district is to save the galena and discard the other metallic minerals. The concentrate must be as clean as possible, not only to save smelting charges, but also to permit the smelters to handle efficiently the concentrate, particularly if the Scotch hearth be used. The galena should be recovered in a size as large as possible.

At one or two plants in the district, hand-sorting of waste is carried on underground, but in general neither hand-picking of waste or ore is

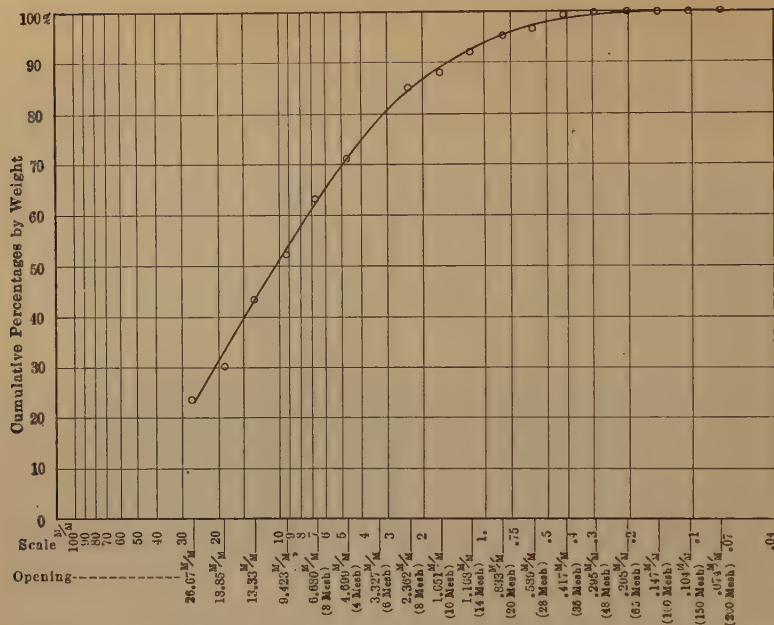


FIG. 11.—LEAD FREE IN 3.81 PER CENT. LEAD ORE WHEN CRUSHED THROUGH CERTAIN SIZED APERTURES.

Figs. 11 and 12 show the amount of lead freed when typical ore is crushed to pass certain sizes. Fig. 11 shows results for 3.8 per cent. ore and Fig. 12 shows results for 6.3 per cent. ore. It will be noted that extremely fine crushing is not required.

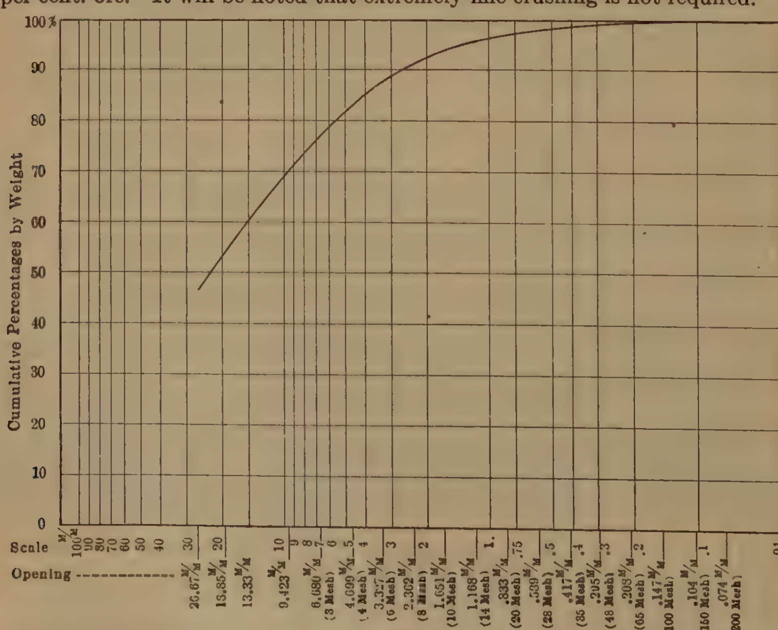


FIG. 12.—LEAD FREE IN 6.3 PER CENT. LEAD ORE WHEN CRUSHED THROUGH CERTAIN SIZED APERTURES.

practiced, on the surface or underground. A certain amount of sorting on the surface is done, but it is not vital. The exception is at Mine La Motte, in the Fredericktown district, where sorting at the surface is an important feature of the metallurgical scheme.

Hand-picking at Mine La Motte

At this famous old property, hand-picking is receiving much attention. The galena-dolomite ore comes to the surface, which permits of steam-shovel mining instead of the use of the usual underground methods. A picking plant is now being constructed at that property to handle the ore, which will be mined by 90-ton shovels. The ore will be loaded to side-dump, standard-gage cars, with air attachment for dumping. The cars will be brought to the picking plant where they will be automatically dumped on a rail grizzly spaced 8 in. The oversize will pass to a Traylor 60- by 84-in. jaw crusher, the product of which will join the undersize of the grizzly and be screened in a 6 by 22-ft. trommel. The trommel will be covered with a punched plate with 4-in. opening, surrounded by a second screen with 1½-in. holes. The oversize of 4 in. will pass to one set of storage bins and the oversize of 1½ in. will pass to a second set of bins. The oversize of 4 in. will be fed by steel-apron feeders to rubber picking belts, 40 in. wide, traveling 30 ft. per minute, from which ore will be picked, the waste passing to a tail-belt for stacking. The ore through 4 in., but on 1½ in., will be fed to another set of rubber picking belts, waste being picked instead of ore. The ore from both systems of picking belts will be crushed to 3 in. and any waste then picked out, while the resulting ore will pass to the mill where it will receive treatment similar to that in the Lead Belt, possibly preceded by bull jigging.

As this plant is now being constructed, no figures regarding tonnages or results are available. The plant is designed to treat 8000 tons in 24 hr., the capacity probably to be increased in the near future.

At first sight, it might appear strange that hand-picking on such an extensive scale should be practiced at this property, while the subject receives so little attention in the Lead Belt. The different character of the ore at this property accounts for it. The ore at Mine La Motte is not disseminated as it is in the Lead Belt. The ore is galena in dolomite and carries as accessory minerals those of iron, copper, nickel and cobalt. The dolomite is shattered and ore deposition has taken place along the fracture planes. It is extremely rare to find galena disseminated through the dolomite. If the dolomite shows no galena on the surface of a specimen, it is extremely rare that any is found within the rock. The galena appears to be deposited solely along the fracture planes and small water courses, and consequently much of the lead is freed in crushing, and much clean dolomite exposed that can be picked out as waste. Physically,

hand-picking is logical, as the character of the ore warrants such a step. The economics of the problem, however, presents another question, which doubtless has been solved by the management.

Crushing

No crushing is done underground in the district, all being done at the mills, with the exception of a few cases where coarse-crushing is done at certain shafts, the crushed ore then being transported to the mill. The usual practice in the district is to centralize the crushing plant at the mill and, as would be expected, the use of the gyratory type of crushers is almost universal, although one of the older plants still retains the jaw type.

A typical installation would be as follows: The run-of-mine ore is fed from the crude-ore bin to the primary gyratory crushers. They are usually either No. 6 or No. 7½ of the standard makes. The ore is generally fed direct without preliminary screening of the fine. The feed may be regulated by arc gates operated by air or by the usual type of air-operated gates. The use of steel-pan conveyors in the capacity of crusher feeders is becoming common practice, as with a slow-moving pan conveyor a very large gate opening can be permitted, thus preventing choke-ups due to large rocks. The pan conveyors prove very satisfactory for this purpose, being operated either by an adjustable ratchet and pawl or by a variable-speed motor. The discharge of the crushers passes to trommels, the oversize in general passing to the fine-crushing gyratories. At one plant, however, the discharge from the first gyratory passes, after screening, to 48-in. Symons horizontal disk crushers. At another plant, the discharge from the first gyratory crushers, after screening, passes to Symons vertical disk crushers, which are capable of making a reduction from 3 in. to approximately ½ in. This arrangement, of course, simplifies the crushing and screening plant.

There is nothing very startling about the coarse-crushing practice of the district, since it operates along standard lines. There are, however, two exceptions. One is the coarse-crushing plant of the St. Louis Smelting & Refining Co., the other is at the property of the Missouri Metals Corporation, at Mine La Motte, already mentioned.

At the National plant a departure has been made from the usual method of coarse crushing, a jaw crusher taking the place of the gyratories for initial crushing. This crusher, which was manufactured by the Traylor Engineering & Manufacturing Co., has a jaw opening 30 by 60 in. and it is being used as a "rougher" for the present installation of gyratory crushers.

A word of explanation may be necessary in order to understand the reason for installing this type of crusher. The present milling plant was

erected 17 years ago, designed to treat 1200 tons in 24 hr. The mill is now handling 2500 tons and it is desirable to handle 4000 tons daily. This increase in capacity is possible with the present mill building, because of the advance in the art and the recently designed flow sheet. The original crushing plant, however, is not capable of handling 4000 tons and it was not considered advisable to add a duplicate crushing unit, as the present No. 6 gyratories cannot efficiently handle the present size of ore hoisted. The present crushing plant consists of two No. 6 crushers, followed by four No. 3 short-head gyratories. The No. 6 crushers cannot handle the present size of ore without arching of the large pieces of rock. Furthermore, the ore was formerly transferred 300 ft. from the shaft to the crushing department by means of a transfer car, the ore being hoisted in 1-ton cars that were carried by the transfer car to the crushing plant. At the time of the original installation such design was logical and feasible, but with the increased tonnage and the ever-increasing wage scale, it became desirable and necessary to replace this arrangement by mechanical methods that would operate without the need of men; consequently, 5-ton skips were installed, which are hoisted by an automatic, electric Nordberg hoist, the ore then being transferred to the crushing department on a steel-pan conveyor. This eliminated the transferring of each individual ore car and made possible the installation of a crusher to handle any size of rock that could be hoisted. The jaw crusher was installed as a rougher in order to handle the large rock and to increase the capacity of the present installation. A gyratory having an opening equal to that of the jaw crushers would have had capacity out of all proportion to the balance of the plant. The jaw type was decided on because of the large opening it afforded in comparison with its capacity. The use of this primary jaw crusher will permit less powder to be used underground, and the sledging of large pieces of ore is entirely eliminated both underground and on top, since anything can be crushed that can be loaded and hoisted. The ore is fed by variable-speed steel-pan conveyors to a pocket, from which it is discharged to the crusher by a rotary grizzly set at 4 in. The oversize passes to the jaw crusher set at 4 in. and the crusher product joins the grizzly undersize and is elevated in a steel elevator to the present crushing plant. Here the undersize of 1¼ in. is screened out, the oversize passing to the No. 6 crushers. The jaw crusher makes 180 r.p.m. and requires from 60 to 100 h.p. This crushing unit has a capacity of 5000 tons a day.

The usual practice in the district, however, is to use gyratory crushers as primary machines, followed by short-head gyratory crushers, Symons crushers or rolls.

The Federal Lead Co. at its No. 3 mill crushes in two No. 7½ Symons gyratories, the discharge after screening passing to four Symons horizontal disk crushers. The discharge is finished in rolls to pass 10 mm. This

plant is the only one in the district to reduce the ore to 10 mm. in an outside crushing plant. The arrangement is highly commendable. At the Federal No. 4 mill the crushing is further simplified: The run-of-mine ore is fed by the steel-pan feeders to two Symons No. 7½ gyratories. The product is elevated and screened, the oversize passing to four Symons fine-reduction disk crushers. The resulting product is reduced to approximately ½ in. and is finished in the mill proper to pass 12 mm.

The usual practice is to crush to approximately 1½ or 2 in. in the outside crushing plant and to make the final reduction to 10 mm. by rolls in the mill building itself. This arrangement is not to be recommended, as any roll trouble, etc., causes a loss of feed. The better method is to crush to the limiting size—10 mm.—in an outside crushing plant and store the crushed ore in fine-ore bins in the concentration department. The ore being dolomite, without the presence of sticky material, no trouble is experienced with the ore "hanging up" in the fine bins.

Roll Practice in the District

There is nothing of particular interest about roll practice in the district. The usual type is the high-speed roll, although two mills are still operating the Cornish type. The high-speed rolls are of the standard makes, the largest rolls in the district being 54 in. in diameter. Rolls that crush sizes larger than 10 mm.—or the upper limiting screen—are operated dry, while rolls crushing material finer than the first limiting screen are operated wet. The usual practice for dry-crushing is as follows: The ore is crushed to approximately 1½ or 2 in. in the crushing plant, then conveyed to and stored in the mill bins. From here, after screening over 10 mm., the oversize is fed to the rolls, the roll product passing to the elevator raising the original roll feed. In other words, the elevator, screen and rolls are in closed circuit, the rolls crushing from 1½ or 2 in. to 10 mm. Graded crushing is not used in the Lead Belt, although graded roll crushing is practiced at one plant in the Fredericktown district, all roll practice at that property being wet, due to mill conditions. Graded roll crushing gave much better operating conditions than were formerly obtained by the use of a closed circuit, principally because the roll spacing could be better adjusted.

Both manganese and rolled-steel roll shells are used. Manganese-steel roll shells—when dressing is required—are ground in a lathe, while the rolled-steel shells are turned in a lathe—integral with the shaft and hearts. Some plants transfer the shells to the machine shop for turning in lathes while other companies turn the shells in the mill building. The plant operating the 54 by 20-in. rolls had no lathe sufficiently large to handle such work, so there was built an arrangement at the roll house by means of which rolls could be turned without the use of a lathe.

The time of turning a 54- by 20-in. shell is from 20 to 30 hr., but the time, of course, depends upon the extent of the corrugations. Formerly, before installing this arrangement, it was necessary to ship these shells to a plant operating a lathe sufficiently large to swing them. The cost of turning under these conditions was $13 per shell, which did not include the cost of labor for removing the shells from the hearts. At present, with the arrangement mentioned above, the cost per shell is from $5 to $8.

Screening Practice

All crushing and screening is done dry in sizes larger than 10 mm. There are many screening devices used in the district but the trommel is the usual type of machine used, and for heavy work it is usually supported on rollers at the feed end, the discharge end being supplied with a gudgeon. For 10-mm. screening, the usual type of shaft and spider trommel is used. The old type of conical trommel, as well as the hexagonal trommel, is used at one plant, but both of these types, as well as the cylindrical trommels, are being replaced by the flat screens of the Ferraris type. These are to be found at two mills where they have replaced or are replacing the cylindrical trommels. The flat screen has many advantages over a trommel, since the screening efficiency is higher and the wear on the punched plate is much less than that needed for a trommel, and, for the space occupied, the flat screen has a much larger capacity than a trommel. The flat screen will also handle a greater volume of water than will a trommel of the same screening capacity. The impact screen, as manufactured by the Colorado Iron Works Co., is being tested at one mill in the district. Flat stationary screens are used following the coarse crushers in one plant, and one plant is using a rotary grizzly for removing the undersize of 4 in. before feeding the original ore to the primary crusher. As the finest screening in the district is done on the second limiting screen, corresponding to a 2-mm. trommel, it will be seen that the screening problem is not very serious. About 80 per cent. of the screening is being done with the standard trommel, and the balance of the tonnage mainly on the Ferraris type of flat screens.

Jigging the Ore

The ore of the district is ideal for jigging and, in consequence, jigging is a subject of considerable importance. This subject will be discussed in detail in another paper, so it will be passed over lightly in this article.

The ore when crushed through 10 mm. readily yields commercial tailing, as at this size it contains much dolomite that is free of lead. This

clean dolomite, however, gradually progresses to the condition of middling where the galena is finely disseminated. When crushed through 10 mm., the first free lead of any great importance occurs on 4.699-mm. screen. The jig tailing will contain from 0.7 to 0.8 per cent. lead. The subject of jigging will be considered under two headings, bull jigging and treatment on the Hancocks.

Bull Jigging.—There is no bull jigging practiced at present in the Lead Belt, as the disseminated character of the ores does not permit of such a step.

At Mine La Motte, however, the character of the ore is such that coarse jigging is possible, as the ore at this property has the galena

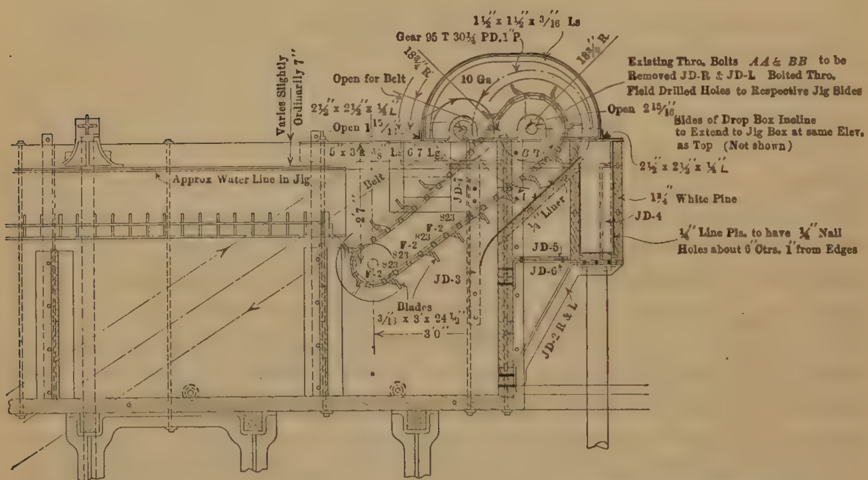
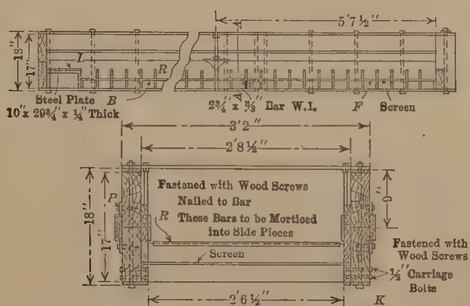


FIG. 13.—DETAILS OF CHAT DRAG FOR STANDARD 25-FT. HANCOCK JIG.

present, not in disseminated particles, but deposited to a very large extent in cracks and minute fissures. During crushing much of this lead is freed, leaving the gangue comparatively clean. With this ore it is possible to make a Hancock-jig tailing containing but half the lead that a similar tailing does in the Lead Belt. This shows the ease with which the lead is freed in the Mine La Motte ore. Test work is now being done on this subject, and it appears that ore through 1 in. and on 10 mm. may yield a commercial tailing. The possibility of jigging this large size is a great advantage, as probably 30 to 40 per cent. of the original tonnage can be discarded at a large size.

Hancock-jig Practice.—The Hancock jig is one of the noteworthy things in the district, since jigging in the Lead Belt is now limited entirely to the use of the Hancock jig. There is one mill where a Cooley jig is

being used in a tailing plant, but with this exception the Hancock rules supreme in the field. This type of jig was first introduced into the district in 1907 by the St. Louis Smelting & Refining Co. Southeast Missouri is probably the largest user of the Hancock jig in the world, and at present there is no probability of replacing it. The use of finer grinding and roughing tables may be an entering wedge for displacing the Hancock, but at present very little thought is being given to the question.



SECTION A-A

Fig. 14.—Screen tray for standard 25-ft. Hancock jig.

The Hancock jig belongs to the type of jig having a movable screen. The screen tray rests in the tank, which is made of 4-in. lumber. The tank is 25 ft. long, 4 ft. 2 in. wide and 10 ft. high. It is divided by five headers into six compartments, the headers being placed at any desired position. The products from the first two compartments are concentrate, the third spigot may yield a concentrate or a product for circulation depending on operating conditions. The fourth and fifth hutches yield middling for crushing, while the tailing is discharged over the end of the tray into the last compartment. A chain drag of the Esperanza type is usually built into the last compartment, which drags the tailing up over the end of the jigs. Details of the chat drag are shown in Fig. 13.

The tray, as built, is 20 ft. 6 in. long and 3 ft. 2 in. wide, outside dimensions, built of lumber, as shown in Fig. 12. One company has replaced the wooden tray by one of steel. A steel tray has many advantages over one of wood, being lighter, and due to the increase in effective screen width, the capacity of the jig is increased about 15 per cent.

The feed to the Hancock is generally crushed through 10 mm. on 1½- or 2-mm. screens. Very efficient work is done on this range of sizes. In Table 3 is shown a screen analysis of a typical jig feed through 10 mm. on 2 mm.

TABLE 3.—*Screen Analysis of Hancock Feed Through* 10 *mm. on* 2 *mm.*

Screen Opening in Millimeters	Mesh	Weight in Grams	Per Cent. of Total Weight	Cumulative Per Cent. of Weight	Per Cent. Lead	Per Cent. of Total Lead	Cumulative Per Cent. of Lead
6.680	3	294	5.7	5.7	1.06	3.0	3.0
4.699	4	1,116	21.6	2.73	1.12	11.5	14.5
3.327	6	1,331	25.9	53.2	1.50	18.7	33.2
2.362	8	952	18.5	71.7	2.10	18.7	51.9
1.651	10	479	9.3	81.0	2.30	10.2	62.1
1.168	14	399	7.8	88.8	3.50	13.1	75.2
0.833	20	214	4.2	93.0	3.80	7.7	82.9
0.589	28	136	2.7	95.7	4.20	5.4	88.3
0.417	35	58	1.1	96.8	5.00	2.6	90.9
0.295	48	27	0.5	97.3	6.40	1.5	92.4
0.208	65	13	0.3	97.6	6.80	0.9	93.3
0.147	100	13	0.3	97.9	7.40	1.1	94.4
0.104	150	11	0.2	98.1	7.60	0.7	95.1
0.074	200	8	0.2	98.3	6.00	0.6	95.7
Pass 0.074	−200	90	1.7	100.0	5.20	4.3	100.0

It may appear strange that the Hancock jig finds such extensive use in southeast Missouri, whereas in the Joplin feed the Cooley jig is the prominent feature of every mill. The Cooley has been tried out in the Lead Belt where it competed against the Hancock. The Hancock gave the better results, the water and power consumption being less and the extraction better than with the Cooley. On a moderate tonnage, the Cooley will yield as good, but no better, results than the Hancock, but when an attempt is made to increase the tonnage, the loss in the Cooley is very great. For equal tonnages handled, the Hancock makes about one-third more middling than the Cooley, both producing the same assay in the tailing. The water consumption of the Hancock was 100 gal. per min. less than with the Cooley, and the Hancock required 6 hp. while the Cooley required 14 hp.

Classification of the Ores

In the past, this district has made use of classifiers for preparing table feed, the usual size treated being the undersize of 1½ mm. Nearly every type of standard classifier has been used, as well as the usual flock of home-made ones. The introduction of the Butchart table has done much to simplify the question of classification, and in practically every plant the classifiers have been either entirely displaced or largely replaced by apparatus for desliming only, so that the classification problem is not very great, being confined practically to a desliming operation.

In the past, use has been made of classifiers for preparing Harz jig feed, and two plants for a while endeavored to use classifiers for preparing

the feed to the Hancocks. This arrangement was not noted for its satisfactory results, and was replaced by screens for preparing the Hancock feed.

The present practice may be discussed under three separate headings —classification, desliming, settling. For all practical purposes, we can say that classification—in the usually accepted sense of the term—is not used in the district, preparation of table feed being done by desliming apparatus.

Much use is made of desliming apparatus for preparing the table feed. The problem is not very serious as, due to the large margin in difference of specific gravity of the gangue and galena, extremely efficient work is not of great importance. The desliming apparatus should eliminate from the spigot product the galena too fine to be saved on the tables, and the overflow should contain a minimum of sands and lead that can be efficiently treated on the tables. This size is usually set at 150 mesh.

TABLE 4.—*Typical Screen Analysis of Product for Desliming*

Screen Opening in Millimeters	Mesh	Per Cent. of Total Weight	Cumulative Per Cent. Weight	Per Cent. Lead	Per Cent. of Total Lead	Cumulative Per Cent. Lead
1.651	10	0.1	0.1	.....		
1.168	14	2.8	2.9	5.7	2.2	2.2
0.833	20	5.8	8.7	5.6	4.3	6.5
0.589	28	8.7	17.4	6.6	7.5	14.0
0.417	35	8.0	25.4	7.4	7.8	21.8
0.295	48	8.0	33.4	8.2	8.7	30.5
0.298	65	6.1	39.5	9.2	7.4	37.9
0.147	100	8.1	47.6	10.5	11.2	49.1
0.104	150	7.5	55.1	10.0	9.9	59.0
0.074	200	5.6	60.7	8.0	6.7	65.7
Pass 0.074	−200	39.3	....	6.6	34.3	

Original assay 7.56 per cent. lead.
This product is the undersize of 2-mm. trommel.

At present, 2-mm. undersize is the largest size being deslimed. Work has been carried on with undersize of 3-mm. trommel with satisfactory results so far as classification is concerned.

Desliming is accomplished by means of Dorr classifiers, drag classifiers, Akin classifiers and desliming cones. The Dorr and drag classifiers, in some cases, send the sand product direct to tables. In other cases, the sand is treated first in classifiers, or desliming cones, before tabling. At one plant the undersize of the 2-mm. screen is sent to a desliming cone, the spigot of which discharges to a belt drag, which in turn yields a sand product for tabling. Other plants deslime the screen undersize in desliming cones, the spigot product going direct to tables. The Delano cone is used at several plants and is giving excellent results.

This subject of desliming will be treated in more detail in another paper, and in order to avoid duplication the subject will not be discussed further.

The invention of the Dorr tank and the discovery of flotation have entirely changed the status of things as regards settling. The Dorr tank has practically displaced all other settling devices, and the reasons for such a change are so apparent that the subject requires no discussion. The subject of Dorr tanks is discussed in more detail under flotation. A few of the mills settle part of the slime in slime tanks in the mill, but the general practice is to send all slime to Dorr tanks situated outside the mill building.

The methods used for settling slime in the mill for preliminary treatment before sending it to flotation are not sufficiently noteworthy to warrant discussion.

Table Treatment

The problem of table treatment is simple and, due to the great difference in specific gravity between the galena (7.4) and dolomite (2.85), close classification is not essential. In the past, the usual method employed was to classify the feed and treat it on Wilfley tables, the usual size treated being the undersize of 1½-mm. trommel, but at present the table feed is deslimed and generally treated on Butchart tables. The usual size feed to the tables is the undersize of 1½- or 2-mm. screens.

A screen analysis of a typical deslimed table feed is shown in Table 5. The product is the undersize of a 2-mm. trommel.

Table 5.—*Screen Analysis of Typical Feed to Butchart Tables*

Screen Opening in Millimeters	Mesh	Per Cent. of Total Weight	Cumulative Per Cent. Weight
2.362	8	Trace	
1.651	10	0.1	0.1
1.168	14	4.5	4.6
0.833	20	9.8	14.4
0.589	28	15.3	29.7
0.417	35	14.7	44.4
0.295	48	14.0	58.4
0.208	65	10.3	68.7
0.147	100	13.5	82.2
0.104	150	9.4	91.6
0.074	200	3.6	95.2
Pass 0.074	−200	4.8	100.0

The table concentrate will average about 77 per cent. lead, while the tailing will contain about 0.3 per cent. lead. The standard tables are in use in the district.

Probably the largest size original feed to be tabled is the undersize of 3 mm. This size yielded good results by tabling, but the wear of the riffles and of the linoleum was so great as apparently to offset the advantages gained by tabling this size. A Butchart table was used for the test, and with 90 tons of feed per table, the tailing assayed between 0.4 and 0.5 per cent. lead. The object of endeavoring to table this coarse feed was to remove the fine sizes from the jigs and, furthermore, to remove from the jigs as much of the galena as possible, thus reducing the loss due to abrasion. The excessive wear of the table tops necessitated replacing the 3-mm. screen with a 2-mm. plate.

The table covering in general use is linoleum, the life of which may be from 6 months to 3 years, depending on the size of feed. One plant is experimenting with the use of rubber to replace linoleum, and discarded vanner belts are now used to some extent for this purpose. In one case, the belts have lasted 19 months and are still in good condition. One plant is using a high-grade rubber for table covering, but as the cost of such a cover is from three to four times the cost of linoleum, its advantage is doubtful.

The concrete deck is being experimented with and apparently will give excellent satisfaction. The first concrete deck in the district was installed by the St. Joseph Lead Co. at its Bonne Terre mill.

The method used for putting concrete decks on Butchart tables is as follows: The deck is first covered with linoleum or canvas and then marked for riffling. The riffling nailing strips are then tacked on the deck, of the thickness of the concrete that is to be used. These strips are not tapered, but are the same thickness for their entire length, the usual thickness used when the feed is to be 1½ or 2 mm. being from 3/16 to ¼ in. For coarser feed, a thicker nailing strip is used. The deck is given no special preparation, except to see that it is free from oil and grease. After the nailing strips are in place, the concrete is applied; this is composed of 2 parts of sand screened through 2 mm. to 1 part of cement. The concrete after setting is given a steel-trowel finish, the finishing being done parallel to the nailing strips. The decks thus made give excellent results, do not crack, and in no way give trouble. After the concrete has set, the riffles are nailed to the nailing strips. The riffles may be of pine or oak, but oak is commonly used, as the life of such riffles is three times the life of those made of pine.

The number of tables to each Hancock jig varies to a great extent. At one plant there are supplied three tables for each jig, at another plant there are ten tables for each jig. This ratio varies according to screen sizes, methods of middling, grinding, etc. The average ratio is one jig to seven tables.

In the past it was customary to combine the undersize of the original 2-mm. screen with the undersize of the middling screen. Fortunately,

this illogical arrangement of things has about disappeared. The general practice now is to treat the original feed and the product resulting from regrinding of the middling on separate tables. The middling is of a different character from the original ore and demands a different form of treatment. With a separate middling-table department, this is possible. Furthermore, there are many other self-evident advantages in this arrangement of keeping the middling and original feed separate.

Recrushing of Middling

This subject is receiving a great deal of attention in the district. Two types of middlings are being recrushed, the jig middling and the table middling. The problem of crushing the jig middling will be first considered.

In the past, the jig middling has generally been crushed in rolls. At present, one company is using Chilean mills for the crushing of the fifth hutch of the Hancocks, and one company has just discontinued the use of Huntington mills. Rolls are used in all plants for crushing middling and are exclusively used for this purpose in five plants. Fine-grinding mills are being introduced at all the plants in order to determine the best mill adapted to this problem. The Huntington and Chilean mills are not desirable for many obvious reasons. The result obtained by crushing the middlings in rolls is not satisfactory, and every attempt is being made to replace the rolls and other crushing machines with one of the newer types of fine-grinding mills.

The first company to introduce a ball mill into the district for crushing of jig middling was the Desloge Consolidated Lead Co., which began operating a 6-ft. by 22-in. Hardinge mill in March, 1916. Since then many different types of mills have been introduced, and fortunately the choice of the different companies was not the same, so at present the following types of recrushing mills are operating: Allis-Chalmers, Hardinge, Marathon and Marcy mills. An Allis-Chalmers ball granulator is operating in competition with a 6-ft. by 22-in. Hardinge at one plant and at another plant an 8-ft. by 30-in. Hardinge is running in competition with a 4 by 9-ft. Marathon. The Marcy mill as yet is not directly competing with any other mill, as is true of the 4 by 10-ft. Marathon.

The following fine-grinding mills are now being tested for the crushing of jig middlings: Two Allis-Chalmers ball granulators, 6 by 4 ft.; three Hardinge mills, one 8 ft. by 30 in., two 6 ft. by 22 in.; two Marathon mills, one 4 by 9 ft., one 4 by 10 ft.; one Marcy mill, 8 by 6 ft. Fortunately, the ore of the different companies is very similar, so that rather conclusive results may be obtained regarding the comparative efficiency of the different types of mills, even though they be not operating side by side on the same feed. It is realized that the only proper manner in which to compare results of two mills is to operate them side by side and

split the feed to them. However, where the ore from different parts of the district is so similar, as is true in the Lead Belt, the results obtained by comparing the work of different mills operating at different plants is indicative of the true efficiency of the different types of mills. The ore from different localities of the district is similar and although certain mines may show a slightly softer or harder ore than others, yet at the end of a month's operation these minor inequalities are eliminated. The problem of recrushing the jig middling in this district is peculiar, in that extremely fine crushing is not necessary, nor logical. It is a well-known fact that, aside from the physical character of the product to be crushed, the size to which crushing should take place depends on the assay of the feed and the market price of the economic metal contained in the product. Assuming 4-c. lead, it is a self-evident fact that fine-grinding of a middling cannot be carried to any great extent.

The feed to the fine-grinding mills is generally from the fourth and fifth hutches of the Hancock jigs, but one plant is crushing the jig middling and the oversize of the 10-mm. screen in the ball mill. The usual practice, however, is to take as feed to the recrushing mill, the jig middling which is through 10 mm. and on 2 mm. The fifth hutch of the jig will contain from 1.5 to 2.5 per cent. lead, and the fourth hutch may assay from 8 to 20 per cent. lead, and it is generally desirable to crush the middling to pass 1½ or 2-mm. trommel. The product should be similar, if possible, to that produced by roll crushing, as the minimum of slime is desired. The feed to the recrushing mills will assay from 3 to 8 per cent. lead, and after crushing through 1½ or 2 mm., an economic tailing and a clean concentrate can be produced and a middling for recrushing in a separate circuit. By economic tailing is meant a product that cannot be further treated and yield a product. Test work has shown that where a jig middling is crushed through a 0.208 mm. aperture—65 mesh—practically 100 per cent. of the lead is free. It is thus evident that extremely fine grinding is not necessary nor desirable. The ideal product would be one that would just pass the limiting screen, 1½ mm. or other size, and have a minimum of material finer than 65 mesh. If a tailing can be made at 1½ mm., it is evident that it is not desirable to produce any larger tonnage below this critical size than is necessary. At the other extreme, no product is desired finer than the lower critical point at which all the lead is freed.

It makes no particular difference whether we accept as correct Rittinger's law or Kick's law, the power required to produce the fine sizes is very great. If 100 per cent. of the lead be free at 0.208 mm., or any other certain size that test work may indicate, then it is self-evident that no advantage is gained by crushing finer than this size. We thus have two well defined limits, between which we desire to distribute the largest possible part of the ground product. At the upper limit a tailing

can be produced that will yield no profit by further treatment. At the lower limit, all the lead is free and no advantage can be gained by finer crushing. The problem of this district, then, from a metallurgical standpoint, is to obtain a mill that will produce a maximum percentage of a product between certain defined limits.

We should not lose sight of the fact that a regrinding mill is operated primarily for making money. The mechanical efficiency of the mill is of secondary importance. One mill might yield a discharge, all of which would just pass the limiting screen, and this product when tabled would yield an economic tailing. With a second mill, the discharge might contain a great deal of oversize and a great deal of slime, neither product being desirable. But in this second case, the apparent mechanical efficiency might be higher than that of the first mill considered. It does not follow that the second mill is the more efficient mill for the purpose. The first mill produced a maximum tonnage of coarse sizes that would yield on tabling an economic tailing and a minimum tonnage of product to be treated by flotation with its many disadvantages. In the second case, although calculation might indicate that the apparent efficiency was higher than was true of the first mill, economically such is not the case, as the oversize would not produce a clean tailing and would need to be reground, while the slime would demand treatment by flotation. What must be considered is *economic recovery* and not *metallurgical recovery.* The primary object of regrinding is to grind only to that size where a tailing can be made that will not pay for further treatment. When that size is reached, further crushing should be avoided, for although the metallurgical recovery is being increased by finer grinding, yet this increased metal recovery is being made at an actual loss, since there is being ground a product that will not pay for recrushing. It would not be considered good policy to take a clean table tailing and send it to a regrinding mill, for further grinding, but if a mill in crushing forms a large tonnage of discharge below the critical size of crushing, this step is actually being taken. Consequently, what is demanded for jig-middling regrinding is a mill that will give a discharge in which the maximum possible tonnage of product is just below the size of the limiting screen that will yield an economic tailing. Every particle crushed beyond the critical size is done only at an economic loss. Relative mechanical efficiency and metallurgical recovery are of no particular importance—what is desired is a maximum economic recovery.

For instance, with 4-c. lead, it would not pay to grind a product containing 0.4 per cent. lead. Regrinding mill "A" might free no lead on such a product, while mill "B" might be so operated as to yield a high metallurgical recovery—both mills would be operated at a loss, but mill "B" would be making the greater loss to the company, even though the recovery in metal were higher.

A second point in importance that should receive attention in comparing the work of recrushing mills is the distribution and character of the economic mineral in the discharge. Considering the problem of southeastern Missouri, it is desirable to have the galena in the discharge as coarse as possible, as the large sizes are more desirable than the smaller sizes at the smelter—principally due to dust losses in the latter. Furthermore, fine sizes of galena signify an undue amount of "slime," which cannot be saved on tables, but demands treatment by flotation. Regrinding mill "A" may show a lower grinding efficiency than does mill "B," but further investigation may show that the latter mill has much of the galena finer than 200 mesh. Such being the case, flotation, with its high operating cost, will be necessary to save the mineral. One should not be short-sighted in comparing the work of recrushing mills; the comparison is not complete until we compare the *ultimate* cost of operating each mill.

In this district the most efficient recrushing mill, considered from an economic standpoint, which is the only logical method of comparison, is one that will treat the largest possible tonnage per kilowatt-hour and produce a discharge, the maximum possible tonnage of which will just pass the limiting screen. The discharge should have a minimum of fine sand and the portion through 200 mesh should also be reduced to a minimum. The galena in the −200-mesh portion should be of granular form so that a maximum amount can be recovered on slime tables—thus reducing to a minimum the lead to be saved by flotation.

I would consider a mill fulfilling the above conditions to be the most efficient for the purpose of middling regrinding in this district.

The problem is the same at each plant in the district and each company is choosing the machine it believes best suited for the purpose.

It may be well to digress for a moment and consider the results obtained with the Marathon mill. This type of mill is comparatively new and it may be well to treat it in some detail.

The principle of the Marathon mill is different from that of a ball mill. The crushing in the Marathon is done on a line contact, while with the ball mill it is done on a point contact. The Marathon works on a positive principle; a ball mill works on a non-positive principle.

Due to the fact that cylinders, and not spheres, are used as the crushing media, a Marathon will contain in the same volume 50 per cent. greater weight of metal than a ball mill. A ball mill to contain weight of metal equal to that of a Marathon, requires to be either 50 per cent. longer or else of greater diameter. As a ball mill is not efficient in long sizes, recourse would be taken to a ball mill of greater diameter. But such a step demands more power, as the center of gravity of the mass of balls is farther from the center of the mill than is the center of gravity of the rods in a Marathon.

These two points constitute the two principal advantages of the Marathon over a ball mill: the line contact gives positive crushing, while the substitution of rods for balls permits a smaller horsepower to be used for equal weight of crushing metal. Assuming that crushing is proportional to weight of crushing medium, the Marathon is more efficient on this score as well as operating on a more efficient crushing principle.

Description of Marathon Mill Used for Tests.—Results are available from tests made on a 4-ft. by 10-ft. mill which was driven by a 50-hp. direct-current motor, using a belt drive. The feed to it was jig middling through 10 mm. on 2 mm., which was dewatered by an Esperanza-Federal drag. The feed contained less than 1 per cent. of material finer than 100 mesh and the moisture was less than 10 per cent., additional water being added at the mill. Provisions were made for obtaining an even feed for any tonnage up to 800 tons a day.

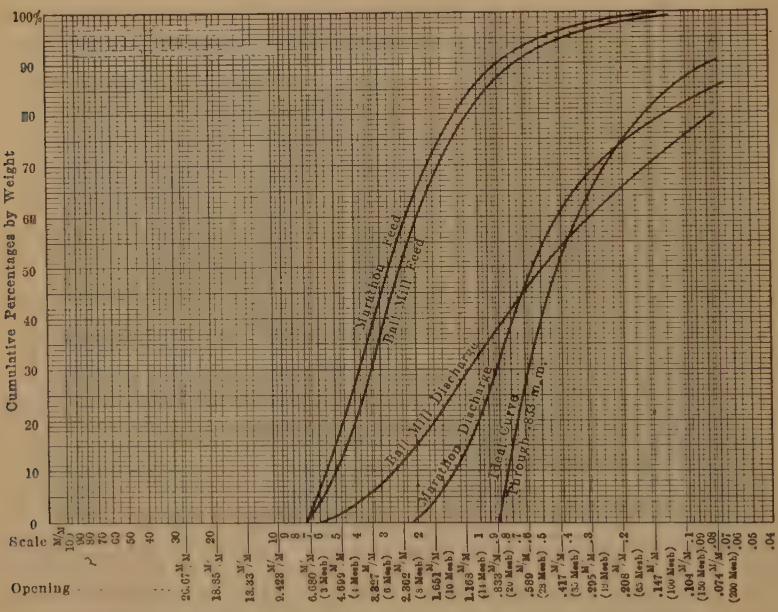


FIG. 15.—PLAT SHOWING COMPARISON OF DISCHARGE OF MARATHON MILL AND A BALL MILL.

The character of the work of the Marathon mill is so different from that of other regrinding mills that it will be of interest to discuss the results of tests obtained in preliminary test work. In these preliminary tests, the Marathon was operated with a central discharge head, the mill being horizontal, and the speed varied from 23 to 29 r.p.m. The amount of moisture in the feed varied from 40 to 50 per cent. The largest-size rods used were 1¼ in. in diameter, and the rod charge 18,000 lb.

Results of Preliminary Tests.—The Marathon mill gives a product containing a maximum amount of granular material in the sizes immediately below the limiting screen. The discharge never contains any oversize or tramp particles as is true of a ball-mill product. The discharge with a heavy tonnage, of course, will be coarser than with a small feed, but no tramp oversize will occur in it. The discharge also contains less fine sand than is found in a ball-mill discharge. These facts will be brought out more forcibly by a study of the accompanying screen analyses and plats.

TABLE 6.—*Screen Analyses of Typical Feed and Discharge of a Marathon Mill*

Feed to Marathon Mill				Discharge of Marathon Mill	
Screen Opening in Millimeters	Mesh	Per Cent. of Total Weight	Cumulative Per Cent. Weight	Per Cent. of Total Weight	Cumulative Per Cent. Weight
6.680	3	3.9	3.9		
4.699	4	11.1	15.0		
3.327	6	17.7	32.7		
2.362	8	18.9	51.6	0.2	0.2
1.651	10	12.7	64.3	2.7	2.9
1.168	14	14.3	78.6	10.7	13.6
0.833	20	9.3	87.6	15.8	29.4
0.589	28	6.0	93.9	18.2	47.6
0.417	35	2.9	96.8	12.0	59.6
0.295	48	1.5	98.3	8.1	67.7
0.208	65	0.8	99.1	4.9	72.6
0.147	100	0.5	99.6	5.2	77.8
0.104	150	...	....	4.1	81.9
0.074	200	0.4	....	2.9	84.8
−0.074	−200	...	....	15.2	

The mill in this test was adjusted for crushing through 1.651 mm. Note the small amount of material on this screen and the large amount of product immediately below the size of the limiting screen. Furthermore, the small amount of fine sand is a very noticeable feature and of particular interest, as no product is desired finer than 0.208 mm. The two screen analyses are plotted to show more forcibly the character of a typical Marathon discharge (see Fig. 15). The feed to the Marathon was a roll discharge—note the practically parallel curves of the feed and product. The small amount of material larger than the limiting screen, when examined, was found to be in tabular form. No original uncrushed particles were present, all particles in the feed having been crushed. The material larger than the limiting screen was not true oversize, but rather those particles of a scale-like nature, one dimension being so small as to prevent crushing.

Fig. 15 shows the comparison between the product of a Marathon mill and of a ball mill, each consuming the same amount of power. The feed to the ball mill was practically the same as to the Marathon, but while the ball mill was handling 250 tons, the Marathon was treating 340 tons in 24 hr. There is a very noticeable difference in the characteristics of these discharges. The ball mill gives a large amount of oversize and a much larger amount of fine sand than the Marathon. The ball mill also makes more slime. If an attempt were made with a ball mill to reduce the amount of fine sand, the amount of oversize would increase and *vice versa.* The curve of the Marathon discharge is nearly parallel to the ideal crushing curve. Note how the ball mill crosses the ideal curve. The Marathon yields a much superior product for concentration.

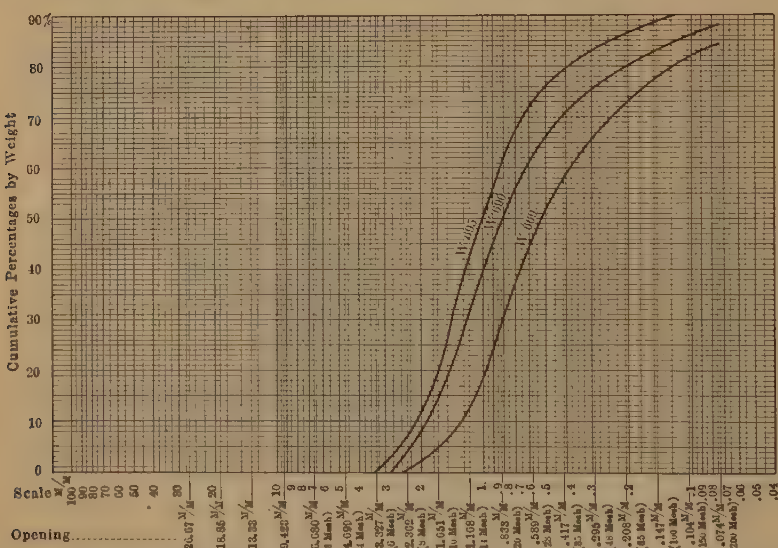


Fig. 16 —Plat showing that increased rate of feed to a Marathon does not result in producing oversize in the discharge.

The fact that increasing the tonnage of feed to a Marathon does not result in production of oversize in the discharge is made evident by Fig. 16. The product resulting from an increased tonnage of feed is uniformly coarser than with a smaller feed and an increased tonnage reduces the slime as would be expected. Note that the three curves are practically parallel, although in curve W 669, the amount was 310 tons, in W 690 the amount was 375 tons and W 695 shows the discharge when 500 tons was being treated. If a ball mill were yielding a normal discharge with 310 tons of feed, the oversize would be greatly increased if the tonnage were increased to 500 tons. Such is not the case with a Marathon mill. For this reason, a Marathon discharge can be sent direct to tables

if so desired—no guard screen is necessary for sizing the discharge. The Marathon mill is positive due to the principle on which it works.

Fig. 17 is presented as showing the similarity between the discharge of a Marathon mill and that from a set of rolls. The feed in both cases was the same. It is interesting to note that the curves are practically parallel, showing the similarity of the product of both machines. The Marathon was making only 2 per cent. more product through 200 mesh than the rolls, but as is seen, the Marathon was making a greater reduction. When the test was made the rolls had just been equipped with new shells, but longer operation, of course, would result in yielding a much inferior product, due to increased wear and corrugations.

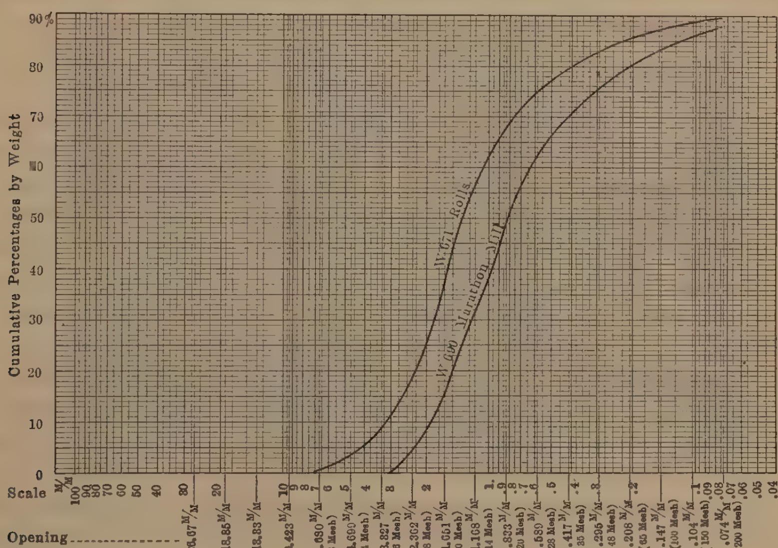


FIG. 17.—PLAT COMPARING THE CHARACTERISTICS OF THE DISCHARGE OF A MARATHON MILL AND A SET OF ROLLS.

In this test the feed was first crushed in the Marathon and then later the entire tonnage was sent to the middling rolls. As the rolls were driven from a line shaft, there was no way of determining the power for operating them, but as they were 42 by 20-in. rolls making 90 r.p.m. and set close with heavy spring pressure, I believe the power they required was at least as much, and possibly more, than required for the Marathon mill. The Marathon was not operating at its highest efficiency, but even under these conditions it showed over 60 per cent. greater efficiency than the rolls, assuming that they required equal power. The feed contained 6.680-mm. particles, and so did the roll discharge, but the largest size in the Marathon discharge was 2.362 mm. Work was done to determine the amount of lead freed by crushing in the rolls as com-

pared with the Marathon. Even though the latter was operating under conditions which were not the most efficient, the lead freed by it was 70 per cent. more than that freed by the rolls.

One reason that the Marathon mill was originally fitted with a feed hopper rather than with a scoop feed was in order that tests might be made to determine what results would be obtained when operating with a feed containing a low moisture content. Difficulty is generally encountered when using a scoop feed in this district if the water is reduced much below 50 per cent., and 40 per cent. appears to be about the lower limit of feeding. Even with this reduced water quantity the tonnage must be reduced. By use of the hopper feed, which consisted of a hopper measuring 15 in. by 15 in. on top and entering the mill on a gradual curve, a much thicker feed was introduced into the mill.

The specific gravity of the middling to be crushed is about 2.85. Consequently, a feed containing 26 per cent. moisture is composed, by volume, of equal parts of water and solid. A feed containing 41 per cent. moisture has 2 volumes of water to 1 of ore; 51 per cent. moisture, 3:1; and 58 per cent. moisture, 4:1. It was hoped that by decreasing the moisture in the feed, a higher crushing efficiency could be obtained, since the volume of feed entering the mill would be greatly decreased. It was thought that a thicker pulp would permit the pulp to remain for longer time in the mill, thus being under the influence of the crushing forces for a longer time. With this idea in mind, the Marathon mill was fed with an increasingly thicker pulp. Using the hopper feed, a point was soon reached where the mill would not take the thick feed, and a water jet was then introduced into the hopper, and later in order to avoid the use of water, an air jet under 80 lb. pressure was installed. By this means a feed containing but 20 per cent. moisture could be introduced. This amount of water was just about enough to fill the voids between the solids. The results obtained from using this thick pulp were unsatisfactory, the resulting product being much coarser than when a higher moisture content was in the feed. It is a false assumption to consider that an increased per cent. of water in the feed increases the rate of progress of the feed through the mill. The rate of travel of the solids is probably somewhat increased, but nothing in proportion to the water ratio. The increased volume of water rapidly passes through the mill, the solids lagging behind. Too thick a feed appears to cushion the rods, lowering, rather than increasing, the crushing efficiency. The crushing efficiency increases with the increase of moisture up to a critical point.

The Marathon mill can be equipped with two types of discharge heads, the central discharge and the rim discharge. The rim-discharge head has certain advantages and disadvantages, for although it allows a freer discharge of the pulp, on the other hand the rods at the discharge end of the mill are not as well cushioned with pulp as are those at the feed end,

and I would look for increased metal wear. All work was done with the central-discharge type.

One remarkable thing with the Marathon mill is the fact that the mineral is discharged in a much coarser size than is done with a ball mill, the galena in the Marathon discharge being in the form of perfect cubes. This results, of course, in less sliming of the valuable mineral.

The decreased sliming of the galena in the Marathon is due to the fact that the action of the Marathon when crushing is similar to that of a set of rolls. The rolls are held apart by means of the shims, thus preventing contact of the steel and consequently preventing excessive sliming. The same principle is in use in a Marathon mill, the ore holding the rods apart in the same manner as do the shims in the rolls. The smaller the tonnage of feed to the Marathon, the less the rods are forced apart, and the distance being less, the resulting product is finer than would be the case with a larger tonnage of feed. As the rods are held apart a certain definite distance and are not in metal contact, the valuable mineral is not slimed as would be the case in a ball mill, as there is no metal to metal contact. Another fact that is of particular interest is that there is a great difference in the condition of Marathon and ball-mill "slime." If a sample of ball-mill discharge be screened through 200 mesh and the same be done with a sample of Marathon discharge, a great difference will be noted in the character of each undersize or "slime" product. The Marathon slime is granular, and the galena is in much larger sizes than is true of the ball-mill slime. The galena in the ball-mill slime is microscopic in size, while in the Marathon slime the galena is much larger and the particles are distinct cubes. If these two products be examined under a microscope, this radical difference of physical character is very noticeable. The Marathon mill is noted for the granular discharge, and it is interesting to find that this granular condition continues even into the so-called "slime." This feature is of particular interest to one desiring fine crushing and a minimum of slime. This advantage is of great importance in the Lead Belt, especially where the desire is to minimize to the greatest extent the amount of lead sent to the flotation plant.

It is generally understood that the rods in a Marathon are parallel. This is not the case, as is evident by a little thought. When a mill is freshly filled with rods and before any ore is fed to it, the rods are parallel. However, as soon as a normal feed is given the mill, this statement no longer holds true. As the feed enters the mill the rods are gradually forced apart, the increase in volume being due to the volume of the ore. Soon a state of equilibrium is reached, and in this state the rods are not parallel to each other. Those at the feed end are farther apart than those at the discharge end of the mill. The reason for this is self-evident. The ore as it travels through the mill is gradually reduced in size, and as

a result some fine sand and slime form. Although some of these finer sizes are probably circulated through the rods a number of times, much of this fine material is carried directly out of the mill by the current of water in the feed. Then, as the ore progresses through the mill, more fine sand and slime are formed and continually washed out of the mill. This results in there being a smaller proportion of ore between the rods at the discharge end than at the feed end. The rods are consequently not forced as far apart as at the feed end and consequently they are not parallel. The angle between the rods, however, must be very small. The conditions under which the mill was operating show this to be but a small fraction of a degree.

The fact that the rods in the Marathon are not parallel to each other is not a disadvantage, but on the contrary, it is a most decided advantage and accounts for the fact of there being no oversize in the discharge. Fortunately, the rods are not parallel, if they were the Marathon would be lacking one of its cardinal points. The angle between the rods, while very slight, is of the very greatest importance. It accounts for the gradual reduction in sizes in the Marathon. If, for the sake of argument, we assume the rod space at the feed end to be normally 1/4 in. and at the discharge end 1/16 in. then if any particle larger than 1/4 in. be in the feed it will be crushed before the smaller sizes. If we assume it is not crushed, but advances a few feet in the mill, let us then consider conditions. The rod space has decreased to 1/8 in. and consequently the particle will be caused to pass through a roll space less than the diameter of the particle, and the latter will be crushed. At the feed end, the sizes smaller than the rod space are largely protected from crushing—the energy first being expended on the larger sizes. Tests appear to indicate that different sections of the mill operate on distinct sizes. There is no hit-or-miss principle in a Marathon. The first foot of length crushes the largest size of the feed to a definite size, the next foot makes a reduction to a second specified size; this gradual and consistent reduction taking place for the entire length. Without doubt, some of the finer particles are crushed at the feed end of the mill, but nothing in proportion to the action on the larger sizes. The action is *selective*, not *haphazard*. If two particles, say 1/4 in. and 1/16 in. in diameter, are introduced into a Marathon, we are very certain what is to happen to the 1/4-in. particle. As it progresses through the mill, it is constantly confined between rods, the space between which is gradually lessening. Finally it will reach a point where the rod space is less than the diameter of the particle, and it will be crushed. The 1/16-in. particle can travel alongside of the larger one without serious injury, as it is protected by the larger sizes which act as shims to hold the rods apart. Consequently, we see that the larger sizes are always crushed before the energy is consumed in useless crushing of sizes already fine enough or else too fine. With a Marathon, there is

no probability of oversize finding its way through the mill, any more than if a particle were to pass through more than a hundred set of rolls in series, the spacing of each being slightly less than that of the preceding set. Eventually the particle would reach a set of rolls it could not pass without being crushed. This same condition exists in a Marathon—positive reduction of all particles larger than a prescribed size and a minimum of energy expended on sizes sufficiently fine.

Experimental work has been done to show that the gradual reduction of the larger sizes is an actual fact and not a theoretical conjunction.

Samples taken from the Marathon show a gradual reduction in the size of the feed passing through the mill—the reduction taking place first in the largest sizes. The results of screen analyses of material taken from each foot of length are plotted in Fig. 18, which shows curves practically parallel.

TABLE 7.—*Screen Analyses Showing the Progressive Work of Crushing in a Marathon Mill*

Screen Opening in Millimeters	Mesh	Feed, Cumulative Per Cent.	Foot No. 1, Per Cent.	Foot No. 2, Per Cent.	Foot No. 3, Per Cent.	Foot No. 4, Per Cent.	Foot No. 5, Per Cent.	Foot No. 6, Per Cent.	Foot No. 7, Per Cent.	Foot No. 8, Per Cent.	Foot No. 9, Per Cent.	Foot No. 10, Per Cent.
6.680	On 3	3.9	2.7	2.1	1.2	0.5	0.3					
4.699	On 4	15.0	11.4	12.9	6.5	3.8	2.2	1.4	0.6			
3.327	On 6	32.7	25.4	24.2	20.6	15.0	9.6	7.1	5.0	2.1	0.7	0.3
2.362	On 8	51.6	47.6	44.4	40.4	33.7	26.1	22.0	19.1	11.8	6.5	4.9
1.651	On 10	64.3	65.2	60.5	55.7	48.8	41.1	36.9	34.6	24.6	17.7	15.5
1.168	On 14	78.6	82.6	77.9	71.7	66.1	58.7	56.2	54.3	43.4	37.4	34.9
0.833	On 20	87.9	90.7	87.5	81.3	77.3	70.9	68.8	68.5	58.4	53.3	51.4
0.589	On 28	93.9	94.9	92.8	87.9	85.2	80.1	78.8	79.3	71.1	67.2	66.4
0.417	On 35	96.8	96.8	95.0	91.5	89.7	85.6	84.7	85.4	79.0	75.6	75.9
0.295	On 48	98.3	97.6	96.4	93.9	92.6	89.4	88.6	89.3	84.8	81.2	82.3
0.208	On 65	99.1	98.0	97.4	95.3	94.2	91.7	90.9	91.5	88.3	84.7	86.1
0.147	On 100	99.6	98.6	98.4	96.7	95.8	94.1	93.3	93.5	92.0	88.7	89.9
0.104	On 150	.....	99.0	98.9	97.6	96.8	95.8	95.1	95.0	94.2	91.6	92.4
0.074	On 200	.....	99.4	99.2	98.2	97.5	96.8	96.2	96.0	95.5	93.3	93.9
Pass 0.074	On−200	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

The above figures are cumulative per cent. of total weight. These screen analyses were made on a Tyler Ro-Tap machine under standard conditions.

It is noticeable that very little slime is formed in the first portion of the mill, the energy being expended on doing efficient work, crushing taking place on the larger sizes. This action is to be expected when one considers the principle on which the Marathon operates—the selective principle, crushing taking place on the larger sizes, reducing them to a certain size and then acting on the next size. Note the gradual elimination of the oversize of 6.680 mm., which disappears at the sixth foot of

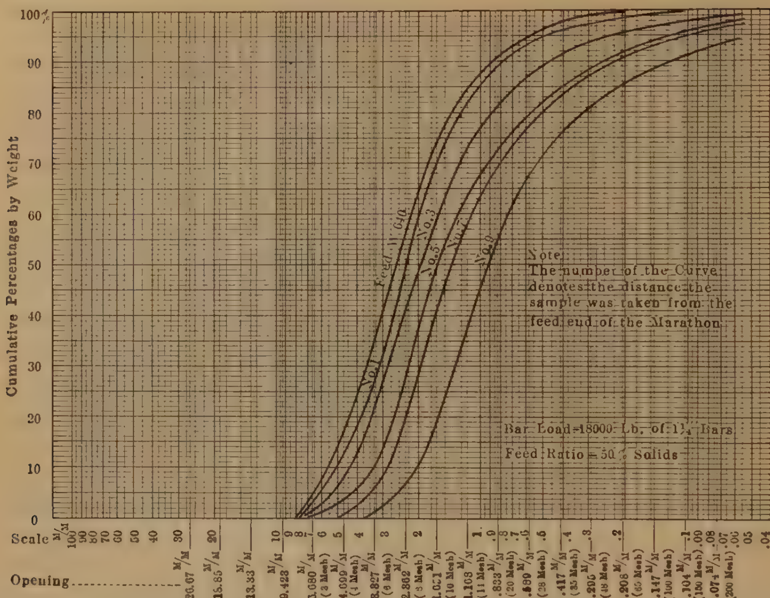


FIG. 18.—PLAT SHOWING THE PROGRESSIVE WORK OF CRUSHING IN A MARATHON MILL.

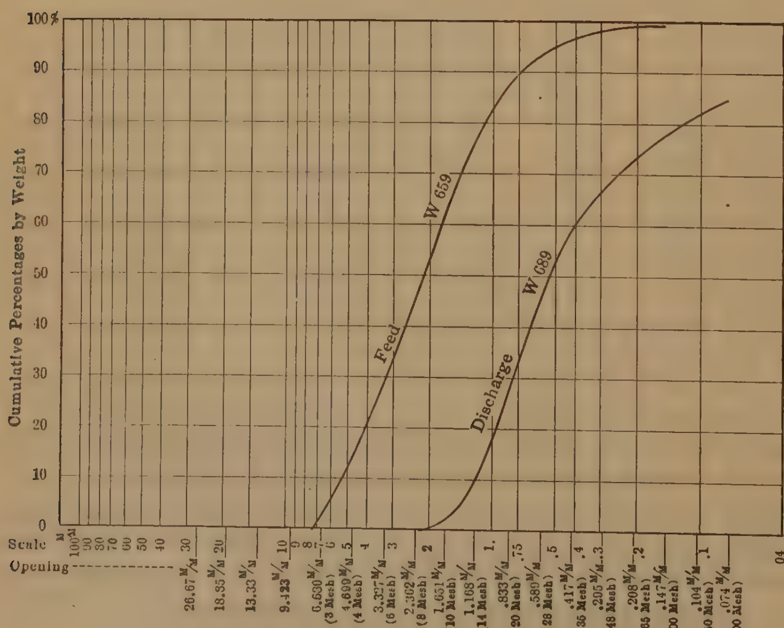


FIG. 19.—CHARACTER OF MARATHON MILL DISCHARGE.

the mill. The feed contained 15.0 per cent. on 4.699 mm., which totally disappears at the eighth foot of the mill. The feed contained 32.7 per cent. on 3.327 mm., but after passing through 2 ft. of the mill, it was reduced to 24.2 per cent., the next foot yielded 20.6, then 9.6, 7.1, 5.0, 2.1, 0.7 and the discharge contained but 0.3 per cent. on this size. The same results will be found if size 2.362 mm. be traced through the mill. Such consistent work in a regrinding mill is very remarkable. The phrase "regrinding mill" is a misnomer when applied to a Marathon; a Marathon is a recrushing, not a regrinding mill.

Rod Wear in the Marathon Mill.—It is generally assumed that the wear of the rods in a Marathon mill is the same at the two ends. Tests show this assumption to be wrong. Fortunately, the rods are not of the same diameter at the feed and discharge ends of the mill, if they were it is probable that the efficiency would be lessened. The rods at the discharge end of the mill wear more than at the feed end. This is to be expected, as more energy is required for crushing the fine sizes and likewise more steel. The difference in wear is not very great, but the difference in weight is a distinct advantage.

Measurements were made on rods which had been in use for a period of 2 to 3 months. This length of time was sufficient to show the character of the wear on the rods and would also determine whether the rods were to retain their circular shape, or to become ellipical. The rods, which were originally 1¼ in. in diameter, were taken by chance and were typical of the total charge. No endeavor was made to sort out even rods. The measurements are given in Table 8.

TABLE 8.—*Measurements* of Marathon Mill Rods*

Feed End of Rod		Discharge End of Rod	
Maximum Diameter	Minimum Diameter	Maximum Diameter	Minimum Diameter
70	68	65	62
63	63	58	56
70	66	65	60
73	69	67	67
69	67	64	61
66	63	61	58
66	63	61	59
67	65	60	59
69	68	66	65
69	67	64	60
1 1/16 in.	1 1/32 in.	63/64 in.	61/64 in.
Average....... 67/64 in.		62/64 in.	

* The unit is a sixty-fourth of an inch.

It will be seen that the rods retain a circular shape and do not wear elliptical, the original rods not being true circles.

As the rod diameter at the discharge end of the Marathon is less than at the feed end, we have another self-adjusting, highly scientific arrangement. The weight of the rods is adjusted to the size of particle that is being crushed. At the feed end the particles are large, so are the rods, while at the discharge end, the rods are smaller than at the feed end, so is the size of ore to be crushed; consequently, the weight of the rods is proportioned to the size of particles to be reduced. This is a logical condition, even though it be beyond our control.

Steel Consumption.—The steel consumption appears to be about 0.25 lb. per ton of feed when crushing the usual jig middling through 1.651-mm. screen.

Design of Marathon Mills.—It may be of interest to the mining profession to know that there is now being considered an entirely new type of

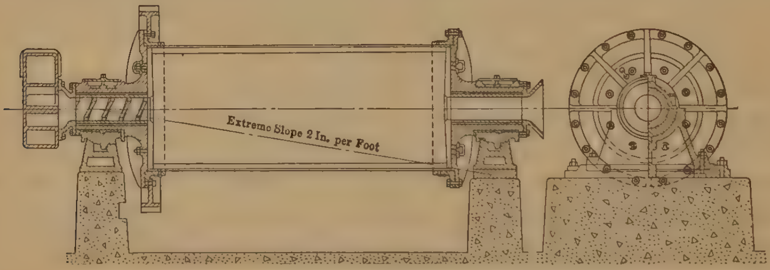


FIG. 20.—NEW DESIGN OF MARATHON MILL.

design for the Marathon. In the proposed mill, the rollers, tires and bearings will be entirely eliminated and the mill will be supported on trunnions at each end, as is done with standard types of tube mills. Arrangement has been made so that the head on the discharge end can be removed if at any time it becomes desirable to do so, as would be the case for lining the mill, etc. This new design has been worked out to comply with a desire on the part of the operators of the district for a Marathon of simpler design than the one now usually made (Fig. 20).

Treatment of Table Middling

The middling from the jigs may contain considerable pyrite, a typical analysis of a high iron-lead middling being 40 per cent. Pb, 15 per cent. Fe, and 1 per cent. Zn. This middling, together with the dolomite-lead middling, is crushed by rolls or regrinding mills. The discharge of the crushing machine is screened on 1½- or 2-mm. screens, the undersize passing to the table department where the tables yield a concentrate, tailing and a middling product, locally called "iron middling" and known

to the mill men as "sulphur." This iron middling contains pyrite, chalcopyrite and some galena and sphalerite.

The analysis of the iron middling will vary in different parts of the district, but typical analyses would be as shown in Table 9.

TABLE 9.—*Typical Analysis of Iron Middling*

Pb, Per Cent.	Fe, Per Cent.	Zn, Per Cent.	Cu, Per Cent.
22.0	18.00	7.0	0.4
30.5	9.65	11.3	...
19.7	14.40	19.5	...

In some cases the amount of copper in the iron middling may be of economic importance, but the iron is of no economic importance. The pyrite is objectionable, owing to the high sulphur content. The question of smelter fume is now receiving considerable attention in the vicinity of the lead smelters of the Mississippi Valley, and for this reason it is further desirable to eliminate the iron from the middling. In some cases the zinc may prove of sufficient value to warrant being saved, as at certain properties the iron middling may contain as much as 10 or 15 per cent. zinc.

There have been several methods used for handling this problem. One plant has roasted the iron middling, crushed it through 40 mesh and treated it on a Ding's magnetic separator. The resulting magnetic portion was shipped as a copper product. Laboratory tests indicate that such a product may contain from a fraction of 1 per cent. to as much as 5 per cent. copper, depending on local conditions. The non-magnetic portion was then tabled, a lead concentrate and a zinc concentrate being made. The grade of such a zinc concentrate will vary at different plants. A 50 per cent. zinc concentrate can, however, be produced under favorable conditions, but the usual zinc product contains 35 per cent. zinc, with possibly 10 oz. of silver. Some plants have cut this product out of the system and stacked it for future treatment. Some plants have kept the middling in circulation, wearing it out or gradually forcing it into the tailing, while others have cut the iron in with the general mill concentrate in order to get it out of the system, but none of these methods have proven altogether satisfactory. The amount of the iron middling may vary from 0.2 per cent. of the original tonnage to as much as 3 or 4 per cent. With the latter figures, the problem may become embarrassing, unless there be available adequate means of handling the product. The advent of flotation and the fine-grinding mill has, however, practically solved this problem. This iron is middling now usually treated as follows: The middling is retabled in order to separate the free lead, and the product, then containing the true iron-lead middling and much free iron, is sent to a 30 by 8-in. Hardinge mill, which is operated with balls, the dis-

charge of which will pass 80 mesh. The discharge is tabled, a lead concentrate being made and an iron tailing. Although much of the lead is free at this size, still the undersize of 200 mesh will show some true middling, but the amount of the iron middling is of very little importance. The iron does not come up in the flotation machine nearly as well as does the lead, and this difference in behavior permits of the pyrite being eliminated from the lead by means of flotation. The results of the following test show the action of galena, sphalerite and pyrite when treated by flotation. The pulp was treated in an agitation machine, crude creosote being used as the frothing agent.

TABLE 10.—*Flotation Test to Determine the Relative Action by Flotation on Galena, Sphalerite, and Pyrite*

Product	Pb, Per Cent.	Zn, Per Cent.	Fe, Per Cent.
Feed	4.22	0.196	6.38
Concentrate	53.40	2.500	5.00
Tailing	0.70	0.030	6.47

The relative recovery of the different metals is: Lead, 84.5 per cent.; zinc, 84.2 per cent.; and iron, 5.2 per cent. It will be noted that the recovery of the pyrite is very low and, consequently, flotation is an ultimate method for separating the pyrite from the galena.

Slime Concentration

The term "slime" has considerable latitude in the district and refers to the product passing a limiting screen of some definite size. Formerly 200 mesh was adopted as the common limit, but the introduction of large-tonnage tables has caused this definition to be slightly changed. These tables treat a "deslimed" feed, all material finer than 150 mesh preferably having been eliminated from the feed. It is thus desirable to overflow material finer than 150 mesh from the desliming apparatus, and this overflow is termed slime, since it is all minus 150 mesh. Some companies, however, still retain 200 mesh as the limiting screen, and references in this article will consider 200 mesh (0.074 mm.) as the limiting size.

Slime treatment in this district can be divided into two divisions: the methods used in the past, and those used at present.

Methods Used in the Past.—These methods depended solely upon gravity as a means of separation, and the collection of apparatus contending with this problem in the district was the same as has been used elsewhere—buddles, vanners, and canvas plants. Buddles have long since disappeared and vanners have been discarded by all but one company. Canvas plants have also been discarded. The latest canvas plant built was described in the *Engineering and Mining Journal* (Sept. 13, 1913).

Present Methods of Treatment.—At present slime is treated both by gravity methods and by flotation. The methods employed for saving the galena by means of gravity apparatus will be first considered. It may be briefly said that reciprocating tables are the only means now used for slime treatment. These tables give excellent results for treatment of material within their range of efficiency, as a table should make a satisfactory saving of galena in sizes coarser than 300 mesh and also do excellent work on the undersize of 300 mesh.

A test on a slime table treating the overflow of classifiers showed the following results: In the feed to the tables, 94 per cent. of the lead was −200 mesh. The resulting concentrate assayed 76 per cent. lead, 89 per cent. of the concentrate being −200 mesh. Of the tailing loss, 98.5 per cent. was −200 mesh and the screen analysis showed this loss to be all −300 mesh. The recovery was 65 per cent.

When flotation was introduced into the district, the tendency was to eliminate the slime tables and treat the entire slime tonnage directly by flotation, but the pendulum is now swinging back, which is the logical course. The usual practice now is to table as much of the slime as possible. The slime tables never make a finished tailing, the material in every case being treated by flotation, but tabling has served to lower greatly the grade of the flotation feed. Table treatment serves to yield a concentrate very much cleaner than is possible with flotation, while the smelting charge on the table concentrate is lower and the tonnage shipped is less than if the slime had been treated directly by flotation.

Some of the plants at present send the overflow of the desliming apparatus direct to the flotation plant, but such a method means a rich feed for the flotation plant. Other plants settle the classifier overflow and give it a table treatment, and by this means at least 65 per cent. of the lead is saved in the form of a table concentrate, assaying at least 75 per cent. lead. One plant is even re-treating by further tabling the middling from the slime tables. Another plant, by careful slime-table treatment, is sending a feed to the flotation plant containing but 2.25 per cent. lead, which feed before treatment by slime tables contained 6 per cent. lead.

Flotation of Slime.—The Federal Lead Co. did the pioneer work in this district on flotation and it was this company that solved the problem of floating the fine galena after it had been considered impracticable.

This company had the first operating flotation plant and knowledge there gained was freely given to other operators in the field, so that now flotation plants are found at every mill in the district. Roughly, the flow sheet used is as follows:

The slime is settled in Dorr tanks, the spigot product, containing 1 part of solids to 4 of water, is treated in flotation machines, with crude

wood creosote as the frothing agent. No acid is used. The concentrate is a finished product, while the tailing is treated on tailing machines.

The introduction of flotation in the district has not greatly altered the metallurgy of the ores. Flotation is not replacing gravity concentration except in the very fine sizes, and economically never can. Fortunately the physical nature of the ore of the district is such that an economic tailing can be made on the jigs and tables, and this fact precludes the possibility of flotation ever encroaching into the field of gravity concentration. The logical field of flotation in the district is treatment of galena particles finer than 300 mesh, as efficient work can be done with tables on coarser sizes than this. It will thus be seen that the field for flotation is sharply defined within certain limits, being limited to the treatment of slime products only—no attempt being made to displace gravity concentration by flotation on the sizes coarser than 200 mesh.

Operating Details.—The usual feed to the flotation plant consists of the overflow of the desliming cones, classifiers, jigs, drags and any other overflows. No attempt is made to keep the primary slime separate from the secondary slime. Part or all of these overflows may be tabled or, on the other hand, they may go direct to the flotation plant without preliminary tabling. The general slime from the mill, which may contain from 3 per cent. to as much as 10 per cent. solids, is settled, Dorr tanks being used universally for this purpose and giving excellent results. The Dorr tanks vary from 36 to 50 ft. in diameter, the depth from 6 to 8 ft. The tanks are built both of redwood and of steel. The slime settles rapidly, so there is no particular problem so far as slime settling is concerned. The capacity of the tanks varies, of course, with the coarseness of the feed and with the ratio of solids to liquid in the feed and in the thickness of the discharge of the tank. The settling area allowed per ton of dry slime varies from 11 to 16 sq. ft.—an average figure for the district would be about 13 sq. ft. of settling area per ton of dry slime. This figure applies when the discharge contains 20 per cent. solids. The power consumed by the Dorr tanks can be neglected, 1¼ hp. being sufficient to operate three 40- by 6-ft. tanks, together with the jack shafts.

Screen analyses of the discharge of the Dorr tanks vary according to local conditions. While one plant is treating a spigot product containing 20 per cent. on 100 mesh, for this district such a product is extremely coarse. In Table 11 are shown several typical screen analyses of flotation feed.

Table 11.—*Screen Analyses of Flotation Feed; Figures Show Cumulative Feed*

+100 Mesh	+150 Mesh	+200 Mesh	−200 Mesh
1.1	4.3	11.4	88.6
1.0	9.0	18.0	81.0
7.0	16.0	26.0	74.0

Material coarser than 150 mesh may be said to be foreign, the desire being to treat by flotation only material that will pass at least 150 and, preferably, 200 mesh. In any event, the lead present in the Dorr discharge is practically all finer than 200 mesh. Screen analyses of the discharge showing lead assays are given in Table 12.

Table 12.—Screen Analyses of Flotation Feed—Showing Distribution of Lead

	Sample No. 1		Sample No. 2	
	Cumulative Per Cent. by Weight	Cumulative Per Cent., Lead	Cumulative Per Cent. by Weight	Cumulative Per Cent., Lead
+100 mesh........	2.5	Trace	2.2	0.7
+150 mesh........	7.0	Trace	1.7	1.1
+200 mesh........	14.0	Trace	16.2	1.6
−200 mesh*.......	86.0	100.0	100.0	98.4

* 95 per cent. of the lead is −300 mesh.

The chemical analysis of the feed varies to some extent, since the lead may vary from 2½ to as much as 6 per cent., and the iron will vary depending upon the amount of pyrite in the original ore, and also upon the methods employed for table middling disposal. An average analysis of the flotation feed is given herewith.

Analysis of Flotation Feed

Pb, Per Cent.	Fe, Per Cent.	Zn, Per Cent.	Cu, Per Cent.	SiO_2, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	Al_2O_3, Per Cent.	CO Ni, Per Cent.
3.0	4.6	0.2	0.05	9.0	26.0	14.0	4.0	Trace

The percentage of zinc may be much higher than 0.2 per cent., in certain cases up to 0.5 per cent. or even higher.

The slime after thickening to 1:4 ratio is sent to the flotation machine, certain plants sampling this feed mechanically and others by hand.

Frothing Agent.—The frothing agent is first added at the flotation machine, no attempt ever being made to add it in the mill proper or in the ball mills. The metallurgy of the ores is such that adding creosote to the ball mills is impracticable. The most common frothing agent used is crude wood creosote, which gives excellent results, is very active, yields a clean concentrate and for over 4 years has held the field against all other oils. The use of creosote for this purpose was initiated by the Federal Lead Co. The specific gravity of the creosote is 1.08.

Creosote is fed to the first compartment of the primary machine in every case. Different methods are employed for feeding the oil, some companies using small plunger pumps, while in other cases it is fed from

pipe lines by use of pet cocks, but probably the best method is by the use of the disk feeder. This device is too well known to require any lengthy explanation. It consists of a disk from 10 to 20 in. in diameter with a face from 1 to 3-in., rotating from 10 to 20 r.p.m. The disk dips into a bath of creosote, the thin film of which is scraped off the periphery by a scraper having a micrometer adjustment. This is doubtless the best method employed for the purpose of feeding creosote. Some plants add the entire quantity of creosote at the first compartment, and even though the tailing from the primary machine be re-treated, no further addition of creosote is made, entire dependence being placed on the initial quantity added. The more common practice with the agitation type of machine is to add the frothing agent at several points. At one plant operating a 24-compartment agitation machine, of the total creosote used, 75 per cent. is added to the first compartment, 13 per cent. is added at the ninth compartment, 8 per cent. at the sixteenth compartment and 4 per cent. at the twenty-first compartment. The creosote consumption varies from 0.7 to 1.2 lb. per ton of dry slime, but the last figure approximates an average much more nearly than the first.

Types of Machine Used.—There are various types of flotation machines used in the district: the Federal, the Janney, the drum, and the pneumatic. In this district the pulp is in every case first treated in agitation machines, the tailing of which is treated in air machines or by those of the drum type, but in no case is the pneumatic type of machine used as a primary machine, its use being limited to that of tailing re-treatment.

The Federal machine, so called because it was developed by the Federal Lead Co., is similar to the standard agitation machines, consisting of an agitation compartment and a "spitz." These machines are generally preceded by two preliminary agitation cells without frothing spitzs, these two being necessary in order to thoroughly incorporate the frothing agent into the pulp. If an attempt be made to take off froth from the compartment to which the creosote is first added, it will be found that the creosote is not well emulsified in the pulp, but appears on the concentrate in large patches. This condition is not true where the Janney machine is used.

The agitation compartment is usually 24 in. square, the agitator being 18 in. in diameter and revolving 280 r.p.m, this speed having been accepted as being the most efficient. For higher speeds, although the extraction is increased, the power consumption increases very rapidly. Each compartment, at the speed indicated, requires from 4 to 4½ hp. The machines vary in length from 12 to 24 compartments.

Although in theory the one-level agitation type of machine is inefficient, so far it has not been displaced to any extent by other machines.

The Janney type of machine is too well known to need description. It is in use in three plants, always as a primary machine and always

preceding machines of the Federal type and preceding the tailing machines. This arrangement is necessary, since the Janney gives a much more violent agitation than the standard type and thus requires much less creosote. The tailing from the Janney machine if necessary is then treated in other types of machines. An attempt to clean up tailing from standard-type machines on Janney machines, resulted disastrously, because the tailing from the standard machine contained much more creosote than is necessary or permissible in a Janney machine, so the resulting Janney concentrate was dirty. The Janney machine takes 10 hp. per cell. It uses much less oil than the standard agitation machine and yields a cleaner concentrate. The usual belt trouble present in the belted type of machine is eliminated, owing to the individual motor drive. Each cell has an individual capacity of about 30 tons of dry pulp in 24 hr., so that 16 cells in series should handle from 450 to 500 tons per 24 hr.

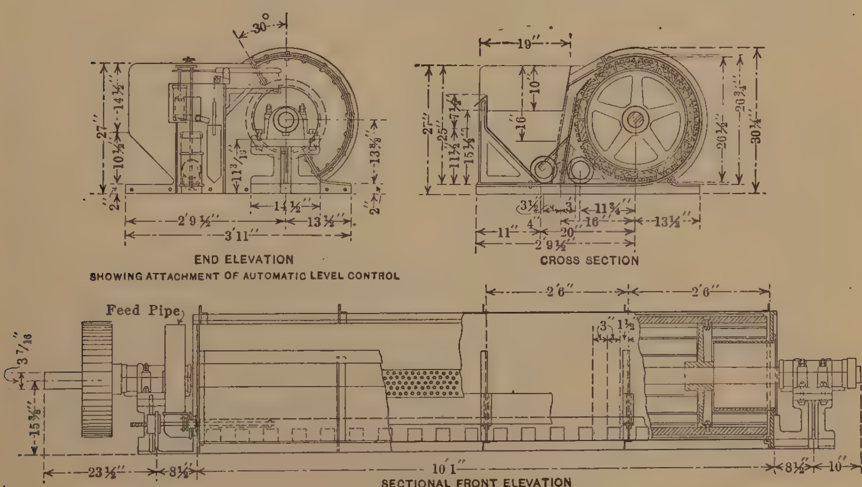


FIG. 21.—ELEVATION OF "K & K" FLOTATION MACHINE.

The pneumatic type is not used as a primary machine in the Lead Belt, but at one plant in the Fredericktown district Callow cells have been tried on primary feed.

The drum type of flotation is sufficiently new to warrant a few words in reference to it. As has been pointed out by G. C. Westby,[10] the drum type of flotation machine was first worked out by Rork and Sandberg. The "K & K" flotation machine is of the drum type, and as some of these machines are being tested in the district, it will be well to describe it.

[10] The Rork Sandberg Flotation Machine: *Engineering & Mining Journal* (Feb. 24, 1917), **103**, 336.

The K & K Flotation Machine.—The K & K flotation machine consists of a cylindrical housing 10 ft. long and 30 in. in diameter, to the front of which is attached a V-shaped compartment called the frothing chamber. The chamber is provided with a lip over which the concentrate is discharged into a suitable launder. Inside the cylindrical housing, or aeration chamber, is a drum comprising a 3 7/16 in. steel shaft on which are mounted four cast-iron spiders that carry the lagging and riffles of the drum. The shaft is supported by two main bearings, one at each end of the machine. The lagging forming the periphery of the drum is so spaced as to leave a 12-in. opening, or slot, between every two pieces. There are 16 of these pieces and each piece is provided with four hardwood riffles, which run the entire length of the drum. The drum revolves from 180 to 200 r.p.m. and has a clearance of 5/8 in. between its outside periphery and the inside surface of the aeration chamber. The shaft passes through the two end plates of the machine by means of two air ducts, and these air ducts are amply large to supply all the air necessary to aerate the pulp as it is revolved between the outside riffled surface of the drum and the inside surface of the aeration chamber. There is no leakage of pulp through these air ducts, owing to a continuous influx of air. This air is discharged by centrifugal force through the slots between each of the 16 pieces of riffled lagging forming the periphery of the drum, and immediately comes into contact with a thin film of pulp that has been circulated by the revolving drum. The pulp enters at one end of the machine and is discharged at the opposite end, and while passing through the machine, a distance of 10 ft., it is also caused to revolve through the aeration chamber. This longitudinal and rotary action causes the feed to move in the path of a helix, during which it is thoroughly aerated, forming a froth that carries the mineral and discharges it over the lip of the frothing chamber, while the tailing is discharged at the other end of the machine.

The floor space required is 4 by 14 ft. and the head room, 3 ft. The capacity of the machine is from 50 to 80 tons per 24 hr. and it requires from 10 to 15 hp. to operate it. The weight of the steel machine is 3500 lb. and that of the wood type is 2500 lb.

Three views of the machine are shown in Fig. 21. This type of machine appears to be very efficient in principle and the aeration is remarkable. The oil consumption is less than in standard agitation machines, and the power per ton treated is doubtless much less than in the usual agitation type of machine. This reason is not far to seek, since no "churning" is done in the machine, the drum touching only the surface of the liquid instead of being submerged as is the case with impellers of the usual agitation type. Sufficient work has not been carried out to determine what the wear will be on the drum and housing. The machine appears to give some trouble with choking when much sand is in the feed,

even though the feed has already passed through the usual agitation machine. The drum machine usually finds employment for tailing re-treatment, although experimental work is being done using this type of machine on original feed.

The flotation practice at all the plants is similar, differing only in detail. The pulp may be treated in standard agitation machines or in a Janney machine, either yielding concentrate and tailing. The tailing, however, is usually re-treated either with air machines or with drum-type machines, the secondary machines always making a finished tailing and a concentrate that may be a finished product or may require re-treatment. The tailing from the primary machines is a product much more difficult to treat than is the original feed, the resulting concentrate being of low grade, assaying from 30 to 40 per cent. lead. It is inadvisable to return this product to the primary machine for recleaning, but excellent results are obtained in treating it by cleaning on an individual Janney machine, which yields clean concentrate and tailing that is sent back to the primary machine.

In some cases the tailing from the primary machine is cleaned on drum-type machines. In general, these machines produce low-grade concentrate—not, however, due to any inherent defect in the machine, but to excess oil in the feed, as the machine requires much less frothing agent than does the usual agitation machine. The drum type of machine floats much of the pyrite that escapes the primary machines. This does much toward lowering the grade of the resulting concentrate, and the use of less oil in the preceding machine would assist in lessening this difficulty. At present the concentrate from the drum tailing machines is a finished product.

Concentrate Disposal.—The concentrate from the flotation machines may be removed mechanically with take-offs or permitted to discharge automatically. Both methods are used and in some cases take-offs have been removed, in other cases they have been added; there seems to be no rule regarding this point.

The concentrate as discharged from the machine carries from 10 to 15 per cent. solids. Spray water is generally used to break down the froth and to transport the concentrate to the pump. This wash water may be fresh water, but it is common practice to bypass part of the concentrate-pump discharge and use this portion as flushing water. In general the froth is not difficult to break down, but a persistent froth, which is probably due to an excess of creosote in the machine, is sometimes broken down by an excess of creosote being added to the pump sump. This is a method not to be commended on a score of cost, but nevertheless is a method frequently employed in emergencies. The froth from the standard type of agitation machine is usually the most difficult to break down, while that from the air machine gives much less trouble.

The froth from the drum-type machines is intermediate between the two. One company finds that by passing the flushing water over quicklime the froth is more readily broken down and that the concentrate settles more rapidly.

The general flotation concentrate passes to a sump and is elevated to a settling tank. If conditions are such as to permit its use, an elevator is preferable to a centrifugal pump for elevating flotation concentrate. In general, the settling tank is some distance from the flotation plant and a centrifugal pump is necessary—occasionally a "booster" is placed in the line when it is of considerable length. The Morris sand pump gives excellent satisfaction for this service, one such pump now having been in use for 19 months without repairs other than packing.

Dorr tanks are used entirely for settling flotation concentrate, and are built either of steel or wood, the average diameter being 40 ft. and the depth varying from 6 to 8 ft. The speed of the rake varies from one to two revolutions per hour. The use of the overload alarm is not common. In some cases the concentrate gives trouble by excessive frothing in the concentrate tank, while at other plants no particular trouble is experienced. This is probably largely due to the use of too much creosote in the flotation plant. At some plants the concentrate as discharged to the Dorr tank is broken down with a spray, while in other cases no spray is used. A froth guard is generally placed about a foot inside the overflow of the tank, extending a foot below the surface. The common practice is to use the standard type of overflow weir, although one plant is using a perforated overflow similar to that used in cyanide practice. As a precautionary measure, the overflow is returned to the original slime tanks.

The Dorr tanks are not operated for a continuous, but for an intermittent discharge. The flotation concentrate is sent to the Dorr tank continuously, but the settled concentrate is drawn off as demanded by the drying apparatus at periods of 1 to 2 hr. One plant permits the concentrate to accumulate for 12 hr., and then draws off the thickened product. Accumulating the concentrate for this length of time appears to give no particular trouble. Intermittent discharging of the concentrate gives a much thicker spigot product than is possible with a continuous discharge, the thickened concentrate containing from 35 to 45 per cent. moisture.

The discharge from the Dorr tank must be dried before shipment, and there are two methods of handling it, by steam-drying tanks and by filters of the drum type.

The steam-drying tank was the first method used in the district, but it is inefficient and is being rapidly displaced by the Oliver filter. The tanks used for drying flotation concentrates measure approximately 10 by 15 ft. and are 3 ft. deep. The entire bottom is covered with a nest of 2-in. pipes heated with steam under 50 lb. pressure. The concentrate was pumped direct to the tank and allowed to settle, and when a sufficient

amount had accumulated, the supernatant liquid was siphoned off and the steam applied. A later modification was to settle the concentrate in a Dorr tank and send the thickened pulp to the dry tanks. The dry tanks require from 24 to 36 hr. for performing their work and the cake, about 12-in. thick, is then loaded by hand to cars. This method of drying concentrate is expensive, requiring much hand labor and the use of much coal. The resulting cake contains from 6 to 15 per cent. moisture. This method of drying concentrate has been discarded by five companies, a drum-type filter replacing the tanks. It is retained by two companies, for purely local reasons. Although there are many disadvantages to the use of the steam-dry tank, it has one advantage over a filter, and that is that a dry cake can be obtained. The cost of installation of dry tanks is, of course, very much less than for a filter. The cost of drying concentrate with the tank is about twice that secured through the filter, the cost being equally divided between labor of loading, firemen's wages and coal. One ton of coal suffices for the drying of 4 or 5 tons of concentrate. The drum filter is rapidly replacing the steam-drying tanks for the purpose of treating flotation concentrate. Pressure filters find no use in the district.

The usual size of the filter in the district is 11 ft. 6 in. by 12 ft. This size has a rated capacity on the flotation concentrate of 50 tons per 24 hr. The feed to it contains from 35 to 45 per cent. moisture, while the resulting cake usually carries 15 per cent. moisture. A drier cake than this is desirable, but it appears difficult to obtain it with a filter.

The feed to the filter is usually heated with steam to a temperature of 130° F. A heated pulp increases the filter capacity 20 per cent. and yields a cake containing 2 per cent. less moisture than if steam were not used. In summer the use of steam is not so essential, but in winter it is a necessity.

The usual cake produced is ⅛ in. thick, slightly thicker when steam is used than otherwise. The lowest moisture reported in the district is 10 per cent., the highest is 18 per cent., while the average cake contains 14 per cent. to 15 per cent. moisture.

The air pressure used for blowing varies from zero to 20 lb., some plants using no air except for intermittent blowing. Where blowing is not done continuously, the resulting cake is drier than otherwise, and under these conditions the canvas is generally blown every 2 hr. Operation of the filter without continual blowing tends to blind the canvas and to decrease the capacity. Probably the best position for blowing is after the cake is removed and the compartment has passed the scraper. The height of the concentrate pulp line inside the tank varies from 10 in. to 4 ft. above the bottom of the filter, a lower level tending toward a drier cake, while a higher level yields a greater capacity.

The average length of life of the canvas is 3 to 4 months, at the end of which time it has become so clogged as to yield concentrate with high

moisture. Acid treatment of the canvas is not practiced in the district, but the use of an acid bath would allow longer life to the canvas.

Both wet and dry vacuum pumps are used, the dry pump giving the more efficient service. The vacuum under normal conditions is 26 in.; with inefficient pumps this may drop to 18 in., yielding a cake with high moisture.

The cake from the filter generally passes to a conveyor belt, which loads it directly into railroad cars, but at one mill it is first passed over a Lowden dryer. The cake fed to this dryer contains 15 per cent. moisture, while the discharged product contains 6 per cent. moisture. This product then discharges either to a storage bin or else to railroad cars.

The flotation concentrate made in this district will average from 45 to 55 per cent. lead. The remainder of the product may be either sulphides or gangue. Any excess of creosote tends toward lower-grade concentrate, both because the gangue floats and because it has the further effect of bringing up zinc. In some cases the low grade of the concentrate is due entirely to the presence of zinc. The zinc mineral (sphalerite) floats nearly as well as the galena. If an extraction of 85 per cent. be effected on the galena, the blende will show an extraction of 65 per cent. Any excess of creosote will cause the zinc to float as well as the lead. This is not true of the pyrite unless the agitation be very violent. In general, the flotation feed, concentrate and tailing show about the same analysis in iron.

A laboratory test gave the following results on a lead-zinc-iron slime. A slight excess of creosote was used, as is evident by the zinc extraction.

TABLE 13.—*Showing the Relative Extraction of Lead, Zinc and Iron by Flotation*

Product	Pb, Per Cent.	Zn, Per Cent.	Fe, Per Cent.
Feed	4.22	0.20	6.38
Concentrate	53.40	2.50	5.00
Tailing	0.70	0.03	6.47
Recovery	84.50	84.20	5.20

This sample contained a much higher percentage of lead and iron than is normally present in the flotation feed. Note the distinct selective action of flotation between galena and pyrite with the use of wood creosote.

The analysis of the flotation concentrate will, of course, vary with the analysis of the feed—the richer feed yielding a richer concentrate, and *vice versa.* A high zinc analysis in the feed will result in a low-grade concentrate, due to the presence of zinc.

A typical analysis of flotation concentrate is

Per Cent. Pb	Per Cent. Zn	Per Cent. Fe	Per Cent. Cu
50.0	3.0	4.0	0.5

With careful tabling of the slime in the mill before it is sent to flotation, the lead saved by flotation will all be through 200 mesh, and nearly all through 300 mesh. A screen analysis of flotation concentrate is shown in Table 14.

TABLE 14.—*Screen Analysis of Flotation Concentrate*

Screen Size	Weight, Per Cent.	Pb, Per Cent.	Total Lead, Per Cent.
+200 mesh	Trace		
+300 mesh	1.3	29.8	0.7
−300 mesh	98.7	53.2	99.3

The screen analysis shows the extreme fineness of the flotation concentrate produced in this district. The subject of the size of particles in the flotation concentrate will be discussed later in the article.

In general—through necessity, not choice—the cake as discharged from the filter is shipped to the smelter. A cake containing less than 14 per cent. moisture is highly desirable, but it appears to be impossible to make a drier cake than this with the extremely fine concentrate we have in this field. The fineness of the product, together with some colloidal material present, probably accounts for the high moisture in the resulting cake. Attempts to dry this cake on different types of home-made driers have not proved satisfactory. The drier to treat the cake from the filter should dry and move the concentrate mechanically to the loading bin or car. The putty-like consistency of the flotation concentrate makes it imperative that the cake be continually broken as it passes over the drier. Mere dragging of the product over a heated surface will not effect an efficient result. This has been proved by trial, as a continuous uninterrupted travel of the cake results in the concentrate "balling up;" the outside becoming dry and hard and the center remaining a pasty mass. The noticeable lack of granular constituents in the product makes the problem the more difficult. It thus becomes necessary to introduce a mechanical cutting device to break up the easily formed, dough-like mass.

The Lowden Rabble Dryer.—Apparently the problem of drying this product has been satisfactorily solved by the invention of the Lowden dryer. One is installed at a flotation plant in the district and the work done by it is so satisfactory that others are to be built by other companies. It consists of a hearth from 6 to 12 ft. wide and from 20 to 30 ft. long, heated by a flame produced either from coal or crude oil. The end of the hearth is close enough to the filter so that the cake may

drop directly to the hearth, where it is caused to progress by means of mechanically operated rakes. A sketch of such an installation is shown in Fig. 22.

The drying is done upon the hearth which is composed of cast-iron plates, beneath which is the flue, through which pass the products of combustion. The plates comprising the hearth are so joined as to prevent the escape of gases into the atmosphere as well as to prevent the loss of the concentrate into the flue below. The manner of providing for expansion and contraction is simple and effective, consisting in anchoring each plate by means of lugs, *N*, which enter corresponding sockets

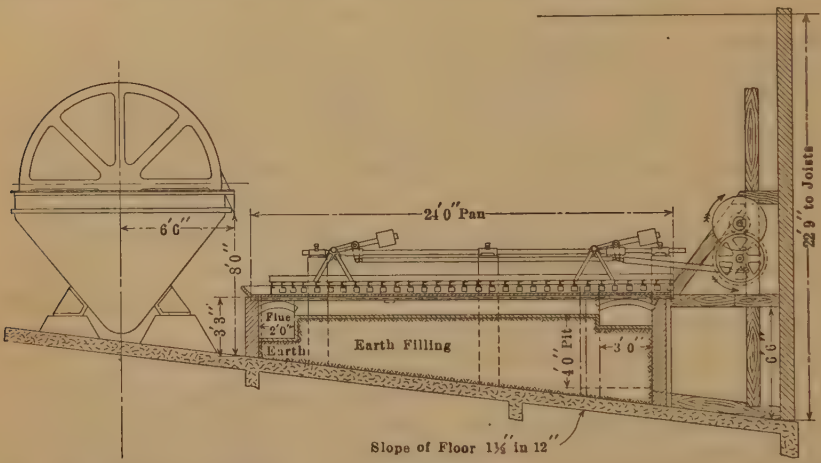


Fig. 22.—Lowden rabble dryer used in connection with a drum-type filter.

imbedded in the side walls of the flue. Provision is made for expansion lengthwise by the overlapping edges which have ample allowance for movement, and sidewise by the lugs, *N*, sliding in their sockets which are designed to permit movement in that direction.

The rake is built of channels, the transverse members bearing rabbles; those on one channel being right-hand and on the adjacent one left-hand, to avoid crowding the material to one side, and to break it up more thoroughly. These rabbles are attached to the channels by bolts through slotted holes, permitting a wide variation of inclination and consequently of speed of travel of the material over the hearth. This adjustability facilitates a rapid spreading of the feed over the entire width of the hearth in case it is delivered at one point, or if receiving the discharge directly from a drum filter wider than the hearth. The rabbles on succeeding transverse channels are so arranged that there is no tendency toward crowding the material to the sides.

The rake is supported and operated through the arms, *E*, attached to the rocker shafts, *F*, which are journaled in carriages, *K* fixed to recipro-

cating bars, H, which are free to move longitudinally in the bearings, I. The arms, E and G, together with the rocker shaft, F, form a ball crank, and the travel of this ball crank is limited by a lug, M, on the arm, G, and two lugs on the carriage, K, the latter having set screws as the actual points of engagement to provide adjustability. These set screws, indicated at L, form the means of easy regulation.

One set screw, at the right in the illustration, controls the depth to which the rake is lowered, and should be so adjusted that the rake just clears the bottom of the hearth. Another controls the height to which the rake is lifted at the end of the stroke and to regulate the length of stroke, as increasing the lift reduces the stroke, and contrariwise.

The range of adjustment of stroke is from an amount less than the distance between the transverse rows of rabbles, to an amount exceeding it, so that the rabbles of one transverse series can be made to enter the

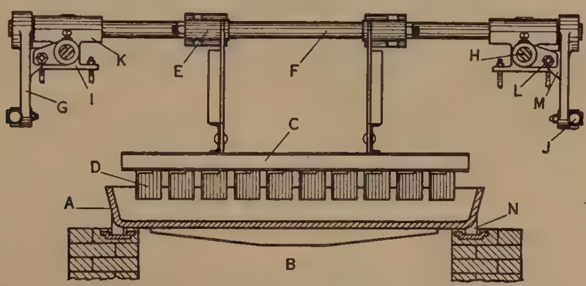


FIG. 23.—DETAILS OF THE MECHANISM OF THE LOWDEN RABBLE DRYER.

ridges of material between the furrows. Sufficient lift is provided at all points of adjustment, and any lift in excess of the necessary amount is not great and from an operating standpoint, immaterial, since the weight of the rake is counterbalanced and practically no power is consumed. The counterbalancing weights have been omitted from the figure for the sake of clearness.

The driving mechanism, not shown, consists of a shaft having a crank at each end, rotated slowly through the medium of suitable gearing. The main driving rods, J, connect these cranks with the arms, G, and rotation of the cranks would cause simple oscillation of G and E, but this is limited by the stops, and the excess of reciprocating motion causes a sliding of the bars, H, in the bearings, I.

Fig. 24 shows the parts near the discharge end of the machine. This is also the driving end, as it is advantageous to pull rather than push a rake of this kind, because all the parts are then in tension and vibration is avoided.

At the feed end, the rake is carried by a rocker shaft and arms similar to those shown, and the lower ends of the vertical levers are connected

to G by side rods, not shown. However, there is this difference between the supports at the two ends: at the discharge end a triangular rigid connection is used to receive the pin which works in the end of the arm, E, whereas at the feed end a simple hanger, pin-connected at top and bottom is provided, as the function of the rocker shaft and arms at that end is only to lift and lower the rake. This flexibility of the feed and hangers permits unequal expansion of the rake and reciprocating bars. A further difference is that the carriages, K, are provided with stops at the discharge end only, the side connecting rods causing the arms at the feed end to oscillate in unison with the stops at the discharge end and permitting free expansion from unequal heating without setting up strains.

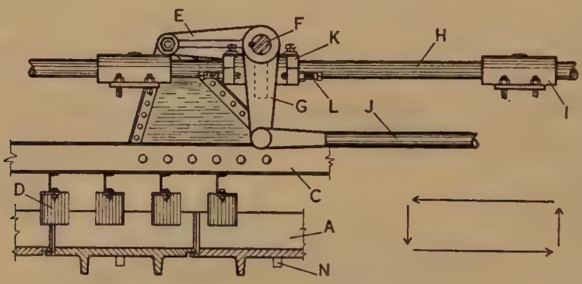


FIG. 24.—DETAILS OF THE MECHANISM OF THE LOWDEN RABBLE DRYER.

The travel of the rake is represented by the four arrows arranged in the form of a rectangle in the lower right corner of Fig. 24. The horizontal movement, both forward and backward, is a truly straight one, and the vertical movement nearly so. This true horizontal movement is of the utmost importance from an efficiency standpoint, as it enables the rake to operate close to the hearth at all points and prevents an accumulation of material at rest which would act as an insulator of heat.

From the foregoing description, the operation will readily be seen to be as follows: Assuming the driving rod, J, to be moving toward the right, the rake will be lowered till the lug on the arm, G, engages the set screw stop, L, when the reciprocating bar, H, will slide to the right, carrying the rocker shaft and rake with it, and this will continue till the crank passes the center and begins to move, J, to the left. As soon as the arm, J, commences the movement toward the left the contact of the lug on the arm, G, with the set screw, L, is broken and the arm, G, swings to the left until the lug engages the other set screw, the partial rotation of the rocker shaft having meanwhile lifted the rake clear of the material on the hearth. Further movement of the arm, J, pushes the rake backward until the driving crank passes the center, when the lowering preparatory to the forward stroke commences. This completes the cycle.

The ordinary types of hearth dryers are rabbled either by link belts working over sprockets and carrying transverse bars provided with rabbles placed upon the chains in intervals, or by one of the numerous forms of "grasshopper" conveyor mechanisms. With the former there is always the danger of material adhering to rabbles to such an extent as to clog them, in which event they will sweep the material from the hearth. With grasshopper rakes, swinging through an arc, it is impossible to rake close to the hearth, with the result that the deep layer of material beneath the travel of the rake acts as a heat insulator and seriously impairs the efficiency of the machine.

The rate of transmission of heat through cast iron is as rapid as can be absorbed by the concentrate in the vaporization of moisture, if the cast-iron plates can be kept free from a layer of insulating material. This the raking mechanism of the Lowden dryer accomplishes, and the objection to the grasshopper dryer in that respect is absent. The possibility of delivering large batches of insufficiently dried material, which is always present with chain-rabble dryers, is also absent, owing to the reciprocating action of the rakes, because any material that adheres to the rabbles falls back upon the hearth in practically the same place from which it was lifted.

The Lowden rabble dryer requires from 70 to 120 lb. of coal per ton of concentrate treated, reducing the moisture from 14 to 6 per cent. The discharge is all small enough to pass a 3-mesh screen and is in excellent condition for treatment at the smelter. The usual practice is to allow 10 sq. ft. of hearth area per ton of concentrate to be dried. It is advisable, however, to build the hearth large enough to avoid crowding.

The Flotation Tailing.—The method of disposal of the flotation tailing is discussed later under the heading of tailing disposal. The operators are not permitted to turn the tailing into the rivers, but are compelled to impound it or use other means of preventing pollution of the rivers. The average tailing made in the district will contain at least 0.6 per cent. lead. There are probably more tailings made containing above this figure than below it. The loss, as is to be expected, is practically all in the undersize of 300 mesh. Herewith is shown screen analysis of a typical tailing from a flotation plant.

TABLE 15.—*Screen Analysis of Flotation Tailing*

Screen Size	Cumulative Weight, Per Cent.	Pb, Per Cent.	Total Lead-Direct, Per Cent.
+100	2.0	0.10	0.3
+150	6.5	0.04	0.3
+200	10.5	0.02	0.1
−200	89.5	0.70	99.3

Over 98 per cent. of the lead in the −200-mesh product was finer than 300 mesh.

Some of the lead in the present flotation tailing can be recovered by further treatment by flotation. The concentrate, however, gradually decreases in lead content as the added flotation takes place. The ultimate tailing produced in the district will be reduced below the present analysis of 0.6 per cent.

The analysis of the flotation tailing is

Pb, Per Cent.	Zn,	Fe, Per Cent.	Cu,	SiO_2, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	Al_2O_3 Per Cent.
0.6	trace	4.5	trace	11	27	14	1.5

Power Consumption.—The power consumption varies, depending upon how far the retreatment of the tailing is carried, and upon the efficiency of design of the plants. With one exception, flotation has been installed at all the plants after the mills were erected. Furthermore, there have been decided changes in the methods of flotation. An increase in mill tonnage and the installation of fine grinding has increased the tonnage treated by flotation, and this has resulted in additions being made to the existing plants, necessitating the installing of secondary machines, pumps, etc. Consequently, the power consumption is more than would be demanded in a new plant designed in light of the present knowledge of the art. The following figures are for the power demanded for producing the concentrate, including pumping of intermediate products, but do not consider power for the operation of the filter or for elevating the final tailing.

The power consumption at different plants is shown, the variation being due to the extent of the treatment and the efficiency of design, as well as to the assay of the feed and of the tailing.

TABLE 16.—*Power Consumption in the District*

	Kw.-Hr. per Ton Dry Slime
Plant A	11.5
Plant B	10.0
Plant C	7.0
Plant D	6.8
Average	8.8

The plants taking but 6.8 and 7 kw.-hr. are making a lower recovery than are the two other plants. The production of clean concentrate and an 82 per cent. recovery will demand a total power consumption, including all power charges for agitation, pumping and filtering, of probably 16 kw.-hr. per ton of original feed. The figures will vary with the efficiency of operation and the type of machine used. The use of the drum and pneumatic types of flotation machines should greatly lessen the power consumption now required for flotation.

Percentage of Recovery.—The average recovery made in the flotation plants is from 80 to 85 per cent. This recovery probably can be increased. The flotation plants recover but a small percentage of the total lead saved, the lead saved in the mill proper being three or four times the amount recovered by flotation.

Economic Aspect of Flotation.—The cost of flotation is many times that of tabling, and the flotation concentrate is much lower grade than is the table concentrate, which shows that every effort should be made to minimize the tonnage treated by flotation. Flotation has many disadvantages; it is no panacea for all the ills of the district. As all the larger companies operate their own smelteries, consideration must be given to the cost of treatment of flotation concentrate by the smelter, together with the high dust loss. What must be considered is the greatest ultimate profit to the company, not the greatest profit to any certain department. Flotation concentrate is no unalloyed pleasure to the smelter superintendent, owing to its physical character. Every pound of lead recovered in the mill by gravity methods yields a greater profit to the company than it does if sent to flotation for treatment. Flotation produces a larger tonnage of concentrate than tabling, and its grade is lower. Due to the larger tonnage of low-grade product, the freight and smelting charges are higher for the flotation concentrate.

While it is not possible to discuss in detail the question of the economics of flotation, it can be said that every pound of lead saved by flotation yields a smaller profit than if the lead had been kept out of the flotation plant and saved in the mill.

Before leaving the subject of flotation, it may be of interest to digress for a moment and to mention a piece of apparatus which has been found of use in investigating problems in flotation. It is a means of obtaining further information regarding the distribution of sizes in the undersize of the finest obtainable screen.

The Thompson Cone Classifier.—At present the finest screen obtainable is 350 mesh. For sizing tests of material finer than this size, elutriation or classification becomes necessary.

The greater portion of the lead in the flotation concentrate is finer than 300 mesh. In order to determine its distribution a sample was submitted to G. W. Thompson, Chief Chemist of the National Lead Co. This sample was treated in a classifier invented by Dr. Thompson, primarily for the determination of the fineness of white lead and other pigments, etc. To my knowledge this apparatus has not been used before for metallurgical work in connection with concentration, and a description of it may prove of interest,[11] as providing a means for obtaining quantita-

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[11] Classification of Fine Particles According to Size. *Proceedings American Society for Testing Materials* (1910), 10, 601.

tive results from sizing very fine material. The following description is taken from Dr. Thompson's paper. It consists of four brass cones, placed one above the other, arranged so that the overflow of the first, passing through a funnel, discharges into the bottom of the second cone, and the overflow of the second cone passes to third, and so on. The apparatus is so arranged that it is operated with the hydraulic current at a constant head. Kerosene has been found most satisfactory as a floating agent, this selection having been made on account of the high wetting power of kerosene and its low viscosity.

The diameters of the cones are, for the first cone, 75 mm.; second cone, 106 mm.; third cone, 150 mm. and fourth cone, 212 mm. As the area of each cone is twice that of the one immediately preceding it, the rising current is one-fourth. Consequently there is a definite ratio of diameters of the products from the cones.

It must not be assumed that this classifier is simple and free from difficulties, and it is probable that for every particular material to be classified some special floating medium could be advantageously used.

A sample to be treated in the classifier is first passed through a limiting screen, the undersize is then diffused in kerosene, transferred to the first cone, and the current of kerosene started at a predetermined rate. The classification usually requires 2 hr. The product remaining in each cone, as well as the overflow, is settled, the kerosene decanted, the product washed on watch glasses, treated with ether and weighed. A sample of flotation concentrate was sized on No. 21 silk cloth, which is approximately 300 mesh, and the undersize was classified. The results are shown in Table 17.

Table 17.—*Classification Test on Flotation Concentrate in Thompson Cone Classifier*

Product	Weight, Per Cent.	Average Size of Particle, Mm.
Coarse Portion*	Trace	
Cone No. 1	12.55	0.0750
Cone No. 2	13.70	0.0500
Cone No. 3	12.30	0.0370
Cone No. 4	20.26	0.0250
Portion No. 5	41.16	0.0125

* Coarse portion is the phrase used in white lead work to signify the oversize portion on the No. 21 silk cloth.

This test gives quantitative figures about the size of the particles of the flotation concentrate produced in the district, which is seen to be extremely fine. These figures may be of interest, as to my knowledge none have been previously published showing the distribution in the flotation concentrate of the sizes finer than 300 mesh.

The samples of flotation feed gave the results shown in Table 18.

TABLE 18.—*Classification Test on Flotation Feed in Thompson Cone Classifier*

Product	Total Weight, Per Cent.	Lead, Per Cent.	Weight, Per Cent.	Lead, Per Cent.
Coarse portion	17.45	0.00	13.52	Trace
Cone No. 1	19.40	1.53	19.42	0.83
Cone No. 2	12.70	1.64	14.25	1.73
Cone No. 3	10.88	1.86	12.76	1.79
Cone No. 4	8.14	1.87	7.29	2.04
Portion No. 5	31.43	2.68	32.76	2.03

Re-treatment of Concentrate

It is a common practice in the district to re-treat the mill concentrate before shipping it to the smelter. One company, however, is shipping the concentrate direct without re-treatment. Part or all of the concentrate may be re-treated. All concentrate produced is either from the flotation plant or from the mill. As yet nothing has been done about re-treating the flotation concentrate. Test work, to be sure, has been carried on by the various companies, but at present no installation is attempting to re-treat the entire flotation concentrate in order to increase the grade. On the other hand, the general mill concentrate—or certain portions of the same—is receiving a second treatment before shipment.

The method employed for re-treatment of the mill concentrate is simple and is approximately as follows: The concentrate is elevated in a bucket elevator and deslimed. The spigot should be free of fine lead that will not readily table with the coarser sizes. The overflow of the desliming apparatus is settled; yielding an overflow to flotation and a spigot for tabling. The coarse material is treated on a coarse table and fine material on a separate table. The Butchart table is commonly used for the re-treatment of the concentrate. The coarse table yields a high-grade concentrate for shipment, a rich middling for regrinding and a "tailing" which goes back into the middling system of the original flow sheet. The fine table yields a high-grade concentrate and a tailing to flotation.

Disposal of Concentrate

There are two distinct classes of concentrates produced in the district, the general mill concentrate and the flotation concentrate. In general, for metallurgical and operating reasons, these products are kept separate at the mills and are shipped in separate cars to the smelter. It is common practice elsewhere to combine the flotation and table

concentrate, as such a combination frequently yields better results on the filters than does the treatment of flotation concentrate alone, but such an arrangement is not possible in the Lead Belt.

Throughout the Lead Belt the mill concentrate is handled and loaded mechanically into box cars. The cars, before loading, are lined with building paper to prevent loss due to leakage. The different methods in use for gathering and conveying the mill concentrate differs only in detail in the various plants. A common method is to launder the concentrate to a central Federal-Esperanza drag operating at a chain speed of 10 ft. per minute. These drags are similar to those used for desliming of the table feed and for dewatering the table tailing. The concentrate discharged from the drag is sufficiently dry to be put directly on a conveyor belt. This same general idea is carried still further by one company which has substituted a cable drag for the chain drag. The cable passes around a tail sprocket in a settling tank to which the concentrate is laundered. This cable, which operates at 60 ft. per minute, has 7-in. lugs fastened to it which drag the concentrate out of the tank and convey it through V-shaped launder to the loading station. This type of dewaterer and conveyor at one plant operates satisfactorily, but the same type of machine at another plant gave continual trouble and was later replaced by a belt conveyor. The Caldecott cone is being used by one company for the purpose of dewatering the mill concentrate. The Allen cone is used at one mill to dewater the general mill concentrate. In any event, whatever dewatering device may be used, the concentrate after being discharged from the dewatering apparatus passes either to a slowly moving conveyor belt, traveling 50 to 100 ft. per minute, or else directly to the car. This belt may discharge directly into the box car or may discharge to a tank where water may be drawn off before loading to the cars.

The overflow from the dewatering device in some cases passes directly to the flotation plant, while at other mills it is settled in tanks and shipped to the smelter or else thickened and tabled to remove the fine sand present. This overflow contains from 60 to 65 per cent. lead, the foreign material being practically all free sand. Re-tabling readily yields 80-per cent. lead concentrate.

The method of disposing of the flotation concentrate has been discussed under the subject of flotation.

The three largest companies in the district operate their own smelting plants, consequently there are no "ore buyers" to be seen in the Flat River district as is the case in the Joplin field. The St. Joseph Lead Co. has its smeltery at Herculaneum, Missouri (see Fig. 1). The Federal Lead Co. and the St. Louis Smelting & Refining Co. both have plants situated in Illinois, the Federal plant at Alton and the St. Louis plant at Collinsville.

Re-treatment of General Tailing

The question of re-treatment of the mill tailing has never received any attention in the past, and there is at present but one plant in the district doing it. In the past there has not been available an efficient means of recrushing the general mill chat and, secondly, there has been no efficient means of saving the lead that would have been freed. These conditions are now changed and the re-treatment of tailing is probably warranted. The development of the regrinding mills has shown a way for crushing the product, while the Butchart table is capable of handling a large tonnage of sand and the slime problem can be solved by means of flotation. It is highly probable that the question of tailing re-treatment will receive attention in the future. The current tailing from the plants can probably be re-treated at a profit. The average mill tailing in the district will average at least 0.8 per cent. lead. In some cases it is higher than this, especially where the jigs are crowded, in which event the jig tails will assay about 1 per cent. lead. The margin of profit through tailing re-treatment will never be large, but such a plant is a safeguard to the general mill operation and has a good moral influence on the millman.

Laboratory work has shown that re-treatment of a 0.9-per cent. tailing by jigging and tabling without further crushing will produce a 0.42-per cent. tailing, at the same time yielding a middling product assaying 4.2 per cent. lead. This middling product contained 62 per cent. of the lead that was present in the untreated tailing, and the ratio of concentration was 7.2 : 1. In this test, 90 per cent. of the lead recovered in the

TABLE 19.—*Screen Analysis of Re-concentrated General Mill Tailing*

Screen Opening in Millimeters	Mesh	Per Cent. Total Weight	Cumulative Per Cent. Weight	Per Cent. Lead	Per Cent. Total Lead	Cumulative Per Cent. Lead
6.680	3	5.4	5.4	0.48	6.2	6.2
4.699	4	18.4	23.8	0.62	27.3	33.5
3.327	6	18.2	41.0	0.57	24.9	58.4
2.362	8	12.7	53.7	0.55	16.8	75.2
1.651	10	7.6	61.3	0.41	7.5	82.7
1.168	14	5.8	67.1	0.32	4.5	87.2
0.833	20	5.3	72.4	0.47	6.0	93.2
0.589	28	7.3	79.7	0.17	3.0	96.2
0.417	35	6.2	85.9	0.08	1.2	97.4
0.295	48	4.9	90.8	0.04	0.5	97.9
0.208	65	3.4	94.2	0.02	0.2	98.1
0.147	100	2.8	97.0	0.00	0.0	98.1
0.104	150	1.3	99.3	0.00	0.0	98.1
0.074	200	0.3	99.6	0.31	0.2	98.3
Pass 0.074	−200	0.4	....	1.76	1.7	

middling was in sizes coarser than 2 mm. The results of the test seem to indicate that by careful rejigging and retabling, a product can be obtained that will pay to be further re-treated by fine grinding and flotation. The tailing from this test gave the screen analysis shown in Table 19.

The loss in the general tailing is in the larger sizes, the loss in the table sizes being of small importance. Re-treatment of the tailing would probably take place along the following lines: The tailing would be screened on 3 mm., the oversize either jigged for a middling product or else ground direct, then screened, the oversize being discarded. The combined undersize would be deslimed, the sand roughed over Butchart tables, yielding a tailing and a middling for cleaning on tables.

At present the only tailing plant in operation is the one at the plant of the Desloge Consolidated Lead Co. The flow sheet used is approximately as follows: The general mill tailing, after sloughing off the excess water, is jigged direct on a Harz 3-compartment jig, no preliminary screening being done. The jig yields a hutch product, a screen discharge and a tailing that passes to the chat wheel for final disposal. The jig hutch and side discharge is raised by a steel-chain elevator that discharges to a 6-ft. by 22-in. Hardinge mill. Incidentally, this Hardinge mill was the first ball mill to be installed in the district. The feed to the mill is approximately 300 tons in 24 hr. Creosote is added to the Hardinge-mill feed with a disk feeder and the discharge passes to an ingenious flotation drag machine. The mill discharge passes to the flotation end of the machine, which has a sloping canvas bottom. Much of the galena in the form of slime is raised in the froth, the slime discharges at the end of the machine, both being again re-treated in the main flotation plant. The flotation machine causes the sand to progress to the lower end of the air compartment, where it is picked up by the drag which discharges it to a flat screen of the Ferraris type, covered with 1½-mm. screen. The oversize is sent to waste and the undersize is tabled, yielding a concentrate and a tailing.

This simple plant is yielding excellent results and being operated at a comfortable profit. One man suffices for plant attendance. The power consumption is about 50 hp. and the upkeep is small. It will be readily seen that under such conditions a small lead saving suffices to pay operating costs. Although it is highly desirable to the management to have this plant operate at a profit, on the other hand a large profit in it would signify poor operation of the mill proper. This plant may be correctly termed a "pilot mill," under which condition its principal function is to indicate any abnormal loss taking place in the mill. At this plant the undersize of the screen that takes the discharge of the Hardinge mill is treated on one individual table. This table then really acts as a "pilot table" as regards the efficiency of the general mill operation. It is so

situated that the jig and table men can readily observe it, thus constantly giving them an index of the general mill condition. Any abnormal losses will be shown on this table within 5 min. of the time such losses begin to occur, and the mill men act accordingly—consequently, this plant, if it fulfills no other function than that of acting as a pilot plant, is an advisable investment.

Tailing Disposal

The question of tailing disposal is of considerable importance in the district. There are two tailing products that demand attention—the mill tailing, consisting of sizes between 10 mm. and 200 mesh, and the flotation tailing, which is finer than 200 mesh.

The method used in this district for disposing of the general mill tailing is unique because, owing to the topography of the country and the limited amount of water available, it is not possible to dispose of the tailing by purely gravity means. In former times the principal method used was to sluice all the tailing into "chat tanks," the overflow of which was sent to the mill storage pond, the chat being drawn off through cocks to cars. The cars were hauled to the dump and emptied, either by dumping or flushing. This means of tailing disposal was unsatisfactory, demanding at least one locomotive and train crew, with a corresponding high cost. If the chat were unloaded by hand from side-dump cars the cost was high, whiie if it were flushed from the cars it demanded a pumping station with its upkeep and power cost. In order to obtain head room for dumping the chat, it was necessary to build and maintain a trestle on which the cars were spotted for emptying. This method of chat disposal had many disadvantages and not very much to recommend it. This was the first method used in the district and, except in one case, has now been entirely superseded by other methods.

There is no attempt made to dispose of the mill tailing by means of elevators, as is the common custom in the Joplin district. The immense tonnages treated in the Lead Belt makes such a method beyond consideration.

The common method in use is by means of belt conveyors. This method gives but little trouble, has an almost unlimited capacity, the labor required for operation is small and the percentage of running time very large. The operation cost is half of that required by chat bins. The conveying belt is 24 in. wide and has a belt speed of 300 to 350 ft. per minute. The usual slope is about 16°. Steeper slopes are liable to give trouble, due to the chat sliding back, although one chat belt has a slope of 23° with a belt speed of 475 ft. per minute.

A screen test of an average sample of feed to this belt is shown in Table 20.

TABLE 20.—*Screen Analysis of General Mill Chat*

Screen Opening in Millimeters	Mesh	Per Cent. Total Weight	Cumulative Per Cent. Weight
6.680	3	6.9	6.9
4.699	4	19.0	25.9
3.327	6	17.9	43.8
2.362	8	11.7	55.5
1.651	10	8.4	63.9
1.168	14	6.5	70.4
0.833	20	5.0	75.4
0.589	28	6.7	82.1
0.417	35	5.6	87.7
0.295	48	4.5	92.2
0.208	65	3.2	95.4
0.147	100	2.5	97.9
0.104	150	1.3	99.2
0.074	200	0.4	99.6
Pass 0.074	−200	0.4	100.0

The moisture in the general tailing may vary from 8 to 20 per cent., the former figure being much drier than the average, while 20 per cent. is probably higher than usual. A chat with 18 per cent. moisture causes trouble by sliding down a belt traveling 350 ft. per minute with a slope of 16°.

Where there is sufficient head room in the mill, the common practice is to launder both the jig and table tailing to a common point where they are dewatered by means of chain drags or by dewatering wheels. Owing to heavy duty of the dewatering devices, when a central dewatering unit is installed, the wheel is usually specified. It consists essentially of a heavy pulley to the rim of which are bolted blades. The wheel revolves 2 r.p.m. and drags the solids out of the tank in which it operates, while the overflow is laundered to the flotation plant.

When it is not possible to launder the tailing to a central dewatering point, it is customary to install drag dewaterers. These drags vary, of course, regarding size, speed and details, but in general they consist of two head sprockets and two tail sprockets over which operate chains, joined transversely by blades. The chain speed is between 20 and 40 ft. per minute. The mechanism rests on a tank with a sloping bottom, the water level being a few feet below the point of discharge for the solids. The drag is fed either at the extreme tail end or near the center. The overflow may be situated at the extreme tail end, at the sides or at the center; in the latter case the opening is supplied with a cover to prevent oversize dropping off the blades into the overflow.

In some cases the chat belt runs the entire length of the mill, the different drags discharging to it for the entire length. The horizontal belt,

which is virtually only a gathering belt, discharges to an inclined belt. The inclined belt discharges the chat from the end of it. When the chat has built up to the discharge point, it becomes necessary to add a second leg, or else to lengthen the present belt. The belts are lengthened by the addition of 75-ft. sections as long as possible, when a new leg is started. Provision may be made for installation of a mechanical sampler.

As a usual thing the standard type of conveyor idler is used. Certain plants are having their own castings made for the idler rollers, an extra

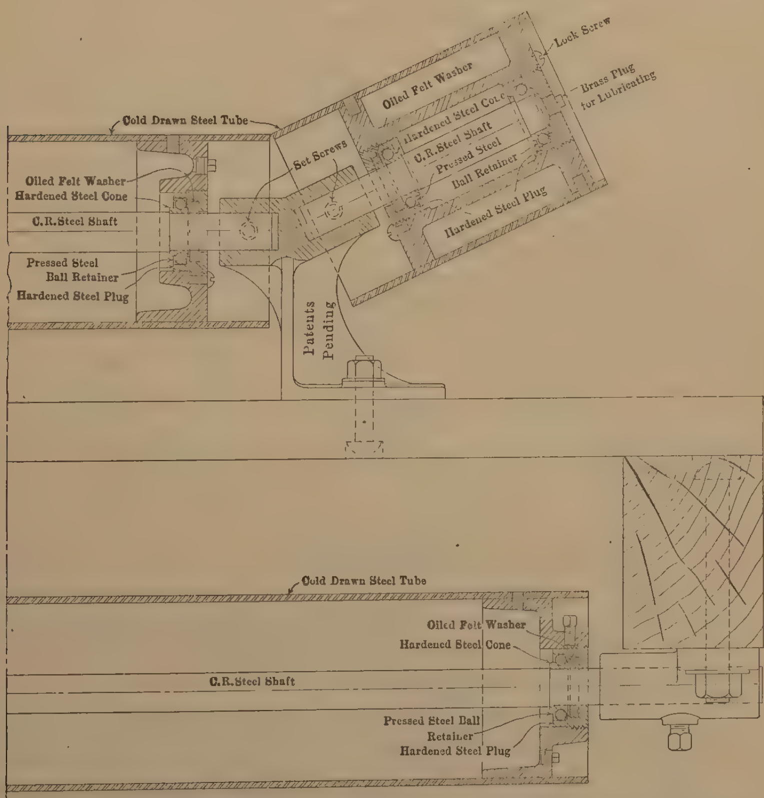

FIG. 25.—SECTIONAL VIEW OF "CONVEYING WEIGHER" BALL-BEARING IDLER.

thick rim being provided. The idlers, however, operate under severe conditions and the wear on them is heavy. The wear of the idler is not due to actual "sticking" of the idler, but to the "slip" which occurs between the belt and the roller. This "slip" is present whenever a belt and pulley are used, and the same conditions occur on a conveyor, while with a wet conveyor the condition is much aggravated. An examination of an idler roller will show a concave surface due to wear. This wear indicates a differential speed between the idler and the belt—the greater

the differential, the greater the wear. In an attempt to overcome the wear, attention has been turned to the ball-bearing idlers. Ball-bearing idlers of one make failed because grit was washed into the bearings and soon caused the idler to stick, causing the belt to cut through the roller and form a knife edge which soon ruined the belt. These idlers might have proved satisfactory in a dry position, but for work on the chat belt they were a dismal failure. They were replaced by the standard type.

A ball-bearing idler designed on most approved lines is now in use in the district and giving results which are highly satisfactory. A sectional view of the "Conveying Weigher" idler is shown in Fig. 25. Fastened to the shaft of the idler is a hardened-steel ball-race cone, the balls confined by a ball retainer. A hardened-steel plug on the outside completes the ball race. The use of felt washers retains lubricant in the idler and at the same time prevents the entrance of grit. The sketch shows the details of construction.

These idlers have given complete satisfaction under the most trying conditions, and have given no trouble by failure to rotate. As they require lubrication only twice a year, the lubricating and inspecting cost is practically eliminated. An additional advantage is that they require approximately only 60 per cent. of the power demanded by standard-type idlers.

As one leg of the chat belt discharges to another, it becomes necessary to clean the discharging belt. This may be accomplished by revolving brushes, water sprays or air jets. The brush causes greater wear than the water or air and, consequently, is seldom used. When it is impractical to use a water spray, recourse is had to the air jet. This consists of an inch pipe, drilled with small holes, about 3-in. centers. The air pipe is set at an angle of 45° with the belt in order that the jets may "rake" the belt as it passes. The belt is housed in to prevent freezing in the winter, the housing and runaway being carried on either steel or wood construction.

The tailing at the top of the conveyors must be sluiced away. Semicircular launders of No. 10 iron are in common use for this purpose, each section being 4 ft. long, bent to a radius of 9 in. They are laid directly on the chat and their position is altered by the man attending to the chat belt. They are given a fall of 2½ to 3 in. per foot.

Flushing water may be either fresh water or tailing from the flotation plant. A common practice is to pump fresh water to the end of the chat belt, while the tailing from the flotation plant is pumped but part way up the chat pile, thus washing out a space that is constantly filled with the incoming chat. The chat piles of the Flat River district are prominent features of the landscape.

Flotation Tailing.—To prevent pollution of streams, tailing is not permitted to flow directly into them but is either impounded or filtered

through coarse-tailing piles before entering the streams. Two companies have built large impounding dams and are impounding all the flotation tailing. The more common practice is to filter the flotation tailing through the chat. The chat forms a very efficient filter and, under proper conditions, will remove the slime from the flotation tailing, giving clear water. At one plant excellent results are obtained by filtering, but it is the safer and more desirable method to impound the flotation tailing.

Means of Conveying and Elevating Ore and Pulp

Means employed for conveying and elevating ore and pulp follow standard practice. In the coarse-crushing plants, steel-pan conveyors are in general use for handling ore from mine size down to 3 or 4 in., and are operated at a speed from 15 to 65 ft. per minute, the steepest steel conveyor in the district operating at a slope of 24°. The width of pan conveyors varies from 30 to 48 in. and the pans are usually protected with oak wearing planks.

The usual type of rubber belt conveyor is used for transportation of ore in sizes below 3 or 4 in., but at Mine La Motte sizes from 8 to 4 in. are carried on rubber belts. The usual practice, however, is to limit the field of the belt conveyor to sizes below 4 in. The average width of belt is 24 in., but at Mine La Motte 48-in. belts are employed. The steepest conveyor belt in the district has a slope of 28°. The belt is 20 in. wide and has a speed of 300 ft. per minute. This slope is due to local conditions, and not by desire, and is not to be recommended for usual practice. Skirt boards are required on the edge of the belt to prevent the large ore rolling off the belt. The ore conveyed is the discharge of the coarse-crushing plant and contains 2 per cent. moisture, a screen analysis of which is given in Table 21.

The belt conveyor is universally employed, with the usual type of tripper, for distribution of the crushed ore to the mill bins.

In the coarse-crushing plant, where local conditions make it impossible to install belt conveyors for raising the ore to a higher level, use is made of continuous steel-bucket elevators. They give no particular trouble, handle a large tonnage and make possible a nearly vertical elevation of the ore. They operate at a speed of 50 to 75 ft. per minute at an inclination of 45° to 65° from the horizontal. The largest elevator in use employs buckets measuring 42 by 24 by 14 in. and has a rated capacity at 60 ft. per minute of 200 tons per hour.

The standard type of belt-and-bucket elevator is also used for elevating ore, both dry and wet. The size of feed, however, in these cases is usually limited to material smaller than 2 in., whereas the steel-bucket elevator is used for larger sizes. The belt elevator is used in the mill

Table 21.—*Screen Analysis of Ore Transported on 20-in. Conveyor up 28° Slope*

Size of Screen, In.	Size of Screen, Mm.	Per Cent.	Cumulative Per Cent.
*3	76.200	3.8	3.8
*2½	63.500	3.6	7.4
*2	50.800	17.1	24.5
*1¾	44.200	11.2	35.7
*1½	38.100	14.8	50.5
*1¼	31.750	11.1	61.6
	18.850	10.0	71.6
	13.330	8.1	79.7
	9.423	4.6	84.3
	6.680	3.2	87.5
	4.699	2.3	89.8
	3.327	1.7	91.5
	2.362	1.5	93.0
	1.651	0.8	93.8
	1.168	0.8	94.6
	0.833	0.6	95.2
	0.589	0.6	95.8
	0.417	0.5	96.3
	0.295	0.4	96.7
	0.208	0.3	97.0
	0.147	0.4	97.4
	0.104	0.4	97.8
	0.074	0.2	98.0
Pass	0.074	2.0	100.0

* Designates round opening, others square.

proper for elevation of the dry ore, as well as for elevating the middling. One plant, however, uses steel-chain elevators for wet work. The belt elevators operate at the customary speeds, from 300 to 500 ft. per minute, varying with the diameter of head pulley. Malleable-iron buckets are usually employed, although the use of manganese-steel buckets is now becoming more common. Manganese-steel buckets are giving excellent results at one plant for elevating the product from the crushers, which are set at 2 to 2½ in. The cost of manganese-steel buckets—in place on the belt—is two and a half times that of malleable-iron buckets. The former, however, after outwearing four sets of the malleable type are still in good condition. The highest belt elevator in the district is 68 ft., operating at 385 ft. per minute with an 18-in. belt. The belt elevator is the usual, and probably the best means, of elevating wet pulp. The centrifugal pump is still in use in many cases where it is necessary to convey the pulp some distance, or where lack of room makes the considera-

tion of an elevator impossible. One plant is making use of the Frenier sand pump for elevating fine material.

The cable drag is used in some plants for moving of the general concentrate, but one plant has replaced the drag with a belt conveyor, resulting in satisfaction and a much smaller upkeep cost. The shaking launder is used at one plant for transporting the mill concentrate to a central point. Usually, if operated with the usual table-head motion, these shaking launders must be limited in weight to about 800 lb. They are constructed of wood, being about 8 in. wide, 4 in. deep and have no iron lining, and the speed of operation is 220 strokes a minute. They are suspended or supported by wooden arms, 10-ft. centers. One plant has replaced the shaking launder by a belt conveyor, with most satisfactory results.

Centrifugal pumps or belt elevators are in general use for elevation and transfer of the flotation concentrate. The pump is used where the pulp must be transferred some distance, while the elevator is employed for elevation of the thickened discharge of the Dorr tanks.

From a study of the different means employed in the district for elevation and conveyance of ore and pulp, apparently the steel-pan conveyor is the most satisfactory method of transporting crude ore. Sizes below 4 in. can be elevated with the steel-bucket elevator. When conditions warrant the installation, the belt conveyor proves the most satisfactory for transporting ore in sizes below 4 in., either dry or wet. The standard belt-and-bucket elevator is the most common method of raising ore or pulp in the mill, the use of the centrifugal pump being confined to a limited field.

Water Supply and Consumption

The primary water supply is usually obtained from the mine water. In a few cases make-up water, when required, is obtained from pumping stations maintained at Big River and Flat River, but in many cases the mine water proves ample for milling purposes. Owing to the fact that the general mill tailing, as stacked, does not contain over 15 per cent. of water, it will be seen that the demand for make-up water is not large.

With a mine-water supply of 700 gal. per minute, a 3000- to 4000-ton mill can be operated, including the water required for the power house.

Available water for milling purposes, although not of excessive amount, is yet sufficient for daily operation. There is seldom, if ever, any delay in the mills due to lack of it. Water is used much more plentifully than is considered good practice, but this is largely due to the comparatively flat country which demands an excessive amount of water for conveying the pulp. The size of the jig products—from 10 to 2 mm.—also demand more water than would be required were the ore crushed finer. The changes in the flow sheets in many of the mills has in certain cases

decreased the slope of the launders from that in the original design. Some mills are able to send the tailing and concentrate direct to belts or drags, while others are required to sluice these products some distance. In one case when the jig tailing was being sluiced to chat tanks, 280 gal. per minute of flushing water being required to transport 388 tons in 24 hr., on a basis of 2500 tons of original feed, the flushing water for jig tailing alone was 1000 gal. per minute. In another case where the general mill concentrate was sluiced to a concentrate drag, as much as 800 gal. per minute of water was required for a 2500-ton mill. Insufficient launder slopes account for the excessive water consumption.

The largest loss of water occurs in the tailing from the flotation plants, which may amount to from 50 gal. per minute per 100 tons of dry slime treated in 24 hr., depending on the ratio of liquid to solid in the flotation feed. Evaporation, minor leaks, etc., account for an unknown amount. Power in the district being cheap accounts to some extent for the high water consumption, as in some cases water is being used as a conveying medium rather than mechanical means.

The mill water is used at atmospheric temperature. One mill, however, combines the mill water and condenser cooling water, and although this arrangement works satisfactorily in the winter time, it has many disadvantages in the hot weather. In the winter, with the outside temperature 20° above zero, the mill water is 50° F. In the summer time with the outside temperature 98°, the mill water reaches a temperature of 110° F. The hot water greatly shortens the life of the belts of the elevators and conveyors. Only the best grade of linoleum will resist the heated water, the cheaper grades quickly going to pieces.

The water used per ton of original mill feed varies from 7 to 11 tons per ton of ore milled. The water is hard and causes trouble by gradually choking the pipes. The mine water may contain as high as 30 grains per gallon of solids, largely present as calcium and magnesium salts. The calcium salts deposit a scale on all the launders, table tops and pipes, and make necessary frequent cleaning of the table decks.

Sampling Methods

Sampling is not given the attention it should receive. This condition exists, not so much because of lack of appreciation of the importance of the subject, but rather because of the arrangement of the plants and the fact that many of the mills were erected before the importance of adequate sampling was fully recognized. Three plants have complete plants for sampling the feed to the mill. At one of the plants the ore is weighed on railroad scales before being emptied into the crude-ore bins. At the other two plants the ore is weighed on the belt as it passes over conveyor weighers on the way to the mill bins. At the other mills

no provision is made for automatic sampling of the mill feed. Hand samples are taken in these plants, usually after the ore has been crushed through the first limiting screen, although at one plant the mill feed sample is taken as it enters the mill, about 2 in. in size.

Of the seven mills in the district, five are equipped with conveyor-belt weighers. The other plants depend for their tonnage figures upon weighing part or all of the mill feed on track scales.

No attempt is made to check up the work of any of the individual sections or departments of the mill by the use of automatic samplers. All mill samples are taken by hand, in consequence of which the results obtained are not of much weight.

There are but two mills in the district using an automatic sampler for the general mill tailing. The other plants depend on hand-sampling.

Some of the plants have mechanical samplers operating in the flotation plants for sampling the feed and the tailing, but the general practice is to depend on hand samples. The question of sampling should receive very much more attention than it does now, and doubtless more attention will be given to this important, but neglected, subject in the future.

Power

Owing to the close proximity of the district to the coal fields of Illinois, coal laid down at Flat River costs $2 per ton. The cost of power varies from 0.5 to 1.0 c. per horsepower-hour, depending on the type of prime mover, and local conditions. Gas engines, reciprocating steam engines and steam turbines are in use, the latest installations in general being large-unit steam turbines. In general the mills are motor driven, but two mills are driven directly from line shafts, which are belted direct to the mill engine. Both direct and alternating current is used, the newer installations being of the latter type. One plant, however, uses only direct current. The low cost of power is one factor that makes possible the profitable operation of the low-grade ore of this district.

Mill Construction

There are seven mills in the district, three of which are of wood construction. The other four mills are of more recent date and are of steel with concrete floors. The No. 4 mill of the Federal Lead Co., which started in January, 1914, is the newest plant in the district. It is built entirely of steel and concrete.

Power Consumption Per Ton of Ore

The power consumption per ton of ore milled depends upon the extent to which regrinding and flotation is carried as well as upon the design of the plant. Average data are shown in Table 22.

TABLE 22.—*Power Consumption*

Operation	Kw.-Hr. Per Ton Ore Milled
Crushing	2.80
Concentration	1.70
Flotation	1.10
Water service	1.45
Total	7.05

The number of tons milled per man varies with the size of the plant and the efficiency of design. The extreme figures would be from 38 to 43 tons per man.

Thawing of Ore

Owing to the lack of extreme cold weather, thaw houses are not in general use, although they are essential at the smelters for thawing the concentrate in the winter months. A thaw house recently constructed for the thawing of ore may prove of interest. A view of the building is shown in Fig. 26.

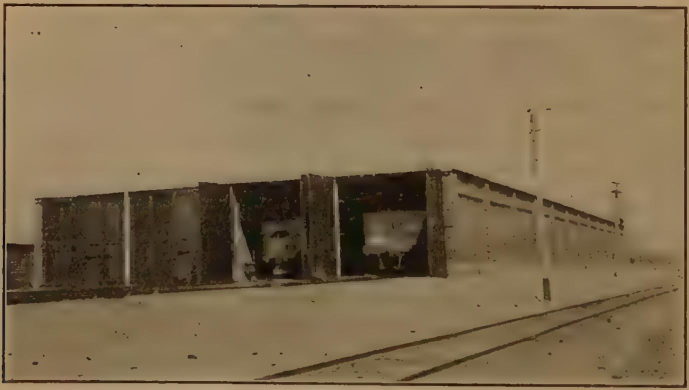

FIG. 26.—CONCRETE TILE THAW HOUSE.

The building measures 66 ft. 8 in. by 240 ft. and is 15 ft. 3 in. high at the center. It is constructed of concrete and hollow tile, is steam-heated and has four standard-gage railroad tracks providing for thawing 24 ore cars at one time. The building is divided into two compartments by a wall running longitudinally down the center, and in the center of each compartment is a line of concrete columns. The walls are constructed of 8-in. Denison hollow tile, with 8 by 28-in. pilasters on the outside of the the side walls and double pilasters for the center wall. Pilasters and columns are placed approximately on 17-ft. 8-in. centers.

The roof, which has a slope of ½ in. per foot, consists of reinforced con-

crete with hollow tile imbedded in the lower side. These tile are 4 by 12 by 12-in. hollow partition tile, placed 3½ in. apart. The concrete between the tile is reinforced with ⅝-in. rods. The roof is 6 in. thick and the tile is used to replace part of the concrete in order to provide a better heat insulator. The roof is divided into four sections by two expansion joints that run through the roof on the two center lines. The roof is waterproofed with "Certified Cement."

In order to reduce the heat losses to a minimum, no windows were provided in the building, which is steam-heated, coils being placed along the side and center walls.

Flow Sheets Now in Use in the District

It may be of interest to show some flow sheets in use in the district, and for this purpose there has been selected the one used by the St. Louis Smelting & Refining Co. and the one used by the Federal Lead Co. at their No. 4 mill.

Flow Sheet to be Used at Mill of St. Louis Smelting & Refining Co.

The flow sheet shown is the one to be used by the St. Louis Smelting & Refining Co., at their St. Francois plant. This mill is 17 years old and was designed to treat 1200 tons a day. The plant is now being altered to treat 4000 tons. The flow sheet to be used in the redesigned plant will be approximately as shown in Fig. 27. A view of this plant is shown in Fig. 28.

This flow sheet will be slightly changed as time permits, the dry-crushing rolls and the 10-mm. trommels in the mill will probably be eliminated and the ore will be crushed to 10 mm. in an outside crushing plant. The flotation-concentrate drying tanks will be replaced by a filter of the drum type, followed by a dryer.

Flow Sheet Used at No. 4 Mill of Federal Lead Co.

A view of this plant taken just after operation commenced is shown in Fig. 29 and the flow sheet used is shown in Fig. 30. The mill started in January, 1917, and is the most recently constructed plant in the district.

The mills operate three 8-hr. shifts 6 days a week. A local law prohibits operation, but allows repair work to be done on Sunday. The mill force is entirely American, mostly local men. The labor is not of a drifting nature, many of the millmen having been with the individual companies for years.

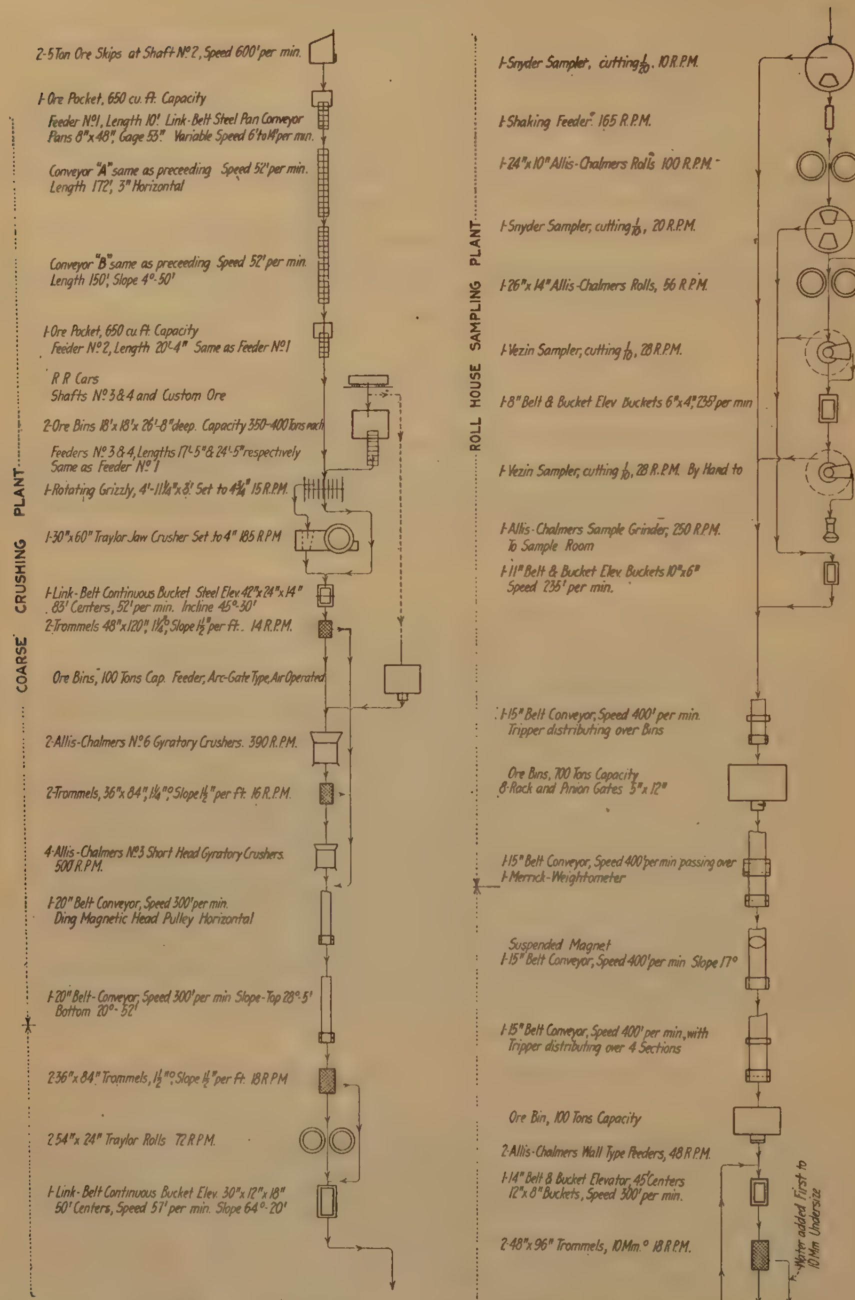

FIG 27.—FLOW SHEET TO BE USED AT ST. FRANCOIS PLANT OF ST. LOUIS SMELTING AND REFINING CO.

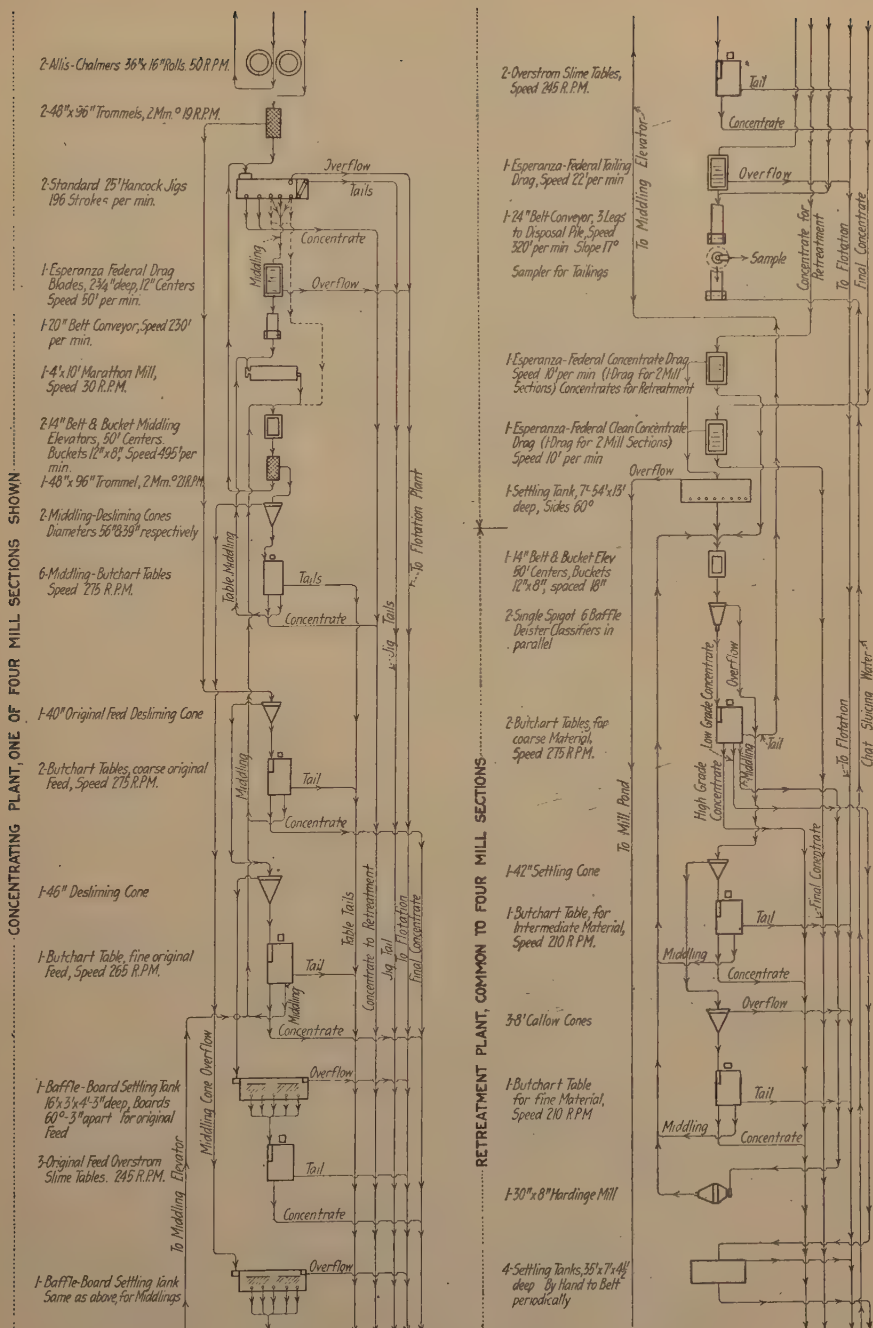

Fig. 27.—Continued.

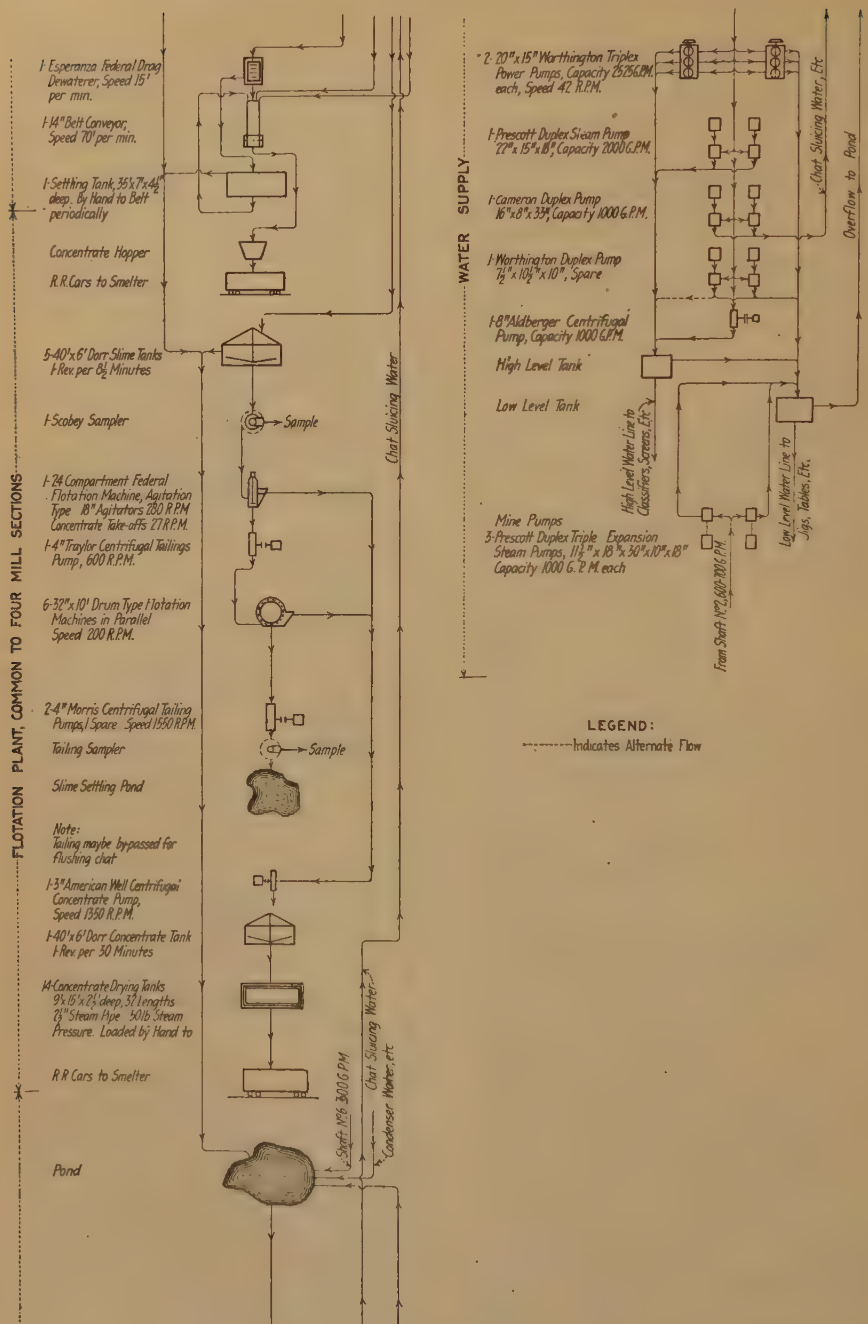

FIG. 27.—CONTINUED.

Future of the District

The "Lead Belt" is destined to operate for many years. Drilling outside the present developed district is showing the presence of more ore. Even though the grade of the ore in the future may decrease, the advance in the metallurgical practice will do much toward offsetting it.

One of the older companies[12] is reported to have made the statement that it had ore reserves to maintain the tonnage then being treated for at least 70 years. This statement may or may not be over optimistic. It does, however, give one some idea of the possible future of this district.

It is impossible to predict what changes will take place in the metallurgical practice. For some time to come jigs will retain their position in the district, and whether the advent of the fine-regrinding mill will eventually oust the jig is a prophecy that cannot be attempted with any assurance. Regrinding mills will doubtless replace the middling rolls, but much work remains to be done. The extraction in the future will be increased over that of the past, the fine-grinding mill and flotation will make possible much finer grinding than has been the practice in the district in the past. Without doubt tailing plants will be installed for the treatment of at least the current tailing, if not for the accumulated supply.

The Fredericktown District

The ores of this district are very diversified. The galena-dolomite ore presents no particular problem, the method of treatment being similar to that used for the ores of the Flat River district. There are two properties working galena-dolomite ore.

There are two properties that have a complex lead-copper-nickel-iron-dolomite ore. The method of treatment of this type of ore has not yet been solved. At present no mill is operating on this ore, although one plant is now being built for the treatment of it. At another property, the treatment of this type of ore is being studied. It will demand finer crushing than the galena-dolomite ore. The galena can largely be separated by tabling, the slime being treated by flotation. The real problem is the separation of the copper, nickel, cobalt and iron. As the iron is largely present as pyrite, it can be largely eliminated by the aid of flotation. This then leaves the copper, nickel and cobalt to be separated. Gravity methods do not yield satisfactory results, although a differentiation takes place between the copper and the cobalt-nickel. The final separation will probably demand leaching or furnace work.

At Mine La Motte, which is a metallurgical Paradise, so far as prob-

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[12] C. E. Siebenthal: Lead and Zinc Resources of the United States. *Proceedings, American Mining Congress* (1916), **19**, 401.

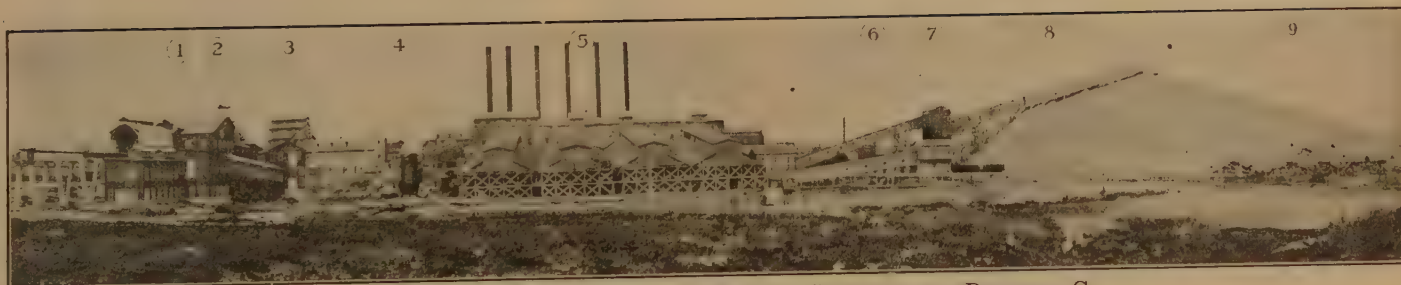

FIG. 28.—ST. FRANCOIS PLANT, ST. LOUIS SMELTING AND REFINING CO.

1. Coarse Crushing Plant.
2. No. 2 Shaft.
3. Roll and Sampling House.
4. Pump House (Foreground).
5. Mill.
6. Concentrate Conveyor (Foreground).
7. Chat Conveyor with Sampling House (Background).
8. Settling Pond.
9. Flotation Plant.

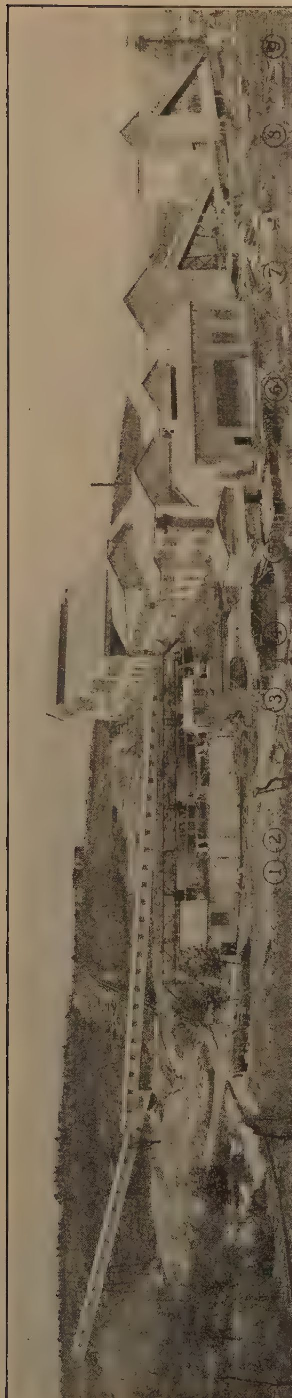

FIG. 29.—No. 4 MILL OF FEDERAL LEAD CO., ELVINS, MO.

1. Water Storage Tanks (Background).
2. Settling Tank (Foreground).
3. Mill.
4. Flotation Plant.
5. Flotation Concentrate Drying Building.
6. Machine Shop.
7. Fine Crushing Plant.
8. Coarse Crushing Plant.
9. Shaft No. 12.

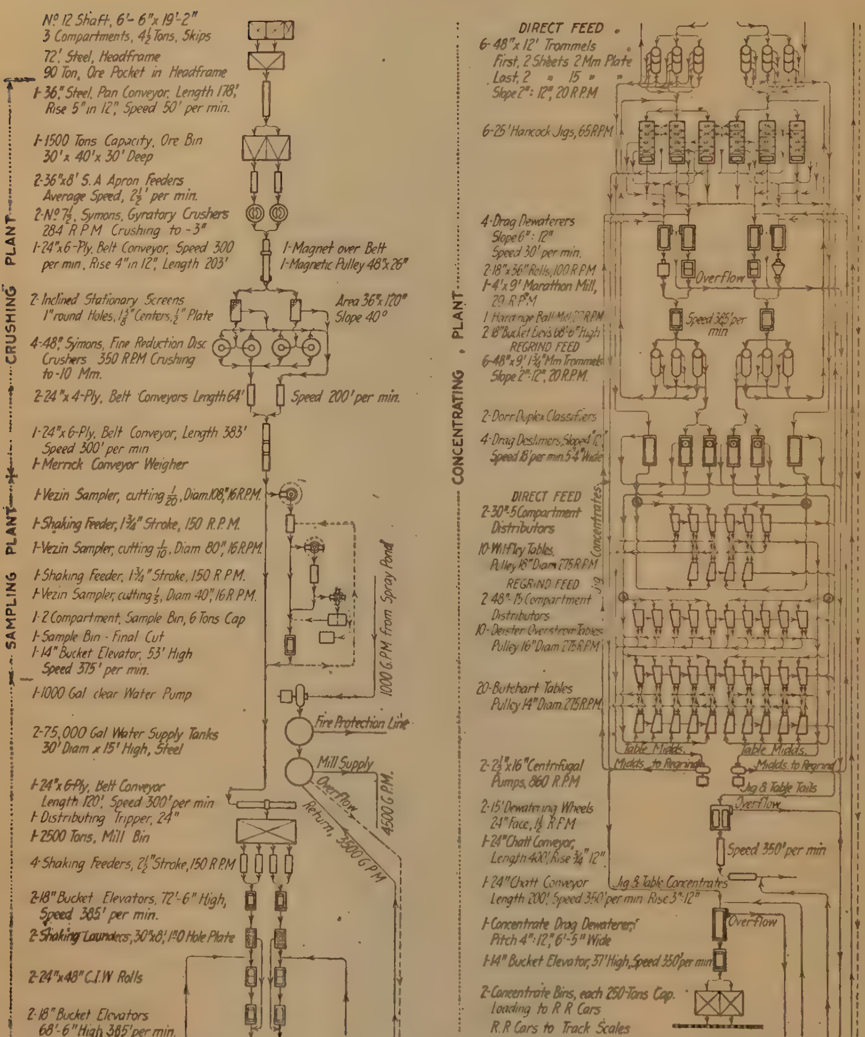

Fig. 30.—Flow sheet used at no. 4 mill of Federal Lead Co., Elvins, Mo.

lems are concerned, in addition to having the above ores, there are several others. At this property chalcopyrite occurs in sandstone and is an ideal concentrating ore. This ore yields no middling, as crushing to the size of the individual grain of quartz (0.417 mm.) serves to free all the copper, which is present as a cement holding the silica grains together.

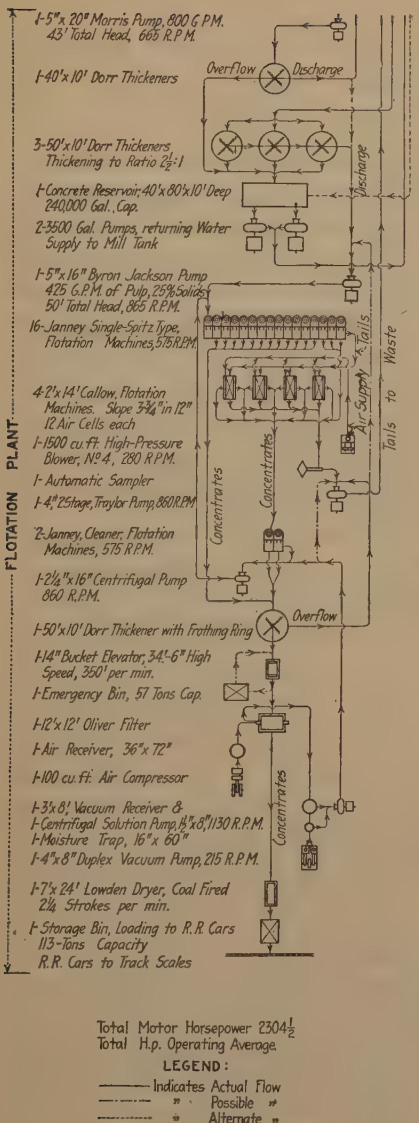

FIG. 30.—CONTINUED.

There is also a cobalt-nickel sandstone ore, which is similar to the copper-sandstone ore. Probably the most interesting and unusual problem is the treatment of the lead carbonate in the sandstone. The sandstone occurs on the surface and is in many cases disintegrated into soil. Cerusite and galena are scattered through the soil in a large number of forms and arrangements. The deposit of this sandstone is enormous and the fact that it lies on the surface makes it possible to mine it with steam shovels. At the mill, the product, which consists of everything from tree stumps to slime, is washed from the cars by hydraulic giants, disintegrated, passed over guard screens, sized and in part handpicked, the material finer than 1½ in. being concentrated on jigs and tables.

Acknowledgments

I wish to take this opportunity to thank all who have assisted in the preparation of this paper, both those who have offered suggestions and those who have so kindly furnished information.

Discussion

The Chairman (O. M. Bilharz, Miami, Okla.).—Mr. Watt's reference to sorting or hand-picking of the ore in the lead district, which is being introduced at Mine La Motte, is especially interesting. I believe, from my former experience in the district, that hand-picking should be practised to some extent by all the companies in the Lead Belt. I know that a great many thousand tons containing only very

small amounts of lead are passed through the mills, which, of course, is not to the advantage of milling machinery. That procedure may be warranted at times of high prices, but I think that hand-picking should ordinarily be adopted to some extent.

Ernest Gayford, Salt Lake City, Utah.—On page 372, speaking of the settling of slime previous to treatment either on slime tables or by flotation, Mr. Watt states: "The settling area allowed per ton of dry slime varies from 11 to 16 sq. ft.—an average for the district would be about 13 sq. ft. of settling area per ton of dry slime. This figure applies when the discharge contains 20 per cent. solids." Mr. Ramsey, of the Dorr company, is here and I would like to ask him whether a settling area of 13 sq. ft. per ton of dry slime per 24 hr. is not considerably above the average area of Dorr settling tanks, and whether 20 per cent. solids is not above the general dilution for that capacity? I should also like to know what is the general average on slimes of that character

E. R. Ramsey, Denver, Colo.—An allowance of 13 sq. ft. per ton does seem rather large. I am not familiar with the settling characteristics of the dolomitic gangue, but as an average over the country, I should say that 6 or 7 sq. ft. would be nearer correct for discharge at that consistency.

L. A. Delano, Bonne Terre, Mo.—We have had an opportunity to try our tanks at the highest capacity, and find that by thickening the slime to 30 per cent. solids we average 12 sq. ft. per ton of dry slime.

Ernest Gayford.—Do you find any definite ratio between the thickness of the discharge and the settling capacity of the tank? Do you consider that the tank has a larger capacity if the discharge is 3 to 1 than if it is 50 per cent. solids?

L. A. Delano.—I think it has greater capacity with the thinner discharge. It depends a good deal on the density of feed to the tanks; in our case the feed contains 2.5 per cent. solids.

Ernest Gayford.—Do you think that the thickness of the discharge has anything to do with its rate of passage?

L. A. Delano.—Yes, I do. If you attempted to thicken to 100 per cent. solids, you would obtain very little capacity, but as density diminishes, capacity will be enlarged.

H. S. Coe, Mound City, Kans.—The rate of overflow that can be taken off as clear solution depends on the rate of settlement of the particular slime. If a tank is being operated for maximum overflow, it would seem possible to withdraw more slime, although in more diluted condition, than if the tank were being operated so as to yield the thickest possible slime. By increasing the thickness of the settled slime, you

necessarily have to discharge more of the pulp as overflow; then, to maintain a given clarity of this increased overflow, a higher settling rate becomes necessary, but since the settling rate with a given feed is limited, it seems unavoidable to reduce capacity as the thickness of the discharged slime is increased.

Ernest Gayford.—We did not find that our Dorr tanks, giving a 3 to 1 consistency of discharge, had an appreciably larger capacity than when they were called upon to deliver at a consistency of 1½ to 1. With that particular slime, a consistency of 1½ to 1 was the limiting thickness, and the consistency of the discharge had no appreciable effect on capacity. The consistency of the feed would vary from 3 to 1 to 3½ to 1.

H. S. Coe.—It would appear to me that in settling 3 to 1, you took off 5 parts of water, while in settling 1½ to 1 you took off 6½ parts. That is only a small difference, but it is that small factor which determines how much more water has to flow over the top of the tank; if the feed is rather thin, the difference will not be appreciated.

R. B. T. Kiliani, New York, N. Y.—Milling practice in Southeast Missouri presents a particular crushing problem, a product being desired which shall contain a minimum of material finer than 65-mesh. For this work, no ball-mill can compete with the Marathon mill, the action of which is somewhat similar to that of a set of rolls. I do not believe that the Marathon mill can be used successfully for the re-crushing of iron middling, as mentioned in Mr. Watt's paper, because that has to be crushed to 65 or 80-mesh. I know of one case where a Marathon mill has been tried in a closed circuit for re-crushing for flotation; apparently it was impossible for the Marathon mill to crush fine, owing to the immense load of circulating material.

L. A. Delano.—We have used two types of fine crushers in our plant, the Hardinge mill and the Allis-Chalmers ball granulator. In the Hardinge mill, a feed ranging from 2 to 9 mm. is crushed to pass a 3-mm. screen. The Hardinge mill produces a little more slime (below 150-mesh) than the Allis-Chalmers granulator; consequently the − 150-mesh portion of the Allis-Chalmers product contains a smaller percentage of lead than the similar product of the Hardinge mill, and that is the point we are trying to gain. On the other hand, if we run the Hardinge mill with almost the same percentage of water as the Allis-Chalmers, we get the same amount of slime but containing a little more lead.

Chairman Bilharz.—What is the experience, Mr. Watt, of the St. Louis Smelting and Refining Co. with the Marathon mill for fine crushing?

A. P. Watt.—We have done no work on what you would call fine crushing in the usually accepted sense of the word. In the Lead Belt,

the re-crushing problem is peculiar in that the necessary reduction is small. At 0.833 mm. (Tyler standard), we can make an economic tailing, and at about 0.208 mm. (65-mesh) practically all of the recoverable lead is free. The two limits thus set for us are to crush to 0.833 mm. while making a minimum that will pass 0.208 mm.

At one of our plants, after analyzing the situation, we put in a Marathon mill, feeling that the Marathon was superior to the ball-mill on general principles. The ball-mill, as I look at it, operates on the hit-and-miss principle, while the Marathon works on the selective principle. Screen analyses in my original paper bear out this statement. On page 358 are shown comparative results of a Marathon mill and a ball-mill. The feed to the Marathon is slightly coarser than to the ball-mill. The diagram indicates that the Marathon discharges a maximum tonnage in the coarse sizes, while in the fine sands the tonnage falls off very rapidly. There is no oversize, the largest particle being about 2 mm., while the discharge from the ball-mill contains particles practically as large as the feed. In that test, the power consumed by the two mills was almost identical, the ball-mill crushing about 250 tons, the Marathon about 350 tons per day. From Table 4 of Mr. Delano's paper (p. 427) relating to the Hardinge mill, and Table 6 of my paper (p. 359) recording the operation of the Marathon mill, it will be seen that the feed to the Hardinge mill contained 65 per cent. of material coarser than 10-mesh, as compared with 64 per cent. in the Marathon feed; the two feeds were therefore practically identical. Examining the products from the two mills, the same tables show that the ball-mill puts only 34 per cent. of the whole between 10 and 65-mesh, the limits between which we desire to have our maximum tonnage, while the Marathon puts 60 per cent. between these limits, while also crushing a much larger tonnage.

I would like to call attention to the screen analyses in Table 7 on page 365, and the curves shown in Fig. 18 on page 366, which are plotted from the data in Table 7, showing the progressive work of crushing by a Marathon mill. It will be seen that the work of crushing is selective, the crushing being done on the largest size first, as would be expected when one considers the principle of operation of the Marathon. It would be very interesting to compare screen analyses of samples taken at every foot in a ball-mill with corresponding samples from the Marathon mill.

Ernest Gayford.—I would like to ask for the relative steel consumption in crushing, as between rods and balls, including waste of rods and balls too small for further use.

R. B. T. Kiliani.—Mr. Watt told me that the steel consumption of the Marathon was about 0.25 lb. per ton, and in the ball-mill, about 0.9

lb. I do not know whether that includes wastage or is only the actual consumption in crushing, but I think that the consumption of 0.9 lb. in a ball-mill is rather high for the slight degree of reduction indicated; it may be due to a dilute pulp or too high a speed.

L. A. Delano.—The ball consumption just quoted by Mr. Kiliani relates to cast-iron balls; where steel balls are used, the consumption is about the same as for steel rods in the Marathon mill, about 0.25 lb. per ton crushed.

A. P. Watt.—In testing our Marathon mill we ran through about 10,000 tons, accurately measured, weighing the rods before and after; the steel consumption was 0.25 lb. Of course, rods must be discarded when they are worn down to about 0.5 in., but as I recall the amount of steel discarded as small-sized rods, it was only 5 per cent. of the original weight; hence the consumption was probably under 0.3 lb. per ton, crushing 10-mm. material through a 2-mm. screen.

R. B. T. Kiliani.—I would like to ask Mr. Watt whether he has any information on the consumption of lining in ball and Marathon mills; also whether he has had trouble with rods kinking and tying themselves around the whole load of rods? I saw a case recently where two rods had tied themselves completely around the others in a Marathon mill, and three pieces had wound themselves into a tightly coiled spring, like a watch spring. Another rod about 6 ft. long had coiled itself into a spring about 6 in. in diameter.

A. P. Watt.—I have no exact information as to the consumption of lining because we never took the lining out, but after about 4 months' operation of the Marathon mill, treating approximately 400 tons a day, the wear of the lining appeared to be very small, possibly ⅛ in. As to the tying of rods into knots, they will do that if they are not taken out in time. We had trouble from that source when we first started the Marathon mill, because the men threw the rods in without placing them parallel. They all had to be taken out, but since then the rods have always been placed parallel, and for nearly a year we have never had any more trouble along that line. The rods would be more likely to cause trouble if they were worn to less than ½ in., but we have never allowed them to reach that size. Previously all our new rods were 1¼ in., but with the new mill that we are putting in, supported on trunnions, we shall use 2-in. rods and expect to reduce any trouble from bending.

Chairman Bilharz.—Both Mr. Watt's and Mr. Delano's papers contain considerable information about flotation, which is an interesting subject in connection with mill practice in Southeast Missouri. I would like to ask Mr. Delano what results have been obtained in the filtering or settling of flotation concentrates.

L. A. Delano.—The concentrates produced by the flotation machines are broken up as much as possible by clear water sprays and lifted by a bucket elevator to a 38-ft. Dorr thickener, from which the concentrates discharge from the spigot to an Oliver filter, while the overflow passes to the slime tanks to avoid any loss. A permanent froth remains on top of the Dorr thickener, and is very difficult to break up. At first we had trouble to keep this froth from running over the baffle ring which surrounds the outside of the tank; but as we gained experience, we found that by keeping the tank below capacity, not allowing it to overflow, the froth gave us practically no difficulty.

The spigot discharge of this Dorr thickener has been as low as 32 per cent. moisture, and has averaged 40 per cent. moisture in a year's run. The cake scraped from the filter averages 12 to 16 per cent. moisture, depending on the condition of the canvas. We found that the canvas becomes blinded or caked in a few weeks' use, the average life being about 3 months for canvas of good quality. On most of our work we have used No. 3 Oakdale twilled cloth canvas, which gives very satisfactory results. The filter cake, averaging 15 per cent. moisture throughout the year, has to be further dried before going to the smelter. Our first drying was by steam coils and vats. This method was rather expensive and is being replaced by mechanical dryers, reducing the moisture to 7 to 8 per cent., which is satisfactory to the smelters. The filtrate from the Oliver filter is clear, and we find that with a 12-ft. Oliver filter we have been able to handle 75 to 80 tons of flotation concentrate a day.

For the disposal of tailings, we tried one experiment with a 3-ft. filter. This reduced moisture to 10 or 11 per cent., but would prove a costly process, and we decided that it would be better to pump the tailings into a pond. We secured an area and built a chat dam around it with a maximum height of about 50 ft. The pump line is 2300 ft. long, through which we are pumping about 200 gal. of pulp per minute, containing 600 or 700 tons of slime per day. The solids settle very rapidly; at first we expected to have a large accumulation of water, but that has not proved to be the case. About three-quarters of the surface of the pond is comparatively dry, with only a little water filtering perfectly clear through the chat dam. Pumping consumes only 18 hp. against a 50-ft. head.

Chairman Bilharz.—How does the cost of filter drying compare with steam-coil drying? As I remember, the cost of steam-coil drying is between 40 and 50 c. a ton. Will you also tell us the screen analysis of the flotation concentrate?

L. A. Delano.—The cost of steam drying is really much higher than that; I am not able to give any actual figures, but I would say that filter drying is much cheaper.

A. P. Watt.—A sizing analysis of the flotation concentrate is given in my paper p. 388. It is not a screen analysis, but the results were obtained on a testing cone as used in white-lead works for the determination of very fine sizes. Dr. Thompson, of the National Lead Co., New York, made this determination. It shows that about 99 per cent. will pass 300-mesh and that 41.16 per cent. is 0.0125 mm. in size, which is extremely fine.

H. S. Coe.—Can you give us any figures as to average capacity of your dryer for flotation concentrates?

F. D. Druding, Flat River, Mo.—Our present dryer is 7 by 21 ft., and is giving considerable hearth trouble, due to buckling of the plates, which we hope to remedy by passing the gases to the head end and back underneath the hearth to the tail end. At present we are drying only 20 to 21 tons per day, reducing moisture from 15 per cent. to 5 or 6 per cent. When we widen our drier to 12 ft., we shall be able to handle as much as the filter will deliver, or at least double the present quantity.

S. A. Ionides, Denver, Colo.—At which end do you apply the heat in your Lowden drier? In the first drier of that type built at the Bunker Hill mine, the heat was applied at the wet end and it gave us no trouble.

F. D. Druding.—I am glad to know that, because we built our furnace the other way and are now remodelling it. At present the gases hit the plates directly, which gives considerable trouble; but we are now arranging to have the gases pass underneath an arch and apply the heat to the plates at the wet end.

C. T. Van Winkle, Salt Lake City, Utah (written discussion*).—Mr. Watt's article on Concentration in Southeast Missouri deals at some length with the crushing problem in that district, and after looking over this portion of his paper I believe that some comments are in order.

Mr. Watt calls attention to his "ideal" curve for a grinding-mill discharge and shows how nearly the discharge from a Marathon mill parallels this condition. An ideal condition would be one approaching perfection, and obviously this is impossible in crushing, no matter what mill is used. Mr. Watt's curve is evidently the one suggested by Sergio Bagnara of the Doe Run Lead Co., in his article in the *Engineering & Mining Journal* of July 14, 1917, as the ideal curve or the curve of maximum efficiency, shown on his chart as the "limit" curve, and this is apparently where the idea originated. If an ideal condition is to be approximated, however, the crushed ore should all pass the mesh at which the mineral is released and stay in the next finer mesh. That is, if, as stated by Mr. Watt, the mineral is all released at 65-mesh, it should pass that mesh

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* Received Dec. 3, 1917.

and remain on 100, and an ideal curve should lie somewhere between these limits. Obviously, we are not yet prepared to attain anything like this condition with present crushing machinery.

In making his comparisons between the discharges of a ball and a Marathon mill, undoubtedly the ball-mill product is that from the Hardinge or overflow type of mill. I am of the opinion that if these comparisons had been made between the Marcy or the Allis-Chalmers and the Marathon, operating under similar conditions, the "ball-mill discharge" would be somewhat different from that so termed by Mr. Watt. For this reason, he should have been more explicit in his statements regarding the type of mill used. The discharge from all ball-mills is not the same as the results in the Missouri district, and while the Marathon undoubtedly makes less −200-mesh material than the ball-mill, there is not the difference shown by the curve in Fig. 15 of Mr. Watt's paper, when compared with the grate ball-mill.

Undoubtedly, in the instance mentioned by Mr. Watt, the power consumption per ton of ore is low, but a reference to Table 6 shows a feed of 3.9 per cent. on 3-mesh and a discharge which just about passes 8-mesh. While this reduction is probably within the economic limit of their practice, the amount of work done is small and as 67 per cent. of the discharge remains on 48-mesh, it is very far from flotation requirements.

Regarding the statement that if the ball-mill feed were increased to 500 tons per day, the oversize would be greatly increased, I would call his attention to the fact that there are ball-mills, such as the Marcy, that are fitted with 1/8-in. to 3/32-in. grates giving a −8-mesh product; 800 tons would be a very moderate tonnage for one of these mills and the grates eliminate the "tramp" or oversize particles he speaks of.

In the rods of the Marathon mill, the 5/64-in. difference in size between the feed and discharge ends would seem to be due to local conditions, and is of no moment either way. According to Blickensderfer,[1] the rods at the Burro Mountain mill showed a taper in a direction toward, instead of away from the feed end. I am of the opinion that when rods are worn down to 5/8 in., or the danger size, there is no great difference in diameter at the two ends.

Thus far it has been impossible to operate a Marathon mill for any length of time without removing the small rods. These rods, if left in the mill too long, bend, kink or break and sometimes bind the whole mass of rods into a bundle, interfering seriously with the efficiency of the mill. For removal of rods it is necessary to take off the discharge head of the mill, and it would seem that this operation has been complicated by the new design of trunnion mounting, Fig. 20. This must make the removal of the head a more difficult and longer operation. The small

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[1] *Trans.* (1916), **55**, 695.

rods taken out of the mill are necessarily "scrapped" and should be taken into account in arriving at the steel consumption. Mr. Watt does not mention whether or not this is done, and without this detail or the amount of time lost by changing rods, the data are incomplete.

There is no doubt that the rod-mill which has been under consideration in the patent office and experimentally for many years is here to stay. Where a uniform product is needed, as in southeast Missouri, it is particularly adapted to the work to be done. There is nothing to indicate, however, that the Marathon is more efficient by four or five times than the low pulp-line ball-mill, as has frequently been stated in the trade journals, since no parallel experiments have been made. Blickensderfer[2] has described the Marathon mill in a very interesting way and has shown comparisons with a Hardinge pebble-mill, but when the work done by these mills is platted by mesh-ton curve the efficiency is more apparent than real. There is no doubt, however, that the Marathon is a theoretically efficient mill; the question is how far this theoretical efficiency is offset by its mechanical difficulties. In the descriptions of many grinding mills, often only the good points are given. What we need are the entire facts and what is to be expected when mills are operating under regular conditions.

A. P. Watt.—(Discussion concluded on page 1089.)

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[2] *Op. cit.*

The Milling Practice of the St. Joseph Lead Co.

BY L. A. DELANO,* E. M., BONNE TERRE, MO.

(St. Louis Meeting, October, 1917)

DURING 1916, the St. Joseph Lead Co. milled 2,505,670 tons of ore. This is a daily operating average of 7855 tons. The economic concentration of such a large tonnage necessarily requires a plant equipped with modern machinery and operated efficiently. The milling process used in producing this output has been a gradual development through the long period of years that the company has been in existence.

HISTORICAL SUMMARY OF MILLING PRACTICE

Ore was first mined at Bonne Terre, Mo., in 1864. The early milling practice was crude. It consisted of breaking the ore into small pieces by hand, pulverizing in a Blake crusher and a pair of Cornish rolls to approximately minus 6 mm., and jigging in the old-fashioned hand jigs. Only the coarse heavy galena was saved, the gangue being shoveled off and discarded. The output for the first year under this method was only 240 tons of pig lead.

The first mill of the company was a combination of extensions and additions built from time to time as the mines developed. It was a museum of early milling machinery, such as jigs, percussion tables, buddles, dolly tubs, log washers, etc., following no definite flow sheet. The chats, or tailings, were hauled to waste dumps by mule and ox teams, as much as 500 tons per day being moved in this manner, until a railroad was built in 1880. This mill was burned in 1883, at which time the annual output had reached approximately 7300 tons of pig lead.

A new mill of 500 tons capacity was designed and built by C. B. Parsons. This mill has been admirably described by H. S. Munroe.[1] The ore was crushed in Blake crushers, Cornish rolls, through 6-mm. screens, and treated by jigs of the Parsons type, percussion tables and log washers, or trunking machines, for cleaning the galena. The middlings were recrushed in rolls and treated on Harz jigs. The mill was operated without change, except later installing Gates crushers and en-

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* Mill Superintendent, St. Joseph Lead Co.

[1] *Trans.* (1888–89), **17**, 659.

larging the capacity to 1200 tons, until 1910, when Wilfley tables replaced those of the Rittinger type.

In 1904, a new mill was built at Leadwood, Mo., with a capacity of 1200 tons daily. The ore was crushed in Gates gyratory crushers and high-speed rolls, and concentrated on Harz jigs and Wilfley tables. In 1906, the Hancock jig was introduced to the district in this mill.

FLOW SHEET
BONNE TERRE MILL

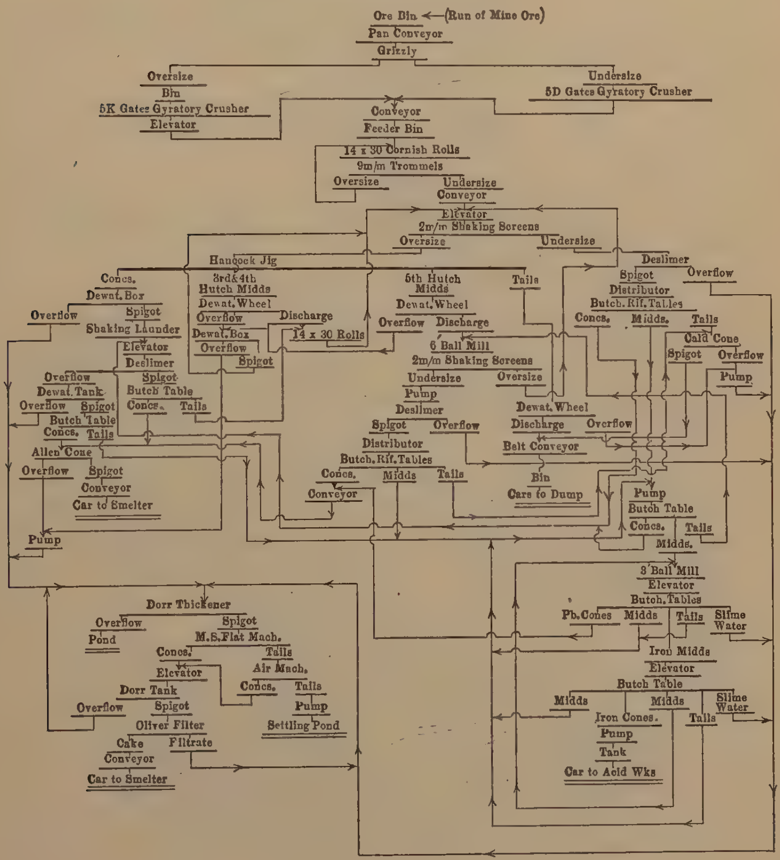

FIG. 1.

In 1909, the Doe Run Lead Co., now a subsidiary of the St. Joseph Lead Co., built a 2400-ton mill at Rivermines, Mo., which in 1913 was enlarged to handle 3600 tons per day. This plant received ore crushed to 4 in. at the mines and, after reducing to 9 mm. at the mill, concentrated it on Hancock jigs and Wilfley tables.

In 1912, the Bonne Terre mill was partially remodeled, Hancock jigs, Richards classifiers, and automatic disposal of mineral and chat were

installed, increasing the capacity of the plant to 1600 tons daily. In the Spring of 1914, the flotation process having been applied to lead ores by the Federal Lead Co., it was installed in the three mills of this company for the treatment of the slimes. The application of modern crushing machinery, jigs, tables, and flotation has resulted in the present system of milling with the ability to handle large tonnages economically.

D.R.L.CO'S #3 MILL RIVERMINES MO.
FLOW SHEET

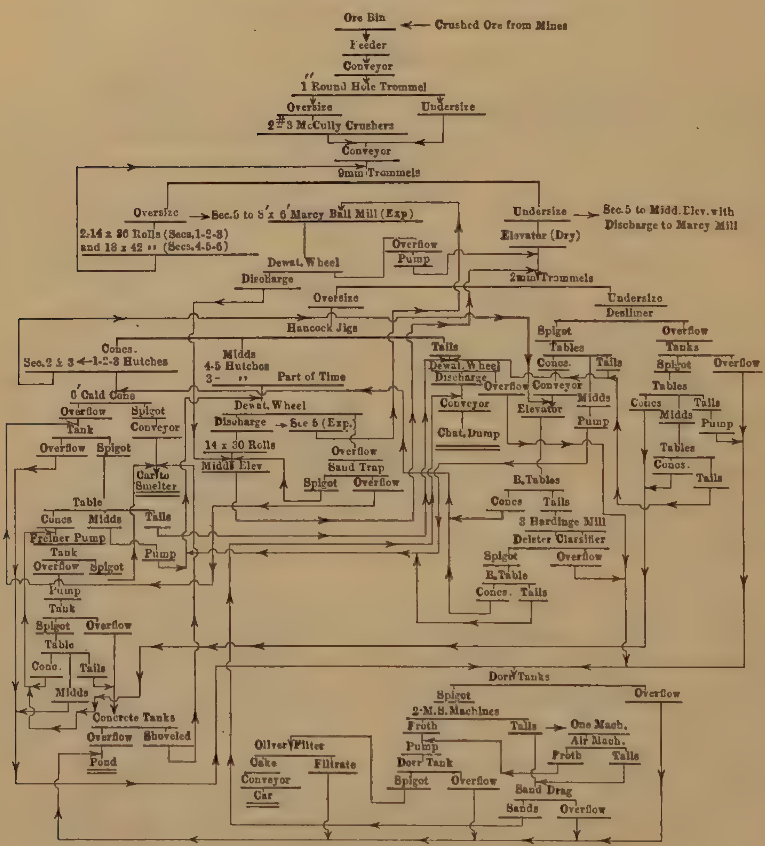

FIG. 2.

DESCRIPTION OF THE PLANTS

The ores are milled at three plants, namely, Bonne Terre, Leadwood, and Rivermines. Daily tonnages milled at each plant are: Bonne Terre, 2100; Leadwood, 1750; and Rivermines, 4000.

The mineral, galena, is disseminated in magnesian limestone or dolomite. It is associated with some pyrite, marcasite, chalcopyrite, and sphalerite, and carries a small amount of silver. The ore in part of the

mines runs high in chlorite (glauconite), and contains a rather large percentage of insolubles. The galena is massive, crystalline to fine granular in character and disseminated in the dolomite. The ore has a specific gravity of approximately 2.8, and a hardness of 3.5.

LEADWOOD MILL
FLOW SHEET

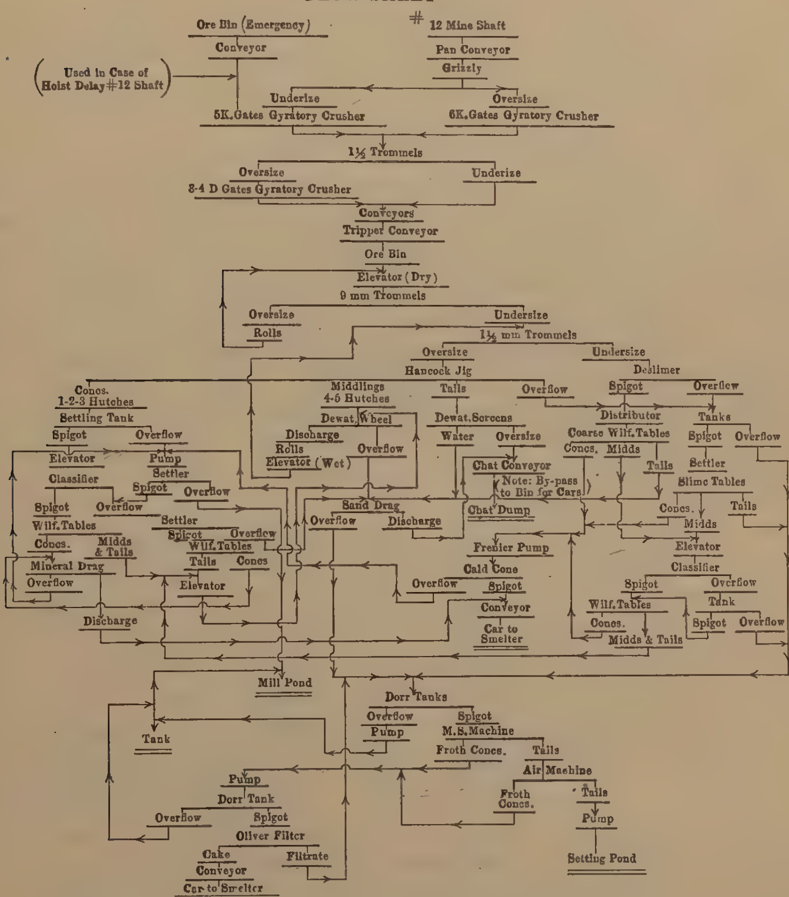

Fig. 3.

The flow sheets of the three mills are shown in Figs. 1, 2 and 3. In this paper a complete description of the Bonne Terre mill practice will be given, and then contrasted with the operation at the Leadwood and Rivermines mills, showing the principal differences in each plant.

The Bonne Terre Mill

The ore is hoisted to a bin of approximately 22 tons capacity. From this bin it is conveyed by pan conveyors to two No. 5 style D Gates and

one No. 5 style K Gates gyratory crushers. A screen analysis of run-of-mine ore feeding the crushers is shown in Table 1.

TABLE 1.—*Screen Analysis of Ore at the Bonne Terre Mill*

Size of Opening	Per Cent. Weight
On 10 inch grizzly	11.1
On 8 " "	12.6
On 6 " "	7.6
On 4 " "	32.5
Through 4 " "	36.2
Total	100.0

Grizzlies with bars spaced with 7½-in. clearance are set on an incline over the two No. 5 D crushers that crush the undersize. The oversize gravitates to a small bin and is crushed by the No. 5 K crusher, and is then elevated by bucket elevator to pan conveyors, which convey all the crushed discharge to belt conveyors. These belt conveyors distribute the ore to feeder bins, scrapers being used to rake it off into each bin. Messiter electric scales attached to the pan conveyors receiving the discharge from the crushers were used for a time, but have been discontinued because of difficulty in keeping them accurately regulated. Table 2 shows a screen analysis of the crusher discharge.

TABLE 2.—*Screen Analysis of Crusher Discharge*

Opening, Diameter	Weight, Per Cent.	Opening, Diameter	Weight, Per Cent.
+ 3 in.	12.9	+ 1 in.	14.9
+ 2½ in.	11.4	+ 9 mm.	12.5
+ 2 in.	17.6	+ 2 mm.	9.9
+ 1½ in.	15.7	+ 150 mesh	3.5
		− 150 mesh	1.6
		Total	100.0

The same Cornish rolls that were installed in the mill when it was built in 1883 are still in use. Thirteen sets are used, fed by Challenge feeders. The shells are 14 by 30 in., one chilled cast iron and the other chilled steel, being used on each set. A groove ¾ in. deep and 1½ in. wide is cut across the face of each steel roll shell to prevent slipping of the ore and consequent grooving of the shell. The roll speed is 10 r.p.m. Weight boxes are used on the rolls instead of springs. The roll discharge is screened through a 9-mm. trommel, 3 ft. (0.91 m.) diameter by 8 ft. (2.5 m.) long, with a slope of 1 in. (25.4 mm.) per foot. The trommel revolves at a speed of 36 r.p.m. The oversize is returned by a bucket

elevator to each roll. The rolls average 185 tons per 24 hr., and have shown by test that 250 tons per 24 hr. can be maintained. The rolls are driven by two 75-hp. and one 100-hp. motors with a total load of 210 hp., or an average of 16 hp. per roll, including the oversize return elevator.

The Concentration System

The mill is divided into four sections after crushing the ore, an equal tonnage being treated in each section. The mill building is built on a level site and it is necessary to elevate and pump the various products for treatment. All line shafts are in the basement with the rolls and other heavy machinery on concrete foundations near the floor level. The undersize from the 9-mm. trommels is conveyed to elevators and water added to sluice the ore into the boots. The elevators are 53-ft. (16-m.) centers, using 8 by 16-in. (20 by 40-cm.) buckets with a belt speed of 400 ft. (121.9 m.) per minute. The discharge from the elevators is screened on 2-mm. screens.

The screens are of the Ferraris shaking type, first used by the Desloge Lead Co. in this district, but have been re-designed at Bonne Terre to withstand the service required and to fit the limited head room. The screens are made at the plant, and cost about $125 each, erected. Three screens are used at each elevator, driven separately from a line shaft, so that repairs can be made, or screen sheets changed, while the other two carry the load. The screen frame is supported by hickory hangers set at an angle of 65° to the frame. The drive-rod is connected to the head of the screen and a jig eccentric on a countershaft over the screen supports. The eccentric is set for 1¼-in. stroke and a speed of 270 to 290 r.p.m. A clear-water spray is used over the screen near the center to wash the fines from the oversize. Various screen openings have been tried, but at present 2-mm. round holes in No. 18 gage sheets are used, the sheets lasting about 40 days. Each screen treats 250 to 300 tons per 24 hr. and requires 0.75 hp.

The Jig Installation

The Hancock jigs treat the oversize of the 2-mm. screens. They are divided into five hutches and a tail compartment. The first and second hutches produce concentrates, sent to the re-cleaning system. The third hutch is arranged for sending its product either to the concentrates or the middlings. The fourth and fifth hutches produce middlings. The cam shaft has a speed of 65 r.p.m., giving the jig 195 strokes per minute. The horizontal throw averages ¾ in. (19.05 mm.) and the vertical throw about ⅝ in. (15.875 m.). The tailings are dewatered by drag scrapers in the last compartment in order to separate them from the slimy water

that overflows the jig and goes to the Dorr thickeners. The jig tailings are then sluiced to concrete tanks and dewatered by shovel wheels, the discharge going to a belt conveyor and concrete bin. These shovel wheels are 14 ft. (4.3 m.) in diameter, and make 1 r.p.m., the discharge averaging 15 per cent. moisture.

The third and fourth hutch middlings are dewatered by a shovel wheel and recrushed by rolls, returning to the jig elevator and 2-mm. screens. These middlings are crushed to an average of 3 mm., releasing the galena in as coarse a size as possible. The fifth hutch middlings, averaging from 1.5 to 3.0 per cent. Pb, are dewatered by a shovel wheel and sent to a 6-ft. ball mill, of which there are two types in the plant.

The Ball Mills in Use

A 6-ft. by 22-in. Hardinge mill crushes the middlings in section 1, and a 6 by 4 ft. Allis-Chalmers granulator is installed in section 3. They are being tested for results before placing mills in the other two sections. As final results had not been worked out on either type of mill at the time of writing, it is possible to give only some of the experimental data at present, showing the grinding efficiency and tonnages treated by each mill.

Results with the Hardinge mill are shown in Table 3.

TABLE 3.—*Data on Hardinge Mill Operation*

Average tonnage ground..	250 tons per 24 hr. (this includes 50 tons of middlings returned from table treatment).
Ball load................	14,000 lb.
Kind of balls............	Cast iron, with 3 in. maximum diameter.
Power required..........	53 hp.
Speed of mill............	30 r.p.m.
Moisture	50 per cent.
Ball consumption........	0.9 lb. per ton crushed.

The screen analysis in Table 4 shows the crushing performed on this class of material.

The discharge from the mill is screened through a 2-mm. shaking screen. The oversize returns to the jig elevator as jig feed, and the undersize is pumped to a deslimer for table and flotation treatment.

Various tests have been tried in crushing by the Hardinge mill, *e.g.*, crushing the entire middlings, returning the oversize to the jig and the undersize to tables and flotation. This method enriched the flotation feed too much, due to the rich middlings from the third and fourth hutches of the jig. Another test was to crush the entire tailings of the jig, the middlings being crushed in the rolls and returning to the elevator, and the oversize of tailings from the ball mill being discarded as chat. The

ball mill lacked capacity to treat the necessary tonnage in this manner, and it was found that the most economical method was to crush the middlings from the fifth hutch of the jig. This gave low tailings on the jig and low-grade slime for flotation, besides allowing a larger tonnage of ore to be treated in the mill section.

TABLE 4.—*Screen Analysis of Hardinge Mill Material*

Feed			Product		
Mesh	Per Cent., Weight	Cumulative, Per Cent., Weight	Mesh	Per Cent., Weight	Cumulative, Per Cent., Weight
On 3	5.34	5.34			
On 4	15.00	20.34	On 4	1.1	1.1
On 6	12.45	32.79	On 6	1.3	2.4
On 8	17.45	50.24	On 8	2.0	4.4
On 10	15.60	65.84	On 10	5.8	10.2
On 14	10.27	76.11	On 14	6.4	16.6
On 20	4.49	80.60	On 20	6.6	23.2
On 28	2.29	82.89	On 28	7.8	31.0
On 35	1.52	84.41	On 35	6.9	37.9
On 48	1.95	86.36	On 48	6.6	44.5
On 65	2.46	88.82	On 65	5.8	50.3
On 100	3.98	92.80	On 100	7.8	58.1
On 150	5.25	98.05	On 150	8.4	66.5
On 200	1.10	99.15	On 200	8.8	75.3
Through 200	0.85	100.00	Through 200	24.7	100.0
Total..........	100.00			100.00	

The Allis-Chalmers Granulator

This mill has a grate through which the crushed product must pass before being discharged. As with the Hardinge mill, various tests are being made to determine its crushing capacity, efficiency, etc. The first tests were to crush the jig tails, cast-iron balls with 3-in. (76.2 mm.) maximum diameter being used as grinding mediums. Ball loads ranged from 9000 lb. to 14,000 lb. (4082 to 6350 kg.), but the results were not satisfactory, the tonnage building up in the mill and choking it. Raising the pulp level improved the grinding slightly. It was found, however, that the balls broke up badly, so a change was made to forged chrome-steel balls of larger diameter. Tests for ball charges, load and pulp level have resulted in gradually raising the tonnage of feed and grinding efficiency. The results shown in Table 5 are the best that have been obtained as yet, further tests being made with a closed circuit.

Table 5.—*Allis-Chalmers Ball Granulator Results*

Character of feed	Jig tails and fifth-hutch middlings.
Ball load	12,000 lb. (consisting of 1000 lb. of 5 in. diam., 3000 lb. of 4 in. diam., 8000 lb. of 3 in. diam.).
Revolutions per minute	23
Grate opening	3/16 in.
Pulp-level distance from center	16 in.
Tons feed per 24 hr	486
Per cent. moisture	62.6
Horsepower	59.4
Tons crushed through 2 mm. per horsepower	6.7

Oversize of 3-mm. screen discarded as chat.

Screen Analysis of Product

Mesh	Per Cent., Weight
On 10	18.7
On 150	62.7
Through 150	18.6
Total	100.0

The discharge from the mill for a while was screened through a 2-mm. screen, the oversize being discarded as chat. Later a 3-mm. screen was substituted, resulting in the table tailings averaging practically the same as with the smaller opening. The oversize of this screen was also discarded, but is now being returned to the jig feed, making a closed circuit on the ball mill. Results from this at present are not complete.

Summary of Ball Mill Practice

It is impossible now to give final results metallurgically as to the effect of ball mills on the milling practice of the ores. However, an increased capacity for the plant is obtained when the middlings are taken from their former closed circuit with the jig and recrushed in ball mills. The undersize of the ball-mill discharge is deslimed, the sands going to separate tables. This gives the jig capacity for more original ore, and the plant now handles 2300 to 2400 tons per day. This alone has justified the installation of the ball mill. On the other hand, rolls cannot grind the low-grade middling as well as the ball mill, the latter giving more material for tables and flotation, which result in lower tailings than the jigs. The regrinding of jig tailings has resulted in lowering the final tailings. Approximately 60 per cent. of the feed to the Allis-Chalmers ball mill is sent to tables, the tailings of which average less than 0.4 per cent. Pb. The flotation plant receives 18 per cent. of the feed and will make minimum tailings there. The tests have not been completed to show the economic effect of lowering the mill tailings, but the results point toward a profit, even with a low-price lead.

Table Concentration

In the summer of 1915, it was decided by the management to give the Butchart riffling a trial in table practice, in order to increase the tonnage and efficiency of the mills. A series of tests was carried out for several months at the Bonne Terre mill to determine the best application of the riffle to the treatment of these ores.

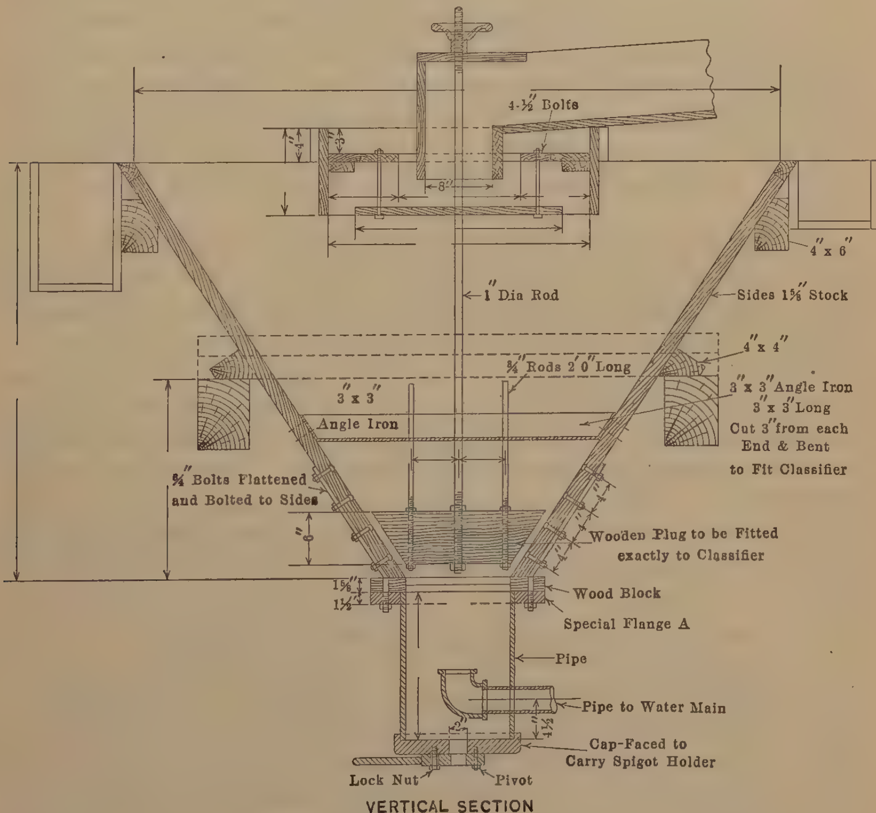

FIG. 4.—THE DELANO DESLIMER.

Previous to this time, Wilfley tables and riffling had been used, the feed to the tables being the spigot products of Richards four-spigot, vortex classifiers. The Butchart riffling worked best, however, with deslimed sands, no separation into different sizes being required. As it became necessary to mix the spigot products of the classifiers, it was decided to change the classification system.

Experiments were conducted to produce a one-spigot classifier that would handle the entire tonnage of the undersize from the 2-mm. shaking screens and deslime it so as to give a table feed practically free from the

minus 150-mesh slimes. The experiments resulted in a one-spigot, two-product, hydraulic classifier, or deslimer, with an inverted conical or pyramidal tank, the entire perimeter being used for the overflow into a surrounding launder at the upper edge. The deslimer is provided with a central feed box, having a baffled bottom to break the velocity of the inflowing pulp. The bottom, or discharge chamber, has a clear-water inlet, which directs the flow upward along the axis of the deslimer. The spigot is inserted in a movable holder, attached to the bottom of the discharge chamber, permitting an instant change in the diameter of opening without wetting the operator. Between the feed box and discharge chamber is a valve with sides parallel to the tank and suspended on the vertical center, and which can be raised or lowered by means of a hand wheel on the suspension rod. This produces a narrow channel between the valve and tank through which the hydraulic water rises and the sands fall.

The feed entering the deslimer is checked by the baffle, causing the sands to settle through a narrow, rising current of water surrounding the valve. This current increases in velocity until the discharge chamber is reached, by which time the sands are thoroughly deslimed. A hydraulic pressure of 5 or 6 lb. per square inch is maintained in the discharge chamber. The width of the sorting column may be varied by raising or lowering the valve. No whirling or boiling currents are possible and the steady rising current gives a very close separation. The great efficiency of the deslimer is due to the better application and direction of hydraulic water supplied. A deslimer 5 ft. 6 in. in diameter treats from 350 to 400 tons per 24 hr., using approximately 120 gal. of hydraulic water per minute.

Table 6.—*Screen Analyses of Deslimer Material*

Mesh	Feed Per Cent. Weight	Spigot, Per Cent. Weight	Overflow, Per Cent. Weight
On 100	64.70	86.8	0.69
On 150	7.41	8.2	5.16
On 200	4.10	2.3	9.32
Through 200	23.79	2.7	84.83
Total	100.00	100.00	100.00

Experiments with the Butchart riffling were carried out with the following objects in view, viz., to determine size and character of feed, to increase the table efficiency, increase its capacity, simplify the flow sheet, reduce water consumption, power and labor. Its ability to treat unclassified material gave it advantages over the Wilfley riffling by eliminat-

ing the multiple-spigot classifier. The concentrates are discharged at the upper edge of the table, the coarser particles being in the upper riffles instead of mixing with the middlings as in the Wilfley riffling. This gives a larger capacity for both the concentrates and middlings separation and enables much larger tonnages to be treated per table.

Tests were made with feed ranging from minus 9 mm., or jig feed, to the finest sands. In each case the riffling was made especially for the size of feed treated. The separation of free galena as concentrates in each case was satisfactory, but in the coarser sizes the separation of the middlings from the gangue was imperfect and would have necessitated recrushing the entire tailings for re-treatment. Final tests showed that the jigs gave a better separation of tailings and middlings on material from 9 to 2 mm., while the undersize of 2-mm. screens, deslimed to remove the minus 150-mesh slimes, and treated on tables, gave satisfactory results.

With this size of feed the tables now treat from 50 to 60 tons per day, giving clean concentrates and as low tailings as with the Wilfley when treating 20 to 25 tons per day. In operation the table requires very little adjustment, even with the load varying 5 or 10 tons. The riffling used in this work tapers from $\frac{7}{16}$ in. (11.1 mm.) to $\frac{1}{8}$ in. (3.2 mm.), with a slope of 1 in 4 in the cleaning zone. The middlings average approximately 10 per cent. of the feed to the tables. This is re-treated on a separate table for two mill sections, the tailings being reground in the 6-ft. ball mill. A concentrate cut is made from this table and the middlings are reground in a 3-ft. Hardinge mill and sent to another circuit. In changing the mill from Wilfley to Butchart system, one Butchart table replaced three Wilfleys, and enabled a saving in power, labor and water. Various pumps, launders, classifiers and settling tanks were cut out, making a much simpler flow sheet.

The possibility of using the Butchart riffling to raise the grade of the mill concentrates was suggested by its large capacity and its ability to treat coarse material, discharging the coarsest particles at the upper edge of the table. The jig concentrates were erratic, ranging from 60 to 70 per cent. Pb and occasionally going lower. Before this they had been washed in a log washer or trunking machine, which made a rather poor separation. Tests with the Butchart riffling showed that the feed should be deslimed and treated on two tables. The spigot of the deslimer or plus 100-mesh material was treated on one table, and the overflow, or minus 100-mesh material, was settled in a tank, the spigot feeding the second table.

Table concentrates were mixed with the jig concentrates to produce a mixture of sizes that would bed well on the table. The riffling on each table was especially arranged for the size and character of the feed. The coarse galena was discharged in the upper riffles, followed by the finer

sizes. A good separation was made in the cleaning zone, the tailings constituting true middlings containing practically no free galena. They averaged 25 to 35 per cent., and are recrushed in the jig-middling rolls, returning to the circuit by the jig elevator. The coarse table now treats from 70 to 100 tons per day and has capacity for a larger range in tonnage.

The fine-table concentrates averaged 80 per cent. Pb, 95 per cent. passing a 200-mesh screen. The tailings average 6 per cent. Pb, and are returned to the sand middlings table.

At present the entire jig concentrates and a portion of those from the tables are treated in the re-cleaning system. The grade of the mill concentrates has been raised from an average of 68 per cent. to 73.5 per cent. resulting in smaller freight and smelter treatment charges, and without affecting the mill recovery.

Concentrates Disposal

The concentrates from the entire mill are sent to a 6-ft. Allen cone for dewatering. The spigot discharges on a conveyor belt to the car, while the overflow returns to the mineral elevator.

The jig-tailings disposal was referred to in describing the jigging. The tailings from the tables are dewatered by Caldecott cones, the sands discharging through the spigot to the chat conveyor. The cones will average from 400 to 500 tons of minus 2-mm. sands per day, the average moisture in spigot being 28 per cent. A screen analysis of the spigot discharge is shown in Table 7.

Table 7.—*Caldecott Cone Spigot Discharge*

Mesh	Per Cent., Weight	Mesh	Per Cent. Weight
+ 10	2.59	+ 65	7.27
+ 14	9.61	+ 100	6.65
+ 20	17.64	+ 150	5.18
+ 28	22.95	+ 200	1.25
+ 35	15.13	− 200	0.86
+ 48	10.87		
		Total	100.00

The sands from these cones, and the shovel-wheel discharge, or jig tailings, are conveyed by a 20-in. (0.5-m.) conveyor to a concrete chat bin with a capacity of 250 tons. Railroad cars of the A-dump type run under this bin, and are loaded by slide gates in the bottom of the bin, and sent to the chat dump. A portion is sold for railroad ballasting and concrete work.

The Pyrites Concentrates

The ore assays approximately 5 per cent. Fe, consisting mainly of pyrite, marcasite and chalcopyrite. This is concentrated in the table

middlings and requires fine grinding to separate the galena, pyrite and dolomite. The middlings from the sand tables are reconcentrated on a middling table and the middlings from this table are sent to a 3-ft. Hardinge mill for regrinding. The mill is operated with a 1000-lb. ball load, replenished with 2-in. cast-iron balls. The product will nearly all pass a 65-mesh screen, and is elevated to a Butchart table. Lead concentrates, assaying 80 per cent., are sent to the Allen cone and middlings are retreated on another Butchart table, which produces galena-pyrites concentrates. The analysis of concentrates shows an average of 41 per cent. sulphur, a portion being shipped to a sulphuric-acid plant for experimental purposes, with a view to recovering the sulphur as acid. The remainder is mixed with the galena concentrates and is shipped to the St. Joseph Lead Co.'s smeltery at Herculaneum for its value as an iron flux.

Treatment of Slimes

The overflow of the deslimers, chat-wheel pits, Caldecott cones and various small settling boxes, is thickened in five Dorr tanks, four of which are 40 by 8 ft. (12 by 2.5 m.), and one 50 by 6 ft. (15 by 1.8 m.). The feed to these tanks averages 2.5 per cent. solids, the total flow being approximately 3500 gal. per minute. The solids average 540 tons per 24 hr., or approximately 12 sq. ft. (1 sq. m.) of settling area per ton of dry slime. The central shafts revolve once in 10 min. The overflow from the tanks returns to the mill pond for use as mill wash water, while the spigot discharge is sent to the flotation machine.

The thickened pulp at a ratio of 2.5 to 3.0 : 1 is treated in a 23-cell Minerals Separation flotation machine, with 24-in. agitating compartments. Creosote oil (hardwood) is added in the feed box and at several alternate cells, a small oil pump being used to give a regular oil feed. The impellers are belt driven from a line shaft, the speed being 335 r.p.m.

The machine is driven by a 100-hp. motor using 83 hp., or an average of 3.8 hp. per cell. The tailings from this machine are retreated in a pneumatic machine. The froth, or concentrates, averaging 50 per cent. Pb from both machines, are elevated by a bucket elevator to a Dorr thickener.

Oil consumption averages 0.55 lb. per ton of dry slimes treated. The slimes average 540 tons per day, or, at present, 23 per cent. of the ore milled. This percentage is higher than in the other mills of the company, due principally to the use of Cornish rolls and the larger proportion of middlings re-ground in ball mills.

Concentrates Disposal

The froth is broken up by clear-water sprays and thickened in a 38 by 6-ft. (11.6 by 1.8-m.) Dorr tank. The overflow goes to the slimes

FIG. 5.—DOE RUN LEAD CO'S MILL, RIVERMINES, MO.

tanks and the spigot discharge, averaging 60 per cent. solids, is sent to a 12-ft. by 11-ft. 6-in. (3.7 by 3.5-m.) Oliver filter. The Dorr-thickener feed averages 90 per cent. moisture. A permanent froth remains on the surface and is kept from overflowing by a baffle ring 12 in. (0.3 m.) inside the overflow and 12 in. above and below the surface of the water. The central shaft revolves twice per hour.

The cake from the Oliver filter averages 15 per cent. moisture, and is conveyed by belt conveyor to cars and shipped to the smelter. At present a portion of this cake is dried in vats, fitted with steam coils, to a moisture of 7 per cent., in order to enable the smelter to handle the output. The filtrate returns to the slimes thickeners. The filter speed is one revolution in 6 min. and 30 sec. The wet cake averages approximately 50 tons per 24 hr. One "blow" compartment is maintained about 6 in. above the scraper, the other compartment blowing after immersion in the pulp, which is kept at a maximum level. The blow pressure ranges from 5 to 10 lb. The vacuum is produced by a No. 5 Root pump, averaging 25 to 20 in. (0.635 to 0.508 m.). The vacuum requires 29.5 hp. and the filter 1.7 hp. The life of a canvas and burlap averages 3 months. After a new canvas is in use from 4 to 6 weeks, steam is exhausted in the pulp for the purpose of heating. This gives a larger capacity without increasing the moisture in the cake. Acid washes have been tried for the canvas without success.

The tailings from the flotation machines are pumped by a 3-in. Morris sand-dredge pump to the impounding pond, against a total head of 50 ft. (15 m.) using 18 hp. The pond dam is built of chat hauled from the mill and is 50 ft. high at its maximum point. The flow of pulp averages 225 gal. (851 l.) per minute, and immediately spreads over the surface of the pond, allowing the solids to settle rapidly. No water remains on the surface, as it filters

rapidly through the chat dam. Consequently the pond will eventually fill up in layers, forming a smooth dry body of compact slime. The filtered water is perfectly clear and free from slime.

The mill water is returned to a pond with an area of 17,000 sq. ft. (1579.3 sq. m.), formerly used for settling the mill slimes before the installation of Dorr tanks. Any slimes now escaping the Dorr tanks are caught in this pond and pumped back periodically for retreatment. The volume of water used in the mill circuit is approximately 3500 gal. (13,248 l.) per minute, or a ratio of 9:1. This compares with a ratio of 17:1 before the change was made in table practice, classification, etc. Loss in mill water, due to pumping to waste of flotation tailings, etc., is made up of mine water and from a pumping station on Big River, 3 miles away.

All machinery in the mill is electrically driven, using an average of 7.29 kw. per ton of ore treated. With the present tonnage treated this equals approximately 925 hp. As much machinery as possible is driven by individual motors, even some of the tables using separate 1-hp. motors. All motors are loaded to capacity to obtain greatest efficiency.

The labor in the mill is all American and much more efficient than a few years ago. The tons of ore milled for 1916 averaged 25 per man shift, with an average of 84 men employed per day, including maintenance. Three shifts are operated per day, 6 days a week, Sundays being used for repairs. The mill recovery varies with the grade of ore, the tailings from the plant being equal to the district average. Future improvements will probably include larger crushers, substitution of other grinding machinery for the Cornish rolls, and extension of table and flotation equipment, with the view of raising the tonnage, recovery, and the cheapening of costs.

The Rivermines Mill

The Rivermines mill is divided into six sections. The ore is crushed to 4 in. at the mine shafts and hauled in A-dump cars by railroad to a large storage bin extending across the mill end. This bin has a capacity of 3500 tons or sufficient for nearly a day's run. A trestle extends beyond the bin, so that the cars may be pinched down grade over the bin and emptied. In winter, steam nozzles are used to thaw the ore sufficiently for unloading. All the ore is weighed on scales on this trestle.

The ore is fed from the bin by roller feeders, operated by a gripping cam, to 1-in. (25.4-mm.) round-hole trommels, 8 ft. (2.5 m.) long by 24 in. (0.6 m.) in diameter. The oversize of these trommels is crushed by two No. 3 McCully gyratory crushers, the crusher discharge joining the undersize from the trommels, and is carried by belt conveyor to two 48-in. by 12-ft. (1.2 by 3.7-m.) trommels with 9-mm. round holes. The oversize from these trommels is re-crushed by rolls. The discharge from the rolls

returns to the conveyor feeding the 9-mm. trommels, the undersize of which goes to an elevator, which raises the dry ore to the 2-mm. trommels on the top floor. From this point the milling practice is practically the same as at the Bonne Terre mill.

The Marcy Mill

An 8 by 6-ft. (2.5 by 1.8-m.) Marcy ball mill has been installed in section 5 for experimental purposes. This mill has been tried in various tests and is now crushing the oversize from the 9-mm. trommels and the dewatered middlings from the fourth and fifth hutches of the Hancock jig. The discharge from the ball mill goes to the 2-mm. trommels, the oversize feeding the jig and the undersize the tables and flotation plant. As in the Bonne Terre mill, final results have not been worked out on this mill. As it is a grate-type mill, the results are similar to those of the Allis-Chalmers granulator. In this case 73 per cent. of the feed to the mill is the oversize of the 9-mm. trommels, the balance being middlings from the jigs and tables, the total averaging 750 tons per day. Approximately 200 hp. is required, the mill using a ball load of 28,000 lb. (13,800 kg.) of 5-, 4- and 3-in. balls; 5-in. balls being used for replenishing. The average moisture is 65 per cent. Screen analyses of the feed and discharge product are shown in the table.

Marcy Mill Results

Feed				Discharge			
Mesh On	Per Cent., Weight	Cumulative Per Cent., Weight	Per Cent. Total Pb Content	Mesh On	Per Cent., Weight	Cumulative Per Cent., Weight	Per Cent., Total Pb Content
1¼ in.	27.7	27.7	27.1	3 mm.	20.1	20.1	5.2
9 mm.	41.3	69.0	44.5	2 mm.	9.1	29.2	3.6
3 mm.	14.1	83.1	16.5	20	21.9	51.1	10.0
2 mm.	7.8	90.9	3.4	28	8.4	59.5	6.8
20	4.4	95.3	1.5	35	5.7	65.2	5.5
35	2.0	97.3	1.5	48	5.2	70.4	7.7
65	0.9	98.2	1.2	65	4.0	74.4	6.8
150	0.8	99.0	1.2	100	3.9	78.3	7.7
−150	1.0	100.0	3.1	150	4.0	82.3	9.0
				200	3.0	85.3	6.1
Total.....	100.0	....	100.0	−200	14.7	100.0	31.6
				Total.....	100.0	....	100.0

The jig tailings are dewatered in chat wheels as at Bonne Terre, and the table tailings in sand cones and a sand drag. They are then stacked

by belt conveyors, forming a very large tailings pile, which has accumulated since the plant started.

The tailings from the flotation plant are pumped by a 5-in. Morris sand-dredge pump to the top of the chat pile and are used for spreading the chat from the conveyors.

The mill water is settled in a pond back of the plant and is pumped from there to a reservoir situated at a maximum elevation in front of the building, from which it is re-used in the mill circuits. The volume of water used is approximately 5000 gal. per minute, or a ratio of 7:1 of ore.

All power is alternating current, an average of 8.24 kw. being used per ton of ore treated. The labor is all American. This mill will be rear-

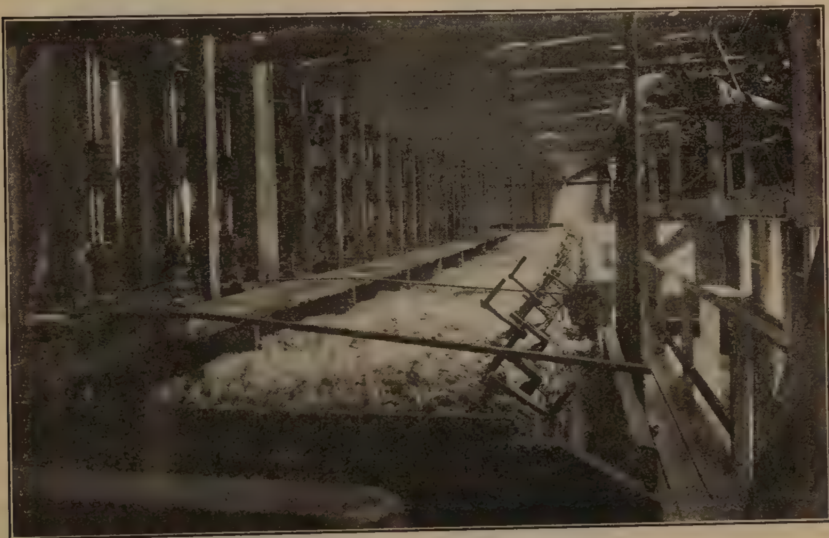

FIG. 6.—FLOTATION MACHINE, RIVERMINES MILL.

ranged later with ball mills for reducing the general tailings and increasing the capacity.

THE LEADWOOD MILL

The Leadwood mill has practically the same milling operations as at the Bonne Terre and Rivermines plants. All the ore is hoisted from No. 12 mine shaft at the mill and is conveyed direct to the crushers. In case the hoist should be out of commission for any reason, the ore from the outlying shafts can be diverted to an ore bin from which it is conveyed to the 5 K Gates crusher, thus preventing an entire shutdown of the mill.

The regular ore feed is screened on a grizzly with bars spaced 8 in. apart. The undersize is crushed in the 5 K crusher and the oversize in the 6 K crusher. The discharge from these crushers is screened through 1½-in. trommels, the oversize being recrushed in three No. 5 D Gates

crushers. From here it is conveyed to an ore bin of 1600 tons capacity, and distributed by a tripper on the conveyor.

The ore from the bin is screened through 9-mm. trommels. The oversize is re-crushed by rolls and returned to the 9-mm. screens. The undersize is screened through 1½-mm. trommels. From this point the concentration is practically the same as at Bonne Terre and Rivermines.

Fig. 7.—The Bonne Terre mill, St. Joseph Lead Co.

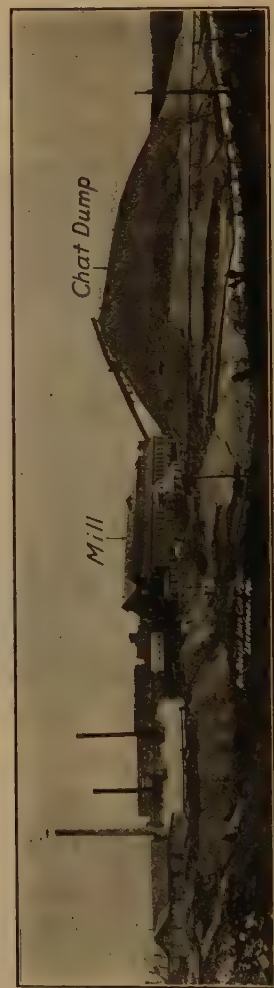

Fig. 8.—St. Joseph Lead Co's mill, Leadwood, Mo.

The jig tailings discharge in the bottom of the tail hutch of the jig and fall over stationary 1½-mm. screens. The oversize discharges on the chat conveyor, while the water drains to the sand drag. Wilfley tables, a portion Butchart riffled, are used for sand concentration. The minus 1½-mm. and plus 150-mesh sands are treated on the coarse tables, the tailings going to a concentrates pit. A drag scraper in this pit de-

waters the sands and discharges them onto the chat belt with 15 per cent. moisture. This drag gives some trouble by breaking, but manganese-steel link chains are now being used in order to reduce this to a minimum.

The fine sands, minus 150-mesh, and slimes are thickened in settlers and concentrated on Wilfley tables, the tailings going to the Dorr tanks at the flotation plant. The overflow of the sand drag also goes to the flotation plant. The slimes are thickened in four 36 by 8-ft. Dorr thickeners. The pulp is agitated in a 16-cell Minerals Separation flotation machine, the same oil being used as at the other mills. The tailings are re-treated by an air machine, and the concentrates are thickened and filtered by an Oliver filter. The flotation plant is lightly loaded, approximately 350 tons dry slimes, and the concentrates produced average 58 per cent. Pb. The slime tailings are pumped to a settling pond behind the chat dump and settle as at Bonne Terre.

The mill chat, jig and table tailings, are conveyed by belt conveyor to a tailings pile. This conveyor inclines at an angle of 26° and the chat is sufficiently dry not to cause any slippage on the belt, which is not cupped.

This mill will be equipped with ball mills for regrinding middlings and tailings as at Bonne Terre. A 6 by 4-ft. Allis-Chalmers ball granulator is installed in one section, but has not started operating, so no data can be given. A 3-ft. Hardinge mill is installed for re-grinding table middlings as at Bonne Terre, but is just starting operation, and no results are obtainable. The Leadwood ores contain considerable zinc sulphides, which will probably concentrate in this circuit.

The milling practice of the St. Joseph Lead Co. is passing through a transitory stage. Considerable experimental work is being done at each plant, a thoroughly equipped laboratory being maintained at each mill. While large tonnages are being treated in the three plants, with the highest recoveries obtainable, it is probable that the introduction of ball mills and further extension of flotation equipment will greatly change the present practice.

Discussion

L. A. Delano.—Since this paper was written, our daily mill tonnage has increased, from 2100 tons to 2700 tons a day. We now use 11,000 lb. of chrome-steel balls of 2½-in. maximum diameter, for an average crushing capacity of 275 tons per 24 hr., this giving a much higher crushing efficiency.

The Chairman (O. M. Bilharz, Miami, Okla.).—Is the increased mill capacity due to improvement in the re-crushing of middlings or to keeping middlings out of circulation and treating them separately?

L. A. Delano.—As was shown in Mr. Rabling's paper (p. 309) the tonnage was raised by about 400 tons by simply changing the arrangement of screen openings in the jig bed and using punched plate instead of woven-wire screen. Then, by adding a ball-mill, we are taking from the jigs a much larger tonnage of middlings than before, or as much as the ball-mill will crush. By sending the ball-mill product to tables and flotation, the jig capacity for original ore is increased.

Chairman Bilharz.—In your opinion, the satisfactory operation of a Hancock jig on a given ore requires adjustment of the length of the different screens and careful selection of screen openings?

L. A. Delano.—I think it does. In order to decide what screen openings to use on the jigs, we did a great deal of experimental work, making screen analyses of the various hutch products, and then hand-sorting the galena, middlings and chat from each size screen to determine the percentage of galena in each. Then adjusting our jig screens, we gradually worked the process up to a very efficient basis.

Ernest Gayford, Salt Lake City, Utah.—I would like to ask Mr. Delano whether the work of the Allen cone has been quite as satisfactory as the manufacturers claim for it.

L. A. Delano.—I am not addicted to bragging about machines, but I can say that the manufacturers of the Allen cone have not overstated its merits. It is the best automatic dewatering appliance we have found. The flow to the tank averages 85 per cent. water and the spigot discharge contains 12 per cent. The galena product goes up the belt conveyor to the car very nicely. On standing in the car, it drains to 5 per cent., and when it reaches the smelter its moisture has drained to 3 or 4 per cent. The overflow from the tank is perfectly clear; the discharge never chokes and the operation is entirely automatic, requiring no power, and very little attention by the operator.

As for its operation on material of lower specific gravity, we tried it for unwatering tailings and got the same character of work, except that the spigot product averaged 28 per cent. water, which may have been due to a larger proportion of voids in the sand, and because the material was lighter. The overflow was clear when handling 300 tons of tailings per day. When dewatering concentrates, at times we had to discharge over 100 tons a day, which was done very successfully with a ¾-in. spigot.

As to the coarseness of the material, the concentrate ranges from 9 mm. to 5 or 6 per cent. through 200-mesh. The tailings range from 2 mm. to 150-mesh; sand coarser than 3 mm., unless it contains a good deal of fines, would probably allow too much water to escape from the spigot. With concentrate, however, it is possible to handle material as coarse as

9 mm. and perhaps larger, because the range of size is complete, the bulk of it being probably about 20-mesh or 1 to 2 mm.

A. P. Watt, Mine La Motte, Mo.—I can give a little information on the Allen cone, where used for dewatering the undersize of a 10-mm. screen, containing a considerable amount of fine material. A 4½-ft. cone will handle 145 tons in 24 hr., and of the solid matter in the overflow, 97 per cent. is finer than 200-mesh. That is exceptionally good work. The spigot product contains 28 to 30 per cent. moisture. In another test we used the same cone to separate the fine sand from the feed going to a flotation plant. The sand discharge will all pass 48 mesh, ranging down to 150 mesh. The Allen cone makes the separation most satisfactorily. The spigot product carries 28 per cent. water and contains 5.8 per cent. finer than 200-mesh which is very largely free galena.

Ore-dressing Practice in the Joplin District*

BY CLARENCE A. WRIGHT,† SALT LAKE CITY, UTAH

(St. Louis Meeting, October, 1917)

THE average lead and zinc content of the ores mined and milled in the Joplin district is low as compared with that of other lead and zinc deposits throughout the United States. Because of this fact and of the high grade of the concentrates, the ratio of concentration is unusually high.

Most of the operating companies have aimed to produce as high a grade of concentrate as possible in order to obtain the highest price for their product. They have done this at the sacrifice of the percentage of zinc recovered. The average percentage of zinc recovery is from 60 to 70 per cent. For percentage of recoveries from actual mill tests, see tables.

The concentration is commonly effected by crushing the ore to ½-in. (12.7 mm.) size and subjecting it to a roughing and cleaning over two or more large Cooley jigs, of the Harz type, the finer sands being treated on the usual types of sand and slime tables. The treatment of slimes by flotation is being tested at several of the larger mills.

Coarse Concentration

When the ore has been hoisted from the mine it is dumped onto a chute leading to a grizzly made of iron bars, usually heavy rails, spaced 4 to 5 in. (101 to 127 mm.) apart. The oversize is caught on the grizzly, the mineralized pieces are broken with sledge hammers so as to pass through, and the boulders that are practically barren are sorted out by hand and trammed to the waste-rock pile. The proportion of waste rock thus sorted out varies from a fraction of 1 per cent. to 15 per cent., depending on the nature of the rock.

Crushing

The undersize material, 5 in. (127 mm.) or less in diameter, passes through the openings of the grizzly into the mill hopper below. From

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* Published with the approval of the Director of the U. S. Bureau of Mines.

† Asst. Metallurgical Engineer, U. S. Bureau of Mines.

TABLE 1.—*Milling Tests*

A. SHEET-GROUND ORE (WEBB CITY DISTRICT)

Screen Analysis of Mill Tailings			Weight of Screen Products, Per Cent.*	Cumulative Weight, Per Cent.	Zinc Values		
Mesh	Opening				Assay Zinc (Zn), Per Cent.	Per Cent. of Total Zinc	Cumulative Per Cent. of Total Zinc
	In Mm.	In In.					
3	6.680	0.2630	29.47	....	0.53	20.0	
6	3.327	0.1310	28.38	57.85	0.68	24.7	44.7
10	1.651	0.0650	15.48	73.33	0.80	15.8	60.5
20	0.833	0.0328	16.10	89.43	0.45	9.3	69.8
35	0.417	0.0164	5.50	94.93	0.42	2.9	72.7
65	0.208	0.0082	1.37	96.30	0.96	1.7	74.4
100	0.147	0.0058	0.63	96.93	1.10	0.9	75.3
150	0.104	0.0041	0.40	97.33	1.25	0.6	75.9
200	0.074	0.0029	0.32	97.65	1.58	0.6	76.5
−200	0.074	0.0029	2.35	100.00	7.90	23.5	100.0
			100.00	....	0.78	100.0	
			Check sa	mple....	0.67		

* Not including mill overflow.

B. SHEET-GROUND ORE (CARTERVILLE DISTRICT)

3	6.680	0.2630	33.58	....	0.31	24.30	
6	3.327	0.1310	30.75	64.33	0.44	31.60	55.90
10	1.651	0.0650	16.84	81.17	0.44	17.30	73.20
20	0.833	0.0328	8.78	89.95	0.31	6.35	79.55
35	0.417	0.0164	4.60	94.55	0.31	3.30	82.85
65	0.208	0.0082	2.57	97.12	0.55	3.30	86.15
100	0.147	0.0058	0.98	98.10	0.57	1.30	87.45
150	0.104	0.0041	0.80	98.90	1.04	1.95	89.40
200	0.074	0.0029	0.21	99.11	1.56	0.80	90.20
−200	0.074	0.0029	0.89	100.00	4.68	9.80	100.00
			100.00	....	0.43	100.00	
			Check sa	mple....	0.49		

C. HARD-GROUND OTHER THAN SHEET-GROUND (COMMERCE DISTRICT)

3	6.680	0.2630	1.12	....	0.60	0.80	
6	3.327	0.1310	37.60	38.72	0.70	30.40	31.20
10	1.651	0.0650	29.21	67.93	0.80	26.95	58.15
20	0.833	0.0328	10.27	78.20	0.90	10.65	68.80
35	0.417	0.0164	5.10	83.30	1.15	6.80	75.60
65	0.208	0.0082	3.42	86.72	1.13	4.45	80.05
100	0.147	0.0058	2.72	89.44	0.75	2.35	82.40
150	0.104	0.0041	2.38	91.82	0.75	2.10	84.50
200	0.074	0.0029	1.54	93.36	0.95	1.70	86.20
−200	0.074	0.0029	6.64	100.00	1.80	13.80	100.00
			100.00	....	0.86	100.00	
			Check sa	mple....	0.85		

	A	B	C
Quantity of ore treated, tons	1,960.00	3,456.00	1,504.00
Quantity of lead and zinc concentrates produced, tons	57.43	65.53	100.46
Quantity of total tailings, tons	1,902.57	3,390.47	1,403.54
Quantity of solids in overflow water from settling tanks, tons	40.22	52.66	122.20
Quantity of tailings without overflow, tons	1,862.35	3,337.81	1,281.34
Quantity of zinc concentrates, tons	42.95	62.765	39.65
Quantity of lead concentrates, tons	14.48	2.765	60.81
Assay of zinc concentrates, per cent. zinc	55.0	61.88	47.03
Assay of lead concentrates, per cent. zinc	1.2	15.00	2.09
Assay of tailings without overflow, per cent. zinc	0.67	0.49	0.85
Assay of overflow, per cent. zinc	7.04	6.38	1.13
Quantity of zinc (metallic) in zinc concentrates, tons	23.623	38.839	18.647
Quantity of zinc (metallic) in lead concentrates, tons	0.174	0.415	1.271
Quantity of zinc (metallic) in tailings without overflow, tons	12.478	16.355	10.891
Quantity of zinc (metallic) in overflow, tons	2.831	3.360	1.381
Total quantity of zinc (metallic) in ore treated, tons	39.106	58.969	32.190
Percentage of zinc recovery	60.4	65.86	57.93
Assay of lead concentrates, per cent. lead			80.93
Assay of zinc concentrates, per cent. lead			2.44
Assay of mill tailings without overflow, per cent. lead			0.20
Assay of solids in overflow, per cent. lead			0.14
Quantity of lead (metallic) in lead concentrates, tons			49.214
Quantity lead (metallic) in zinc concentrates, tons			0.967
Quantity of lead (metallic) in mill tailings without overflow, tons			2.563
Quantity of lead (metallic) in overflow, tons			0.171
Total quantity of lead (metallic) in ore treated, tons			52.915
Percentage of lead recovery			93.06
Quantity of lead and zinc (metallic) in ore treated, tons			85.105
Quantity of available lead and zinc in concentrates, tons			67.861
Percentage of lead and zinc recovery combined			79.74

the hopper the ore is fed by an incline into a crusher of the Blake type where the larger-size material is reduced to 2 in. or less in diameter.

At a few mills, a perforated shaking screen having about 3/8 to 3/4-in. (9.5 to 19-mm.) holes is used instead of an incline, a stream of water being played upon the material as it passes over the screen. By this means the finer material is more or less washed through, thus eliminating much unnecessary crushing of fines and increasing to some extent the relative capacity of the crusher and rolls, the undersize being fed directly to the feed or "dirt" elevator.

As stated above, the crushers used in this district are of the Blake type and vary in size from 8 by 14 to 12 by 24 in. The discharge from the crusher is fed directly to a set of rolls and is largely reduced to ½-in. size or less. As a rule, only one set of rolls, better known as "first" rolls or primary rolls, is used for crushing to this size, although a few of the larger mills have two sets. The primary rolls are generally 36 by 14 in. (0.92 by 0.36 m.) although both 30 by 14 and 42 by 14 are used at many mills. The roll shells are of local make and consist of ordinary hard iron. Closed rolls are generally used, while a few mills are equipped with spaced rolls which are run at a higher speed than the closed type. The advantage of the spaced rolls is the production of less slimes.

After the ore has passed through the primary rolls it is elevated to a trommel, usually with ½-in. round-hole perforations, the undersize going to the roughing jigs and the oversize returning to one or two secondary rolls, or "return" rolls, for further crushing. The product from the secondary rolls is fed back to the feed or "dirt" elevator and over the trommel again, making the recrushing a closed circuit so that all the material must pass through the ½-in. openings of the trommel.

The oversize from the trommels handling the material from the feed or "dirt" elevators is crushed by one or two sets of return rolls, usually 30-in. (0.76 m.) rolls. As mentioned above, all the material is crushed to pass the openings of the trommel before it is fed to the jigs.

The next step in crushing for coarse concentration is the freeing of mineral from included mineral particles obtained from the different jigs. This product from the jigs, which needs further grinding and subsequent concentration, is locally known as "chats." The set of rolls for handling this material is also known as "chat" rolls, and jigs that treat the crushed product separately are called "chat" jigs.

It is in this step of crushing that the least effective crushing is found in the mills of the district. For this size and character of material, steel shells are much more efficient and should be used.

Coarse Screening

The screens most commonly used throughout the district are the usual cylindrical trommel. For fine screening, both trommel and shaking screens are used, although the former finds more favor among millmen.

In the Joplin mills the trommels are run at speeds of 15 to 28 r.p.m. with slopes ⅝ to 2 in. (15.9 to 50.8 mm.) per foot. For coarse screening the usual sizes of trommels are 36 by 96 in. (0.92 by 2.44 m.), 48 by 96 in. (1.22 by 2.44 m.), and 48 by 144 in. (1.22 by 3.66 m.), with ⅜ to ⅝-in. (9.5 to 15.9-mm.) openings at the head of "rougher" jigs and ⅛ to ¼-in. (3.2 to 6.4 mm.) openings at the head of sand or "chat" jigs.

It has been stated that the discharge from the feed or "dirt" elevator is fed to a trommel; the undersize from the trommel passes to the rougher

jigs and the oversize is crushed through the "return" rolls and returned to the feed elevator. Where the entire feed passes eventually through the openings of the trommel, the trommel being in closed circuit with the return rolls, the millman is apt to be somewhat reluctant to determine the best conditions for obtaining the most efficient and economical results, as to screening and capacity, as long as the main purpose of the screen is accomplished. Frequently, in the mills of this district, the speeds of trommels are too fast for efficient screening. By increasing the speed of a trommel beyond certain limits, the screening efficiency is reduced, the quantity of oversize is increased, the wear on the rolls is greater because of the increased quantity of oversize, and the undersize that should have passed through the screen openings is reduced still finer by the rolls in closed circuit. The screening is done wet. In case the trommel has not sufficient slope, with reference to the speed and diameter, the feed is liable to be too heavy, preventing the undersize from passing freely through the screen. This is especially true of the finer-sized material when not enough water is used to wash the finer particles through.

In most cases, poor screening efficiency of trommels results from speeds being too high. The best speeds, under proper conditions, are usually 15 to 21 r.p.m. and should not, as a rule, vary much above or below these speeds.

Jigging

The most common type of jig used in this district is known as the Cooley jig, and is similar in principle to the Harz jig. It is of the fixed-sieve type, the water being forced up and down through the stationary screens or grates by the action of plungers in adjacent compartments. The number and the size of the compartments for each jig depend on the size and character of the ore treated.

Outline of Concentration System.—In general, a system of "roughing" and "cleaning" is followed in which the feed is given a preliminary cleaning that eliminates the greater proportion of waste material on one or two "rougher" jigs and the enriched product, which will assay anywhere from 10 to 25 per cent. zinc, is cleaned on a "cleaner" jig for the final treatment, bringing the zinc tenor up to 50 to 60 per cent. The "chats" or included mineral particles from both the rougher and cleaner jigs, which together assay 4 to 8 per cent. zinc, are re-crushed and either returned to the rougher-jig feed or treated separately in a "chat" jig. The tailings from the rougher jig are usually dewatered by means of a dewatering trommel having 1½-mm. to 2-mm. openings while they pass over the outside of the trommel as it slowly revolves. The undersize from this dewatering screen flows to the settling tanks and the oversize to the tailings elevator as waste. The overflow from the tailing end of the cleaner jig, and other jigs if used, also passes to the settling tanks.

The rougher jigs usually consist of five or six cells with a screening or grate area of 30 by 42 in. (0.76 by 1.06 m.) to 36 by 48 in. (0.92 by 1.22 m.). The speed of the shaft connecting the plungers and eccentrics varies from 90 to 120 r.p.m.; the length of stroke of the plungers ranges from $\frac{5}{8}$ to $1\frac{1}{4}$ in. (15.9 to 31.75 mm.). The cleaner jigs have six to seven cells with a grate area somewhat smaller than that of the rougher jigs. The speed of the shaft connecting the plungers and eccentrics for cleaner jigs is 160 to 200 strokes per minute, with length of stroke ranging from $\frac{3}{8}$ to $\frac{3}{4}$ in. (9.5 to 19 mm.). The chat or sand jigs, which are not commonly used, are smaller in size and consist usually of four to five cells, and are operated at a higher speed and shorter stroke.

The material fed to the first cell of the rougher-jig is, as a rule, not graded or classified. A bed 5 to 6 in. (127 to 152 mm.) deep is formed and as a result of the pulsating action of the water the lighter gangue material, such as flint, settles on the surface or top of the bed, the heavier free grains of lead and zinc working to the bottom. The downward suction causes the finer grains of mineral to continue through the screen or grate openings into the hutch of each compartment. The strength of the suction is increased by leaving the gates of the hutch partly open. The accumulation in the hutch is known as "smittem" and is further treated on the cleaner jig. The bed products of the first two or three cells are also drawn off and pass together with the hutch product to the cleaner jig. The chats or included mineral particles, usually from the last hutch and the last two or three beds, are drawn off and re-ground before further treatment.

Various tests have indicated that a few minor changes in the flow sheet or treatment of the ore would seem feasible to effect more efficient jigging and make a greater saving possible. Although many tests were made throughout different mills, results of only a few tests are given in order to bring out the important points under discussion.

Efficiency of Jigs.—Before the various causes of loss from jigs, especially rougher jigs, are discussed, a few results of efficiency tests on rougher jigs are given to show the average percentage of efficiency of these jigs in handling zinc-blende ore.

If we take the treatment on rougher jigs as one in closed circuit, which can be so considered in the examples given because the chats or middlings are re-ground and returned to the feed of the rougher jig, the percentage of efficiency or recovery can readily be calculated by the formula $R = \frac{100 \times c(h-t)}{h(c-t)}$ where h is the assay of the head or feed to the rougher jig, c is the assay of the concentrate product from the jig, and t is the assay of the tailing as it leaves the jig.

In the first example the feed assayed 2.85 per cent. Zn, the concentrate, c, = 25.5 per cent. zinc, and the tailing, t, = 1.10 per cent. zinc. Sub-

stituting these figures in the formula, we find that the efficiency or percentage of zinc recovery from the rougher jig in the first case was 64.17 per cent., or, in other words, this rougher jig only saved 64.17 per cent. of total zinc in the feed from the time the ore was fed into the first cell until it was discharged over the last cell of the jig. The tailing, discharged from the last cell, however, was dewatered over a trommel screen before it was allowed to leave the mill as waste. By dewatering this tailing a good proportion of the slimes were eliminated from the bulk of the tailing, and since the slimes, as a rule, assay considerably higher than the coarser particles, the tailing assay was lowered from 1.10 to 0.70 per cent. zinc.

Example No. 1 *Sheet-Ground Ore*

Feed = 2.85 per cent. zinc.................................... h = feed
Concentrates = 25.5 per cent. zinc.............................. c = concentrates
Tailing = 1.10 per cent. zinc (discharge of last cell)............. t = tailing
Dewatered tailing = 0.70 per cent. zinc

$$R = \frac{100 \times c(h - t)}{h(c - t)} = \frac{100 \times 25.5(2.85 - 1.10)}{2.85(25.5 - 1.80)} = 64.17 \text{ per cent.}$$

Example No. 2 *Sheet-Ground Ore*

Feed or head = 1.13 per cent. zinc.............................. h = feed
Concentrate = 15.0 per cent. zinc.............................. c = concentrate
Tailing = 0.50 per cent. zinc (discharge from last cell)........... t = tailing
Dewatered tailing = 0.38 per cent. zinc

$$R = \frac{100 \times c(h - t)}{h(c - t)} = \frac{100 \times 15.0\,(1.13 - 0.50)}{1.13\,(15.0 - 0.50)} = 57.67 \text{ per cent.}$$

Example No. 3 *Hard-Ground Ore Other Than Sheet*

Feed or head = 1.20 per cent. zinc.............................. h = feed
Concentrate = 9.57 per cent. zinc.............................. c = concentrate
Tailing = 0.55 per cent. zinc (discharge from last cell)........... t = tailing
Dewatered tailing = 0.34 per cent. zinc.

$$R = \frac{100 \times c\,(h - t)}{h\,(c - t)} = \frac{100 \times 9.57(1.20 - 0.55)}{1.20(9.57 - 0.55)} = 57.47 \text{ per cent.}$$

Example No. 4. *Miami District Ore*

Feed or head = 1.99 per cent. zinc.............................. h = feed
Concentrate = 14.1 per cent. zinc.............................. c = concentrate
Tailing = 0.87 per cent. zinc (discharge from last cell)........... t = tailing
Dewatered tailing = 0.82 per cent. zinc

$$R = \frac{100 \times c(h - t)}{h(c - t)} = \frac{100 \times 14.1(1.99 - 0.87)}{1.99(14.1 - 0.87)} = 59.98 \text{ per cent.}$$

Losses from Jigs.—The feed to the rougher jig in most cases is ungraded and consists of coarse and fine material. An example given below shows the results of screen analyses of the feed, the total tailing from rougher, and the dewatered tailing, and also indicating the assays of the various screen products. Table 2 represents the results of the treatment of sheet-ground ore over a rougher jig.

In the treatment of the sheet-ground ore we find, in part A of Table 2, that 3.28 per cent. of the feed to the rougher jig consisted of material finer than 200-mesh, with a zinc content of 5.77 per cent., which represented 16.81 per cent. of the total zinc in the feed.

In part B of Table 2, which represents the screen analysis of the tailing before it was dewatered, we find that the assay value of the material finer than 200-mesh was 4.50 per cent. zinc. The percentage of material finer than 200-mesh in the tailing was 3.62, which represented 32.26 per cent. of the total zinc in the tailing.

The screen analysis of the dewatered tailing, part C of Table 2, showed that there was still 1.09 per cent. of the material finer than 200-mesh contained in the tailing, with an assay value of 4.20 per cent. zinc, representing 12.12 per cent. of the total zinc contained in the dewatered tailing which was discarded.

It is reasonable to believe from the above figures alone that it would be advisable to deslime at the head of the rougher jig. However, the importance of desliming is more strongly emphasized by the fact that the dewatering screens, as used at the end of the jigs in the Joplin district, are as a rule not efficient, which means a loss of fine material in the dewatered tailing. The average screening efficiency of the dewatering screens in this district can be given as 40 to 60 per cent.

In some mills no dewatering screens are used, a rectangular box being added to the end of the jig. From the bottom of this box the coarse tailing is discharged through a spigot while the fines and water are allowed to overflow at the top. To assist in forcing as much fines and slimes as possible to overflow with the water, thus eliminating them from the spigot discharge, hydraulic water placed in the box is used in a few mills, but with rather unsatisfactory results on the whole.

Another point in favor of desliming at the head of the rougher jig is that the greater the quantity of slimes in the feed the more values, or fine mineral particles, will be forced over the jig by the flow of the top water. This fact is especially true as the quantity of water added to the cells of the jig increases the top water of each succeeding cell. Better work can also be obtained from the jig by having clear top water.

It has been stated that the chats from the rougher jig are recrushed and given a further treatment either over a chat jig or by returning them to the rougher jig. In crushing the chats and treating them over a chat or sand jig, as practised in this district, the usual way is to clean direct on the sand jig without giving the crushed material a preliminary roughing. It would seem advisable, in case there should be a relatively large quantity of chats produced from both the rougher and cleaner jigs, to treat the recrushed chats over a rougher chat jig and the roughed product over a cleaner chat jig. If only a moderate quantity of chats were produced it would be possible to crush them to a certain size, about ⅛ to

TABLE 2.—*Results of Screen Analyses in Treatment of Sheet-Ground Ore Over Rougher Jig*

A. FEED TO THE ROUGHER JIG

Screen Size	Size of Opening In Mm.	Size of Opening In In.	Weight of Screen Products, Per Cent.	Cumulative Weight, Per Cent.	Zinc Content by Assay, Per Cent.	Per Cent. of Total Zinc	Cumulative Zinc Content, Per Cent.
3	6.680	0.2630	9.26	9.26	0.55	4.53	4.53
6	3.327	0.1310	44.08	53.34	0.65	26.18	30.71
10	1.651	0.0650	22.10	75.44	1.02	20.07	50.07
20	0.833	0.0328	10.14	85.58	1.16	10.46	61.24
35	0.417	0.0164	5.55	91.13	1.68	8.30	69.54
65	0.208	0.0082	3.16	94.29	2.15	6.04	75.58
100	0.147	0.0058	1.27	95.56	3.17	3.60	79.18
150	0.104	0.0041	0.97	96.53	3.74	3.22	82.40
200	0.074	0.0029	0.19	96.72	4.52	0.79	83.19
−200	0.074	0.0029	3.28	100.00	5.77	16.81	100.00
			100.00	....	1.13	100.00	
B. TAILINGS FROM ROUGHER JIG BEFORE PASSING DEWATERING SCREEN							
3	6.680	0.2630	7.66	7.66	0.53	8.06	8.06
6	3.327	0.1310	46.80	54.46	0.35	32.48	40.54
10	1.651	0.0650	23.83	78.29	0.32	15.10	55.64
20	0.833	0.0328	9.57	87.86	0.26	4.93	60.57
35	0.417	0.0164	4.25	92.11	0.23	1.94	62.51
65	0.208	0.0082	2.08	94.19	0.32	1.31	63.82
100	0.147	0.0058	0.98	95.17	0.51	0.97	64.79
150	0.104	0.0041	1.06	96.23	1.14	2.40	67.19
200	0.074	0.0029	0.15	96.38	1.85	0.55	67.74
−200	0.074	0.0029	3.62	100.00	4.50	32.26	100.00
Total or average.....			100.00	....	0.50	100.00	
C. DEWATERED TAILINGS FROM ROUGHER JIG							
3	6.680	0.2630	7.20	7.20	0.51	9.69	9.69
6	3.327	0.1310	50.49	57.69	0.36	42.30	51.99
10	1.651	0.0640	27.81	85.50	0.34	25.04	77.03
20	0.833	0.0328	8.72	94 22	0.28	6.46	83.49
35	0.417	0.0164	2.84	97.06	0.25	1.91	85.40
65	0.308	0.0082	0.98	98.04	0.27	0.69	86.09
100	0.147	0.0058	0.41	98.45	0.43	0.46	86.55
150	0.104	0.0041	0.39	98.84	0.96	0.98	87.53
200	0.074	0.0029	0.07	98.91	1.90	0.35	87.88
−200	0.074	0.0029	1.09	100.00	4.20	12.12	100.00
Total or average.....			100.00	....	0.38	100.00	

$^3/_{16}$ in. (3 to 4 mm.), treat the crushed chats over a rougher jig, and give the enriched product obtained a final treatment over the main cleaner, in which case a cleaner chat jig would not be used. The middling or chats from the rougher chat jig could be either returned to the chat rolls and retreated over the chat jig or sent to sand rolls for further treatment on tables.

Another way in which the chats from rougher and cleaner jigs could be treated would be to crush them to table size, giving the crushed material a preliminary treatment over roughing tables, to eliminate a large proportion of waste sand, and to treat the product over the usual sand tables.

One of the chief reasons for treating the chats separately, and not returning them to the rougher-jig feed, is simply the fact that the re-crushed chats is an enriched product, assaying 4 to 8 per cent. zinc, whereas the zinc contained in the feed to the rougher jig is considerably less, assaying about 1.5 to 3 per cent. zinc. A higher percentage of recovery is possible from a richer feed than from a leaner one, so that a greater percentage of the zinc in the relatively richer chats could be saved by treating them separately than by returning them to the rougher-jig feed and mixing them up with the relatively much lower feed.

Still another point in favor of treating the recrushed chats over chat jigs is that returning them to the rougher jig increases the quantity of fines and slimes in the feed to the rougher jig. When the quantity of these chats is relatively large, and, after being crushed, they are returned to the rougher, the capacity of the rougher jig is reduced to a certain extent, and the chance for loss in the finest sizes is increased. This also brings out the advisability of desliming at the head of the rougher jig, the reasons for which have already been given.

The efficiency of cleaner jigs used in the mills of this district varies from 75 to 85 per cent. At the end of the jig there is usually a dewatering box similar to the kind used at the end of roughers where dewatering screens are not used. The tailing is discharged from the box by way of a spigot and the overflow passes out into settling tanks. The tailing is seldom discarded, but reground for further treatment.

Concentration of Fine Material

During the past few years there has been a marked improvement in the treatment of the finer material of the lead and zinc ores in this district. A few years ago tables were seldom used, but now nearly every mill contains 3 to 10 tables, and in a few of the more recent mills 20 or more tables are used. This small number of tables is due to the small tonnages of the ore treated, and the attempt to minimize the quantity of fines.

Outline of Practice in the Treatment of Fine Material

The feed to the table section or, as it is locally termed, the "sludge" section of the mills, comes chiefly from dewatering boxes at the end of jigs. At a few mills a good proportion of the finer sands and slimes are eliminated from the feed to the jigs and diverted off by launders to the settling tanks. The settling tank commonly used is rectangular in shape, about 12 ft. (3.6 m.) wide, 20 to 25 ft. (6 to 7 m.) long, 2 to 3 ft. (0.6 to 0.9 m.) deep at one end, and 6 to 8 ft. (1.8 to 2.5 m.) deep at the other end, with the bottom sloping toward one corner.

During the past year or two, Dorr thickeners have been installed at several of the mills and, although their purpose has been chiefly the settling and thickening of slimes, in a few instances they have been used for the coarser sands in giving a more uniform feed to the tables.

In most cases the feed enters a V-box with a continuous discharge, thus eliminating a good proportion of the coarser sands from the feed overflowing at the top of the V-box into the settling tank, and incidentally increasing to some extent the settling capacity of the large tanks.

The material from both the V-box and settling tank discharges into an elevator by which it is raised and discharged to a trommel. The undersize from the trommel passes to classifiers and to the tables, while the oversize is either discarded as tailing or crushed by rolls and returned to the trommel in closed circuit.

Fine Screening

The type of screen most commonly used in the table section of the mills is the trommel. In a few mills, however, a flat screen known as the Henry screen, and a local product, is used. In general, the screening

TABLE 3.—*Screen Analysis of Oversize from Sand Trommel (Sheet-Ground Ore)*

Mesh	Screen Opening		Weight of Screen Products, Per Cent.	Cumulative Weight, Per Cent.	Zinc Values		
	In Mm.	In In.			Assay Zinc, Per Cent.	Per Cent. of Total Zinc	Cumulative Per Cent. of Total Zinc
20	0.833	0.0328	45.06	....	1.70	52.00	
35	0.417	0.0164	40.87	85.93	0.75	20.80	72.80
65	0.208	0.0082	9.26	95.19	1.55	9.75	82.55
100	0.147	0.0058	2.36	97.55	4.85	7.75	90.30
150	0.104	0.0041	0.32	97.77	4.85	0.75	91.05
200	0.074	0.0029	1.07	98.84	5.15	3.75	94.80
−200	0.074	0.0029	1.16	100.00	6.65	5.20	100.00
Total.	......	....	100.00	....	1.47	100.00	

efficiency of trommels on fine material is not as good as on coarse because the smaller openings have more of a tendency to blind and the finer feed as it becomes more or less dewatered tends to ball up on the screen. The efficiency of trommels for screening out the fines of certain products in the table section of mills throughout the district varies considerably, as has been found from a number of tests.

In mills where the oversize from sand trommels is discarded as tailing, a screen analysis should be made and the various screen products assayed to determine in what sizes the mineral values are and what portion is worth saving if possible. To show the distribution of zinc in an oversize from a sand trommel the screen analysis in Table 3 is given.

Classification of Fines

In the general practice of the district, the undersize from the sand trommel passes into classifiers, usually of the ordinary V-box type with hydraulic water passing up through the sorting column, and discharges to the tables. At this point it is believed that a decided advantage could be effected, increasing the efficiency of the classifiers and decreasing the losses in slimes, by the separation of the slimes from the sands before classification. Desliming could be accomplished by the use of the well-known types of desliming classifiers, such as the Dorr, the Akins, and the Federal-Esperanza, or some similar type of drag classifier. Both the Dorr and Akins have been tried with satisfactory results, in some of the larger mills of the district, and it is believed that any of the deslimers mentioned above would greatly increase the efficiency of the table section of any mill. Not only would a better classification of the feeds to the various tables be obtained, but the proportion of water to solids could be regulated for any thickness of pulp desired for each table. Again, if the larger proportion of the slimes from the classifier feeds be eliminated, the classifiers can be more easily adjusted. Treated as slimes, which would go direct to some tank for thickening if necessary, the slime pulp would be much less diluted than when it passed over the various classifiers, and diluted with hydraulic water from each classifier, for the latter method requires a greater ratio of thickening previous to subsequent treatment, whether on slime tables or by flotation.

As stated above, the usual classifiers generally used are V-boxes with hydraulic water, placed in consecutive order above each table, the overflow from one flowing by means of a launder to the next.

The Richards-Janney type of classifier is used in a few mills, while a local type, known as the Bird classifier, is used to some extent.

As an example of the classified products fed to tables from the V-box type of classifier, Table 4 is given.

TABLE 4.—*Screen Tests of Spigot Products from Classifiers (to Tables)*

Mesh	Feed to Classifiers		Underflow from Classifier 1		Underflow from Classifier 2		Underflow from Classifier 3	
	Weight, Per Cent.	Cumulative, Weight, Per Cent.	Weight, Per Cent.	Cumulative, Weight, Per Cent.	Weight, Per Cent.	Cumulative, Weight, Per Cent.	Weight, Per Cent.	Cumulative, Weight, Per Cent.
20	0.36	0.36	0.79	0.79	0.73	0.73		
35	15.03	15.39	25.85	26.64	21.49	22.22	16.12	16.12
65	32.14	47.53	38.49	65.13	42.99	65.21	40.26	56.38
100	18.98	66.51	15.75	80.88	15.67	80.33	18.24	74.62
150	18.71	85.22	10.78	91.66	11.27	92.15	14.29	88.91
200	6.52	91.74	3.60	95.26	3.59	95.74	4.95	93.86
−200	8.27	100.01	4.72	99.98	4.25	99.99	6.13	99.99
Total	100.01	......	99.98	......	99.98	......	99.99	

	Underflow from Classifier 4		Underflow from Classifier 5		Underflow from Classifier 6		Underflow from Classifier 7	
35	9.32	9.32	3.17	3.17	0.69	0.69	0.23	0.23
65	35.41	14.73	25.60	28.77	11.24	11.98	5.35	5.58
100	21.46	66.19	26.96	55.73	26.17	38.10	16.41	21.99
150	18.25	84.44	24.62	80.35	34.18	72.28	33.84	55.83
200	7.58	92.02	10.00	90.35	13.36	85.64	16.55	72.38
−200	7.95	99.97	9.63	99.98	14.42	100.06	27.61	99.99
Total	99.97	......	99.98	......	100.06	......	99.99	

	Underflow from Classifier 8		Underflow from Classifier 9		Overflow from Classifier 9			
35	0.10	0.10	....	....	3.81	3.81		
65	1.33	1.43	1.04	1.04	0.14	3.95		
100	7.34	8.77	4.78	5.82	4.52	8.47		
150	24.75	33.52	26.95	32.77	4.38	12.85		
200	21.90	55.42	20.31	53.08	2.97	15.82		
−200	44.57	99.99	46.91	99.99	84.17	99.99		
Total	99.99	......	99.99	......	99.99			

The classifiers are of the pointed V-box type using hydraulic water. Launders connect the nine classifiers, and each classifier delivers to a separate table. For classifiers 8 and 9 hydraulic water is not used. The dimensions of each classifier and the size of the spigot-discharge opening are given in Table 5.

TABLE 5.—*Size of Classifiers*

Classifier No.	Top (Square), Inches	Depth, Inches	Spigot Discharge Openings, Inches
1	16 by 16	24	1
2	21 by 21	26	7/8
3	22 by 22	30	7/8
4	28 by 28	36	7/8
5	30 by 30	46	3/4
6	35 by 35	46	1/2
7	41 by 41	54	3/8
8	48 by 48	54	3/8
9	45 by 45	54	3/8

TABLE PRACTICE

To show the character of the table practice in the Joplin district, the results of several tests with tables in various mills are given.

Table 6 shows that when the ore fed to tables is more or less "chatty," the zinc mineral not having been entirely liberated from the gangue by crushing, the percentage of recovery is lower than when the ore is "free," that is, when the zinc mineral breaks fairly free from the gangue when crushed to table size. It is believed that when the ore is chatty, the percentage of recovery could be raised several points by recrushing the middlings from the tables.

TABLE 6.—*Results of Tests of Table Sections in Various Mills*

Test No.	Number of Tables in Operation	Zinc Content of Feed to Tables, Per Cent.	Zinc Content of Concentrates Produced from Tables, Per Cent.	Zinc Content of Tailings from Tables, Per Cent.	Percentage of Zinc Recovered	Character of Ore Treated
1	1	6.35	58.0	3.75	43.8	Chatty
2	1	4.90	59.2	1.50	71.2	Free
3	2	2.95	55.6	1.35	55.5	Chatty
4	2	4.85	57.4	1.70	67.0	Chatty
5	2	4.70	57.0	1.55	69.0	Chatty
6	2	3.12	54.5	1.00	69.1	Chatty
7	2	4.30	56.2	1.35	70.2	Chatty
8	2	5.65	57.1	1.62	73.5	Free
9	2	7.75	57.2	2.25	73.9	Free
10	3	3.75	52.8	1.20	69.5	Chatty
11	3	6.75	54.6	1.55	79.4	Free
12	4	4.65	42.0	1.30	74.3	Chatty
13	6	3.95	59.3	1.19	71.3	Free
14	7	4.65	55.0	1.42	71.3	Chatty
15	9	5.25	52.1	1.00	82.5	Chatty-free
16	10	4.90	57.5	1.01	80.8	Chatty-free

Table 7 represents results in the table section of a mill treating a fairly free ore over 10 concentrating tables. The overflow from the last classifier, consisting mostly of slimes, was returned to the settling tanks. Since the tests were made, however, one Dorr thickener and four additional tables have been added to take care of the slimes overflowing the settling tanks and classifiers in the mill.

Table 7.—*Results of Tests with a Series of* 10 *Concentrating Tables*

A. Speed, 239; Length of Stroke, 1 In.

Size of Screen, Mesh	Size of Openings		Feed			Tailing		
	In Mm.	In In.	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content
10	1.651	0.0650	0.40	0.98	0.10			
20	0.833	0.0328	0.87	0.72	0.15	0.58	0.41	0.30
35	0.417	0.0164	44.18	1.08	9.90	52.85	0.26	15.60
65	0.208	0.0082	35.06	4.86	35.25	33.68	0.52	19.95
100	0.147	0.0058	10.47	13.87	30.05	4.37	1.75	8.70
150	0.104	0.0041	1.41	16.68	4.90	0.85	2.22	2.15
200	0.074	0.0029	5.13	12.90	13.70	0.92	3.30	3.45
−200	0.074	0.0029	2.48	11.58	5.95	6.75	6.49	49.85
			100.00	4.83	100.00	100.00	0.88	100.00

Feed, 4.83 per cent. zinc; concentrate, 58.2 per cent. zinc; tailing, 0.88 per cent. zinc; percentage of zinc recovery, 83.1 per cent.

B. Speed, 242; Length of Stroke, 1 In.

35	0.417	0.0164	36.55	0.75	5.95	45.20	0.35	23.80
65	0.208	0.0082	45.61	3.30	32.80	43.14	0.55	35.70
100	0.147	0.0058	10.79	16.30	38.30	7.49	2.05	23.10
150	0.104	0.0041	1.43	20.30	6.35	1.83	2.10	5.78
200	0.074	0.0029	3.54	14.80	11.45	0.89	2.40	3.24
−200	0.074	0.0029	2.08	11.40	5.15	1.45	2.85	8.38
			100.00	4.70	100.00	100.00	10.66	100.00

Feed, 4.70 per cent. zinc; concentrate, 61.8 per cent. zinc; tailing, 0.66 per cent. zinc; percentage of zinc recovery, 86.9 per cent.

TABLE 7.—*Results of Tests with a Series of* 10 *Concentrating Tables.*—(*Continued*)

C. Speed, 242; Length of Stroke, 1⅛ In.

Size of Screen, Mesh	Size of Openings		Feed			Tailing		
	In Mm.	In In.	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content
35	0.417	0.0164	15.73	0.51	1.60	25.98	0.26	8.6
65	0.208	0.0082	47.87	1.44	13.80	56.26	0.72	51.7
100	0.147	0.0058	20.53	10.80	44.30	12.07	1.60	24.7
150	0.104	0.0041	3.96	12.50	9.90	0.75	1.81	1.7
200	0.074	0.0029	7.95	13.38	21.25	3.47	1.64	7.3
−200	0.074	0.0029	3.96	11.60	9.15	1.47	3.15	6.0
			100.00	5.0	100.00	100.00	0.78	100.0

Feed, 5.0 per cent. zinc; concentrate, 58.78 per cent. zinc; tailing, 0.78 per cent. zinc; percentage of zinc recovery, 85.8 per cent.

D. Speed, 243; Length of Stroke, ⅞ In.

35	0.417	0.0164	4.21	0.36	0.30	7.59	0.26	3.45
65	0.208	0.0082	40.04	0.98	8.20	58.82	0.36	37.30
100	0.147	0.0058	30.75	4.55	29.25	25.14	0.83	36.70
150	0 104	0.0041	11.69	11.10	27.10	4.89	1.34	11.55
200	0.074	0.0029	7.33	12.59	19.30	1.97	1.34	4.65
−200	0.074	0.0029	5.98	12.70	15.85	1.59	2.27	6.35
			100.00	4.78	100.00	100.00	0.57	100.00

Feed, 4.78 per cent. zinc; concentrate, 53.92 per cent. zinc; tailing, 0.57 per cent. zinc; percentage of zinc recovery, 89.0 per cent.

E. SPEED, 245; LENGTH OF STROKE, 1 IN.

35	0.417	0.0164	1.00	0.74	0.15	1.39	0.45	0.60
65	0.208	0.0082	23.89	0.52	2.50	34.51	0.40	13.40
100	0.147	0.0058	39.99	3.62	29.00	42.20	0.80	32.70
150	0.104	0.0041	15.46	6.40	19.80	13.46	1.00	13.10
200	0.074	0.0029	11.36	10.31	23.45	5.02	1.25	6.10
−200	0.074	0.0029	8.30	15.10	25.10	3.42	2.50	34.10
			100.00	5.0	100.00	100.00	1.03	100.00

Feed, 5.0 per cent. zinc; concentrate, 58.0 per cent. zinc; tailing, 1.03 per cent. zinc; percentage of recovery, 80.8 per cent.

TABLE 7.—*Results of Tests with a Series of* 10 *Concentrating Tables*.—(*Continued*)

F. SPEED, 245; LENGTH OF STROKE, ¾ IN.

Size of Screen, Mesh	Size of Openings		Feed			Tailing		
	In Mm.	In In.	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content
65	0.208	0.0082	5.04	0.55	0.53	18.61	0.47	5.80
100	0.147	0.0058	27.44	1.10	5.79	62.42	1.32	55.00
150	0.104	0.0041	28.78	3.70	20.43	6.26	2.58	10.80
200	0.074	0.0029	21.72	7.70	32.10	6.16	2.88	11.85
−200	0.074	0.0029	17.02	12.60	41.15	6.55	3.78	16.55
			100.00	5.22	100.00	100.00	1.50	100.00

Feed, 5.22 per cent. zinc; concentrate, 54.05 per cent. zinc; tailing, 1.50 per cent. zinc; percentage of zinc recovery, 73.3 per cent.

G. SPEED, 270; LENGTH OF STROKE, 9⁄16 IN.

35	0.147	0.0164	0.16	1.60	0.06	0.45	0.72	0.25
65	0.208	0.0082	2.58	1.38	0.70	5.02	0.51	2.25
100	0.147	0.0058	43.82	2.45	21.40	40.51	0.70	24.45
150	0.104	0.0041	20.57	4.25	17.42	27.48	1.03	24.45
200	0.074	0.0029	19.67	7.30	28.64	14.26	1.55	19.10
−200	0.074	0.0029	13.18	12.10	31.78	12.28	2.78	29.50
			100.00	5.01	100.00	100.00	1.16	100.00

Feed, 5.01 per cent. zinc; concentrate, 55.44 per cent. zinc; tailing, 1.16 per cent. zinc; percentage of zinc recovery, 78.5 per cent.

H. SPEED, 270; LENGTH OF STROKE, 9⁄16 IN.

35	0.417	0.0164	0.45	3.85	0.35	1.37	1.15	1.15
65	0.208	0.0082	1.26	3.13	0.80	0.78	0.40	0.20
100	0.147	0.0058	29.44	1.22	7.10	14.35	0.40	4.20
150	0.104	0.0041	35.21	3.76	26.10	35.07	0.90	23.05
200	0.074	0.0029	18.89	6.69	24.90	29.53	1.45	31.30
−200	0.074	0.0029	14.75	14.00	40.75	18.90	2.90	40.10
			100.00	5.07	100.00	100.00	1.37	100.00

Feed, 5.07 per cent. zinc; concentrate, 45.16 per cent. zinc; tailing, 1.37 per cent. zinc; percentage of zinc recovery, 75.0 per cent.

TABLE 7.—*Results of Tests with a Series of 10 Concentrating Tables.*—(*Continued*)

I. SPEED, 274; LENGTH OF STROKE, 5/8 IN.

Size of Screen, Mesh	Size of Openings		Feed			Tailing		
	In Mm.	In In.	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content	Per Cent. of Weight	Zinc Assay, Per Cent.	Per Cent. of Total Zinc Content
35	0.417	0.0164	1.04	4.70	0.94			
65	0.208	0.0082	0.92	2.80	0.49			
100	0.147	0.0058	21.68	3.35	14.05	8.24	0.50	2.45
150	0.104	0.0041	40.73	4.30	33.90	32.59	0.60	11.75
200	0.074	0.0029	26.92	6.10	31.76	34.17	1.50	30.90
−200	0.074	0.0029	8.71	11.20	18.86	25.00	3.64	54.90
			100.00	5.17	100.00	100.00	1.66	100.00

Feed, 5.17 per cent. zinc; concentrate, 49.6 per cent. zinc; tailing, 1.66 per cent. zinc; percentage of zinc recovery, 70.2 per cent.

J. SPEED, 276; LENGTH OF STROKE 3/4 IN.

65	0.208	0.0082	0.70	5.99	0.75	1.86	1.72	1.70
100	0.147	0.0058	3.78	1.24	0.85	4.58	0.51	1.25
150	0.104	0.0041	27.08	1.44	7.00	13.01	0.62	4.30
200	0.074	0.0029	42.15	45.30	40.21	38.91	1.60	32.65
−200	0.074	0.0029	26.29	10.82	51.20	41.64	2.75	60.10
			100.00	5.55	100.00	100.00	1.90	100.00

Feed, 5.55 per cent. zinc; concentrate, 50.82 per cent. zinc; tailing, 1.90 per cent. zinc; percentage of zinc recovery, 68.3 per cent.

General Review of Table Practice

At several mills in the Joplin district the table practice is good, but in general attention is given to the grade of concentrates produced and not enough attention is paid to the working conditions of each table. If the essentials of good table practice were observed, a higher recovery of products of the desired grades would in many cases be possible.

In the mills of the district the main bulk of the sand that reaches the table section of a mill is the undersize of the dewatered tailings from the rougher jigs and the overflow water from the other jigs.

The efficiency, or the proportion of undersize in the dewatering-trommel feed that passes through the screen openings, is 40 to 60 per cent. As the efficiency of the dewatering screens varies widely, it would seem advisable to determine their efficiency in the mills where they are used, for a considerable proportion of table-size material containing zinc

may pass over the screen with the coarse tailings. If such be the case, increased screening efficiency and better saving of the values might readily be effected.

Screen analyses have shown that before being dewatered the tailings from rougher jigs usually contain 10 to 30 per cent. of table-size material, the proportion varying with the size to which the ore was crushed before jigging and with the character of the ore.

Table 8 shows the tonnage of a classified feed on nine concentrating tables in actual practice in a mill treating sheet-ground ore. It also contains figures indicating the percentage of zinc recovery and other data for each concentrating table.

TABLE 8.—*Results of Tests Showing Tonnage of a Classified Feed, Treated on Nine Concentrating Tables**

(Covering a period of 75 hr. actual running)

Table No.	Assay Feed, Per Cent. Zinc	Assay of Tailing, Per Cent. Zinc	Assay of Concentrates, Per Cent. Zinc	Quantity of Zinc Concentrates Produced, Tons	Quantity of Zinc in Feed, Tons	Quantity of Zinc in Concentrate, Tons	Per Cent. of Zinc Extraction	Quantity of Tailings	Total Quantity of Original Feed to Tables	Tons of Original Feed per Hour
1	4.70	1.00	51.8	7.105	4.582	3.680	80.3	90.390	97.50	0.730
2	4.30	0.60	51.7	7.713	4.584	3.988	87.0	98.887	106.60	0.795
3	4.00	0.75	53.1	5.481	3.582	2.910	82.4	84.059	89.54	0.670
4	4.00	0.70	53.7	3.654	1.990	1.662	83.5	46.106	49.76	0.370
5	4.10	1.10	53.2	3.653	2.602	1.943	74.7	59.807	63.46	0.470
6	4.00	1.10	51.3	1.827	1.265	0.938	74.1	29.803	31.63	0.235
7	4.80	2.35	47.9	1.015	0.905	0.486	53.7	17.835	18.85	0.140
8	5.20	3.50	44.2	1.217	1.516	0.538	35.5	27.933	29.15	0.220
9	5.25	2.20	34.3	1.015	0.560	0.348	62.1	9.645	10.66	0.080
	4.34	1.10	50.5	32.680	21.586	16.493	76.5	464.470	497.15	3.710

* For the size of material treated on each table see Table 4.

Products from the Tables

As previously stated, the mill practice is usually to make a clean lead concentrate and a clean zinc concentrate and to return the intermediate mixed lead, iron, and zinc concentrate to the original table feed. It is believed that a better practice would be to let this product accumulate and treat it separately from time to time, as the iron sulphide unless eliminated in some way must eventually contaminate the lead or zinc concentrates. If there is little lead in the ore, the iron sulphide might better go with the lead concentrate than be returned to the feed.

As regards the middlings from tables, the usual practice in the mills is to return them to the original table feed, although in a few mills they are treated on separate tables. Returning these middlings to the feed cuts down the relative capacity of each table and does not allow the

tables to do the most efficient work. This is especially true if the table middlings are not re-ground before being returned to the system. When the middlings consist of free mineral and gangue and an already concentrated product, it would seem advisable to treat them on separate tables rather than to mix them again with the feed. By treating the middlings separately, the quantity of middlings produced from the tables is cut down, a relatively greater quantity of tailings is discarded, and the relative capacity of the tables increased.

Stage Concentration

Where the ratio of concentration is as high as it is in the mills of this district, it would seem best to use a "roughing" and "cleaning" system in concentration by tables in order to obtain high recoveries. The change would require the use of two or more roughing tables of relatively large capacity and the treatment of the enriched product from these tables on the usual sand tables. It might also be possible to eliminate a considerable proportion of the lead by this treatment before the material is fed to the sand tables. The chief advantage of the plan would be the higher recoveries obtainable and a greater relative capacity of the sand tables or, possibly, a decrease in the number of sand tables needed. This plan is being carried out in a few mills with good results.

Flotation

A number of flotation tests of the lead and zinc ores show that it is a relatively easy matter to float the sulphides. However, certain conditions, such as the quantity of floatable material available without further crushing and the possibilities of finer grinding, must be met, and there are problems to be worked out to insure success on a commercial scale. Although for the present flotation may not prove as important in the mills of the Joplin district as in many of the larger copper and zinc mills of the West, it is believed that before long many of the mills in the district will have small flotation units for saving a large proportion of the values in the fines now going to waste. Several mills are already using flotation successfully; with others it is still in the experimental stage.

The Bureau of Mines, in coöperation with the Missouri State Geological Survey, started flotation experiments in the Joplin district in the latter part of the year 1914 and continued them until the spring of 1915.

Preliminary results indicated that separation of the sulphides from the gangue is not difficult, and that a fairly good grade of concentrate could be obtained by the use of rougher and cleaner cells. Oils with a coal-tar base gave a high recovery but a low-grade concentrate, a considerable amount of gangue being entrained in the froth.

Other oils gave a higher-grade concentrate but not as good an extraction under like conditions. The best results were obtained from wood creosote and pine oils. As the coal tars and some petroleum products are good collectors and the pine oils and other suitable oils from wood distillation are good frothing agents, it would seem that a mixture of these oils, such as 80 per cent. coal tar and 20 per cent. pine oil or wood creosote, or other combinations of these oils, would be best suited for treating the Joplin ores by flotation.

Several tests were made with a mixture of turpentine and rosin. The rosin was dissolved in the turpentine in such proportions that the three different mixtures used contained 10, 20 and 30 per cent. rosin respectively, the remainder being turpentine. The tests made were mostly roughing tests to find out what the mixture would do with respect to grade of concentrate and percentage of recovery. The 20 and 30 per cent. mixtures gave the best results in both respects. By using from 1 to 5 lb. of mixture per ton of ore treated in an acid pulp of about 5:1 thickness (water to solids), the rough concentrates varied from 37 to 56 per cent. zinc, with zinc recoveries of 50 to 80 per cent. In most cases the higher the grade of concentrate obtained the lower the recoveries. It must be stated, however, that the relatively low recoveries may be considered good in view of the fact that the zinc content of the heads were low, assaying 1.05 to 2.18 per cent. zinc. It is believed that much higher recoveries could have been secured with richer feeds.

To verify the data obtained from this preliminary work, slimes produced from one of the sheet-ground mines near Webb City, Mo., were shipped to Salt Lake City, Utah, where the Bureau of Mines in coöperation with the University of Utah is conducting flotation experiments. The results (Tables 9, 10 and 11) of the experiments, which were performed by O. C. Ralston and G. L. Allen, showed that it is fairly easy to float the sphalerite from the gangue by using warm solutions and about 1 lb. per ton of any suitable oil, either from wood or coal distillation, and that acidity, although it does not seem to be necessary, allows the froth and tailing to separate more quickly. Cold solutions gave as high recoveries as warm solutions, but the grade of product was not so good.

Results in Flotation of Joplin Sheet-ground Slimes

Conditions of tests: assay of slimes tested, 8.54 per cent. Zn, 0.51 per cent. Pb, 0.7 per cent. Fe; charge in each test, 500 grams, with 1500–2500 c.c. H_2O; temperatures of pulp, 60° C., except in test No. 9, which was run at 20° C.; length of agitation, 18 min.

TABLE 9

Test No.	Acid, Pounds per Ton	Oil, Kind Used and Pounds per Ton	Weight of Froth	Grade, Per Cent. Zinc	Per Cent. Zinc Recovery	Remarks
1	7.36	1.37 S.S. No. 4 pine oil...	105.5	35.9	88.8	
2	14.72	1.37 S.S. No. 4 pine oil...	96.5	39.3	88.8	
3	36.80	1.37 S.S. No. 4 pine oil...	95.5	38.9	87.0	
4	73.60	1.37 S.S. No. 4 pine oil...	100.0	38.8	91.0	Distant froth line

Rougher Tests

Test No.	Acid, Pounds Per Ton	Oil, Kind Used and Pounds per Ton	Weight of Froth	Grade, Per Cent. Zinc	Per Cent. Zinc Recovery	Remarks
5	....	0.5 S.S. No. 4 pine oil....	104.5	36.5	89.5	
6	....	1.0 S.S. No. 4 pine oil....	86.5	43.2	87.5	
7	....	2.0 S.S. No. 4 pine oil....	103.5	36.2	87.8	
8	....	4.0 S.S. No. 4 pine oil....	94.0	39.7	87.5	
9	....	1.0 S.S. No. 4 pine oil....	120.5	31.6	89.2	Cold (20° C.)
10	....	2.5 crude cedar oil.......	93.5	40.1	87.8	Rapid separation
11	....	1.8 S.S. No. 8 pine oil....	73.5	49.6	85.5	Very rapid
12	....	1.7 wood creosote........	93.0	40.8	89.0	Strong froth
13	....	2.0 S.S. gas creosote......	78.0	46.8	85.8	Strong froth, rapid
14	....	1.7 eucalyptus oil........	110.0	34.2	88.2	Strong frother

Cleaner Test

Samples of each froth from rougher tests were taken and the remainder saved for a cleaning test. The combined rougher froths made enough material for two cleaning tests, the results of which are given in Tables 10 and 11.

TABLE 10.—*Cleaning Test No. 1*

Quantity of Froth Used, Grams	Assay of Head		Quantity of Product, Grams	Assay of Product		Percentage of Recovery
	Zinc, Per Cent.	Lead, Per Cent.		Zinc, Per Cent.	Lead, Per Cent.	
500	39.8	1.3	366.5	51.7	2.35	95.3

TABLE 11.—*Cleaning Test No. 2*

Quantity of Froth Used, Grams	Assay of Head		Quantity of Products		
	Zinc, Per Cent.	Lead, Per Cent.	Froth, Grams	Middling, Grams	Tailing, Grams
479	39.8	1.3	346	47	77

Quantity of Froth Used, Grams	Assay of Products						Percentage of Recovery		
	Froth		Middling		Tailing		Froth	Middling	Tailing
	Zinc, Per Cent.	Lead, Per Cent.	Zinc, Per Cent.	Lead, Per Cent.	Zinc, Per Cent.	Lead, Per Cent.			
479	51.5	2.04	13.5	2.66	1.0	0.4	93.2	3.3	4.0

Oils used:

Sunny South No. 4, pine oil, of General Naval Stores Co., N. Y.
Crude cedar oil, from Idaho cedar swamps, Government work.
Sunny South No. 8, special pine tar oil. General Naval Stores Co., N. Y
Beechwood creosote, sp. gr. 1.078. Mirck & Co.
Salt Lake gas creosote, fraction 500–612° F. Salt Lake gas works.
Eucalyptus oil, Atkins, Kroll & Co., San Francisco, Cal.

In Table 9, tests Nos. 1 to 4 were run to determine the effect of various quantities of acid on the percentage of extraction, grade of concentrates, and to obtain a sharp line of separation between the yellow froth and white tailings, the quantity of oil used being constant. The results indicate that the quantity of acid used has little effect on either the grade of concentrate or the percentage of extraction, but it was found that the line of separation between concentrate and tailing, as seen through the glass side of a laboratory machine, is much more quickly and better defined by the use of a large quantity of acid.

The next four tests, Nos. 5 to 8, were run with varying amounts of oil, some acid having been added to clean the surfaces of the sulphide minerals and to allow the experimenters to see the froth and tailing separate more quickly. No distinct effect was observed except that 1 lb. of oil per ton of ore treated seemed to give the highest grade of concentrate. Test No. 9 was run to parallel test No. 6, except that a cold pulp at 20° C. was used in place of the warm pulp at 60° C. used in the other tests. The lower grade of froth proves the greater ability of froth and tailing to separate from each other at the higher temperatures. The remainder of the tests Nos. 10 to 14, were run with various kinds of oils.

After sampling the froth concentrates they were mixed together and two tests (Tables 10 and 11) were run using the flotation cell as a "cleaner" unit. The previous tests had been run with the cell as a "rougher" unit with a high recovery of zinc as the end in view. When cleaning the rougher froth a concentrate of high-zinc tenor should be the aim, but these tests were carried to nearly complete removal of the zinc, so that only a small amount of middling was left for returning to the rougher cell, and the concentrates obtained were perhaps not quite as high in zinc as could be secured by returning a larger quantity of middling.

A scheme making use of rougher and cleaner units of flotation cells seems essential in order to get a high recovery and concentrates of acceptable grade.

In the experiments made in Joplin, the thickness (ratio of solids to water) of the pulp treated ranged from 1:3 to 1:7. In practice, the most favorable ratios for different ores would, of course, have to be determined by experiment. In general, a mixture of fines and slimes requires a denser pulp, whereas for a mixture consisting wholly of slimes (finer than 200-mesh) a thinner pulp is desirable. Thickening of the slimes

from the Joplin district mills would be necessary, and could be accomplished by the use of Dorr thickeners or some other device that would give the flotation machines a uniform feed of constant density.

The addition of acid may not be absolutely essential for all the ores of this district, although the local tests showed that a small quantity of acid is desirable, especially in cleaning the rougher concentrates. In testing without the use of acid it was found that sodium carbonate was beneficial in assisting the selective action of the oil mixture, thus raising the grade of concentrates produced.

Cost Data

Average detail costs of mining and milling in the Joplin district would be difficult to obtain. Each company has its own system of keeping costs, and mining conditions differ somewhat in different parts of the district. Most of the large companies keep detailed costs, while some of the smaller mines keep only general costs. The three examples of itemized cost data in Tables 12, 13, 14, are for relatively large mines that have been operating some time. Therefore, the figures can be considered somewhat below the average for the district. In the new zinc fields, recently opened near Tar River and Picher, Okla., the mining and milling costs are considerably higher, ranging from $1.50 to $2.50 per ton, but the higher lead and zinc content of the ores makes the costs per ton of concentrates produced much lower, the average at present being $20 to $30 per ton.

Table 12.—*Mining and Milling Costs of a Representative Sheet-ground Mine in the Webb City-Carterville District*

Mining Costs per Ton

Item	1914	1915	1916
Ground boss	$0.0102	$0.0117	$0.0119
Drilling	0.1927	0.2608	0.2735
Blasting	0.1721	0.1857	0.1792
Roof trimming	0.0111	0.0160	0.0182
Loading	0.1509	0.2125	0.2317
Conveying to shaft	0.0705	0.1106	0.1427
Hoisting	0.0367	0.0405	0.0487
Lighting	0.0070	0.0062	0.0052
Miscellaneous (superintendence, insurance, etc.)	0.0209	0.0403	0.0711
Totals	$0.6721	$0.8843	$0.9822

Table 12.—*Mining and Milling Costs of a Representive Sheet-ground Mine in the Webb City-Carterville District*—(*Continued*)

Milling Costs per Ton

Item	1914	1915	1916
Culling	$0.0195	$0.0120	$0.0107
Crushing	0.0448	0.0322	0.0376
Elevating	0.0402	0.0430	0.0392
Jigging	0.0318	0.0358	0.0387
Tabling	0.0364	0.0320	0.0372
Water	0.0070	0.0059	0.0063
Power	0.0279	0.0291	0.0306
Miscellaneous (superintendence, insurance, etc.)	0.0202	0.0226	0.0257
Totals	$0.2278	$0.2226	$0.2260

Total Mining and Milling Costs per Ton of Ore Mined

Items	1914	1915	1916
Mining	$0.6721	$0.8843	$0.9822
Milling	0.2249	0.2214	0.2255
Pumping	0.0587	0.0507	0.0443
Miscellaneous	0.0523	0.0307	0.0240
Gen. administrative	0.0398	0.0252	0.0435
Totals	$1.0478	$1.2123	$1.3195
Total cost per ton of concentrates produced	$33.3750	$44.9813	$51.9855
Tons of lead concentrates produced per 100 tons of ore mined and milled	1.0109	0.9423	0.9686
Tons of zinc concentrates produced per 100 tons of ore mined and milled	2.1285	1.7528	1.5886
Tons of concentrates produced per 100 tons of ore mined and milled	3.1394	2.6951	2.5572

TABLE 13.—*Mining and Milling Costs of a Sheet-Ground Mine in which Mining Consists Principally in "Taking up Stope"*

Mining Costs per Ton

Items	1914*	1915	1916
Labor, without shoveling	$0.2621	$0.1657	$0.2558
Labor, shoveling	0.1362	0.1713	0.1802
Salaries	0.0215	0.0117	0.0132
Explosives	0.2036	0.0920	0.1464
Fuel	0.0537	0.0348	0.0471
Liability insurance	0.0159	0.0146	0.0196
Paid on account of injuries to employees	0.0028	0.0111	0.0020
Water	0.0077	0.0043	0.0047
Lubricating oil	0.0073	0.0041	0.0046
Repairs, supplies, and sundries	0.0410	0.0315	0.0468
Totals	$0.7518	$0.5411	$0.7204

Milling Costs per Ton

Items	1914*	1915	1916
Labor	$0.0825	$0.1062	$0.1298
Salaries	0.0215	0.0117	0.0132
Hard iron	0.0327	0.0266	0.0297
Belting	0.0228	0.0083	0.0112
Fuel	0.0258	0.0190	0.0206
Liability insurance	0.0033	0.0046	0.0058
Paid on account of injuries to employees	0.0001	0.0008	0.0005
Water	0.0077	0.0044	0.0047
Lubricating oil	0 0028	0.0022	0.0017
Repairs, supplies, and sundries	0.0309	0.0295	0.0436
Total milling costs	$0.2301	$0.2133	$0.2608
Total mining and milling costs	$0.9819*	$0.7544	$0.9812
Total cost per ton of concentrates produced	$26.3600	$50.1940	$55.3083
Tons of lead concentrates produced per 100 tons of ore mined and milled	0.5922	0.0785	0.1014
Tons of zinc concentrates produced per 100 tons of ore mined and milled	3.1326	1.4244	1.6726
Tons of concentrates produced per 100 tons of ore mined and milled	3.7248	1.5029	1.7740

* Both heading and stope were mined during the year 1914 with a much smaller tonnage; this accounts for the relatively higher costs.

Table 14.—*Mining and Milling Costs of a Representative Mine in the Commerce District, Oklahoma*

Costs per Ton

Items	1914	1915	1916
Ground labor	$0.2885	$0.4001	$0.4667
Power	0.0725	0.0929	0.1368
Timber	0.0131	0.0028	
Powder	0.1267	0.1591	0.1513
Mill labor	0.1328	0.1896	0.2451
Mill fuel	0.0218	0.0222	0.0362
Mill supplies	0.1117	0.1078	0.1178
Shop labor	0.0427	0.0478	0.0568
Shop supplies	0.0594	0.0633	0.0645
Top labor	0.0758	0.0841	0.1161
Pumping	0.0140	0.0202	0.0291
General expenses (including overhead)	0.0475	0.1265	0.1599
Total mining and milling cost	$1.0065	$1.3164	$1.5803
Total cost per ton of concentrate produced	$18.8052	$30.4910	$39.6203
Tons of lead concentrates produced per 100 tons of ore mined and milled	1.6450	1.6582	1.2078
Tons of zinc concentrates produced per 100 tons of ore mined and milled	3.7066	2.6588	2.7757
Tons of concentrates produced per 100 tons of ore mined and milled	5.3516	4.3170	3.9835

Discussion

H. A. Wheeler, St. Louis, Mo.—This paper by Mr. Wright is one of the most thorough and exhaustive studies that has been made of the milling practice in the Joplin district. Mr. Wright spent about two years there, in behalf of the U. S. Bureau of Mines and the Missouri Geological Survey, and this paper, which is the result of his work, will repay careful reading by any student of ore dressing and is invaluable to an operator in that district.

The Joplin district is unique in its operating, mining, and milling practice. It started over 60 years ago with gopher holes, when a man with $50 or $100 capital could lease an acre of ground; if he happened to strike ore, he set up a hand jig, and if he made money, he bought a steam mill of 50 to 100 tons capacity on day-shift. The largest mill recently built at Joplin has a capacity of 1500 tons, though a 500-ton mill today is considered large and very few exceed that capacity.

The district started with the leasing system and the early "lots" were only about an acre in size, or 200 by 200 ft.; today the leases range

from 5 to 160 acres and average 10 to 40 acres. There has never been any incentive to build large plants, nor any justification for them in view of the small size of most of the leases, which are taken for 10 years. The upper rich orebodies are very erratic and may last 6 months or 6 years. At the expiration of the lease, the land owner can put you off, with only the right of the lessor to remove his machinery. Most operators work under a second lease with royalties ranging from 7 to 35 per cent., averaging 10 to 15 per cent.

It is only quite recently that operators paid any attention to the mineral that got beyond the jigs, even after the hand jig developed into a steam mill, and the employment of slime tables is comparatively recent. Hence the Joplin milling practice is very simple; it requires very little labor, and calls for a very modest investment.

Joplin is a district which is apt to invite hostile criticism from the stranger, as everything seems crude, elementary and still of the pioneer type. But when you investigate the district thoroughly, you will find that the practice is admirably adapted to local conditions and the majority of outside engineers who have plunged in to remodel it have failed. If you have a lease on only 5, 10 or 40 acres—very few exceed 40 and many in Miami are less than 20—and if your orebody is of uncertain size and your lease runs for only 10 years, the plant investment must be kept to a strict minimum and one cannot afford to erect the fine large plants that are to be found in Southeastern Missouri, in which large tracts are owned in fee simple.

Joplin has not enjoyed as much technical skill as it deserves and an important step toward improving its practice is contained in this extremely valuable paper of Mr. Wright's. Most of the ores in the district are pure, quite free, and hence easily separated. Consequently they are able to make very high-grade concentrates, the average of the district being 58 to 60 per cent. zinc in the blende concentrates and 78 to 80 per cent. lead in the galena concentrates.

When speaking of mill tonnages in Joplin, we labor under serious disadvantages, as I am not aware of any mill that weighs its ore; estimates are by the "can" or bucket. There are very few skips or mine cages, as buckets locally called "cans" are used universally for hoisting the mine "dirt." As the trammers are paid by the can, there is a strong inducement to manipulate their loading so as to increase the number rather than the weight of the cans, and a liberal allowance must be made when estimating the tonnage actually hoisted and milled.

The Chairman (O. M. Bilharz, Miami, Okla.).—The tests on which Mr. Wright's paper is based were made in 1915, and cover principally the low sheet-ground of the Webb City and Joplin field. His paper has been of great value, I think, as an incentive to the operators to make

improvements, a number of which have been made during the last year in the old district as well as in the new Oklahoma-Kansas field. The cost of mining and milling at Miami during 1917 can probably be put close to $2 a ton.

H. A. Wheeler.—I neglected to state that Mr. Wright also describes some experiments which he made on flotation of the Joplin ores, and a few mills have introduced that system. It is a district where undoubtedly flotation has a fine future, but the expense of erecting high-grade plants is usually not encouraging, with the short life of most of the orebodies. Not owning the fee and the leases being short-lived, with usually a heavy royalty, the plant investment is kept as low as possible; consequently the mills are built light and cheaply, so that they can move off to another lease when the ore is exhausted.

F. H. Gartung, Miami, Okla. (written discussion*).—It might be interesting to state in more detail some of the peculiar conditions which have affected the milling practice of the Joplin District.

1. Land ownerships have, as a rule, consisted of small acreages and the leases thereon called for a separate mill on each 40 acres. This resulted in a large number of small, cheap mills, as in the early days a very low mining and milling cost, along with a small investment, was generally necessary for realizing a fair profit.

During the high prices which followed shortly after the beginning of the present war, it was realized more than ever that the mill losses were very serious, and the operators at once started to improve their milling methods, by adding more tables, using mechanical classifiers and deslimers, Dorr thickeners, slime tables, and flotation methods, with the result that in the newer mills great steps were made toward a better recovery from the zinc and lead ores. A few figures will make clear how this added investment was highly profitable to the operator, while also conserving the lead and zinc resources of the district:

Price of Concentrate, per Ton	Recovery, 65 Per Cent.		Recovery, 75 Per Cent.	
	Value per Ton Broken	Net Value, Less 20 Per Cent. Royalty	Value per Ton Broken	Net Value, Less 20 Per Cent. Royalty
$ 40	$1.20	$0.96	$1.38	$1.11
65	1.95	1.56	2.25	1.80
135	4.05	3.24	4.67	3.74

From these figures it appears that, with the price of concentrates at $40, and recovery 65 per cent., 1 ton of rock, less 20 per cent. royalty, is worth $0.96; while with recovery of 75 per cent., under the same condi-

* Received Oct. 8, 1917.

tions, 1 ton of rock has a value of $1.11, or $0.15 more. With concentrates at $65, the difference would be $0.24, and with $135 concentrate the difference would be $0.50 per ton of rock broken.

This saving would justify an increased plant investment and an increased milling cost, which would naturally follow the installation of other machinery necessary for this better recovery. Thus we find that the price of ore might affect the economic recovery of the metal.

2. The relative small value of a ton of ore-bearing rock, under normal conditions, makes a low mining and milling cost absolutely necessary; hence the mills were built to be operated as cheaply as possible and handle large tonnages. This would naturally result in coarse crushing, in order to handle large tonnage, and also to reduce the amount of sliming, since the ores slime much worse than the rock, as shown very clearly by Mr. Wright's tests.

3. The preference which ore buyers have shown ordinarily for the high grades of ore has been the cause of the high ratio of concentration.

A great many of Mr. Wright's suggestions have been accomplished in different ways; for instance, instead of desliming ahead of the rougher jig, some of the mills are desliming on the cells of the rougher jig by a skimming device, by means of which a certain part of the top water of each cell is drawn off and flumed to the settling tanks to be treated on tables. A better efficiency in dewatering the rougher-jig tails has also been attained in some mills by means of trommel screens. Table practice has also been improved considerably by means of better classification and desliming.

The scarcity of such data in the Joplin District as are contained in Mr. Wright's report make it worth a great deal to the operator, and should encourage more mill tests than have been made heretofore, which will all help to increase the efficiency of our milling practice.

A Standard Screen Scale for Testing Sieves*

(St. Louis Meeting, October, 1917)

Since the adoption by the U. S. Bureau of Standards several years ago of specifications for standard 100- and 200-mesh sieves, frequent requests have been received that this Bureau test and certify sieves of other sizes than these. With a view to the adoption of a series of standard testing sieves which might be of use to all industries making fineness tests, this Bureau for two years has been studying the question of such a standard screen scale. Various scales that have been proposed were considered, and information was sought of representative firms in the various industries interested as to their requirements. Manufacturers of sieves have also been consulted as to the desirability of different screen scales and the practicability of their manufacture. As a result of this study of the question, a conference was called at the Bureau of Standards, Apr. 20, 1916, of representatives of various committees of the American Society for Testing Materials, American Society of Civil Engineers, American Institute of Mining Engineers, American Foundrymen's Association, Mining and Metallurgical Society of America, American Water Works Association, American Institute of Metals and the American Spice Trade Association; also representatives of the Committee of Revision of the U. S. Pharmacopœia, the U. S. Geological Survey, the U. S. Bureau of Mines, the U. S. Office of Public Roads and Rural Engineering, the U. S. Office of Grain Standardization and the U. S. Bureau of Standards; also representatives of a number of private firms engaged in industries in which sieves are used, such as the glass, the drug milling, the abrasive, the asphalt, the mining, the spice, the chemical, and the graphite industries; also representatives of the firms in this country manufacturing wire cloth and sieves.

This Conference, after considering the various screen scales either proposed or now in use, adopted as a Standard Screen Scale that given in Table 1, and recommends that it be adopted generally by scientific,

* Adopted by a conference of representatives of various scientific and technical societies, government bureaus, and private firms, held at the Bureau of Standards, and recommended for general adoption in the interests of securing uniformity of usage.

TABLE 1.—*Standard Screen Scale*

Based on a 1-mm. opening sieve with the square root of 2, or 1.4142, as the ratio of the openings of successive sieves coarser than 1 mm., and the fourth root of 2, or 1.1892, as the ratio of the openings of successive sieves finer than 1 mm.

	Opening	Mesh	Wire Diam.	Ratio	Tolerances	
				Wire Diam. to Opening	Mesh	Diam.
8-mm. Sieve:						
Metric	8.00	1.0	2.00	0.25	±0.01	±0.08
Customary	0.315	2.54	0.079	....	±0.03	±0.003
5.66-mm. Sieve:						
Metric	5.66	1.4	1.48	0.26	±0.01	±0.08
Customary	0.223	3.56	0.058	....	±0.03	±0.003
4-mm. Sieve:						
Metric	4.00	2.0	1.00	0.25	±0.02	±0.05
Customary	0.157	5.1	0.039	....	±0.05	±0.002
2.83-mm. Sieve:						
Metric	2.83	2.75	0.81	0.29	±0.02	±0.05
Customary	0.111	7.0	0.032	....	±0.05	±0.002
2-mm. Sieve:						
Metric	2.00	3.9	0.56	0.28	±0.04	±0.05
Customary	0.079	9.9	0.022	....	±0.1	±0.002
1.41-mm. Sieve:						
Metric	1.41	5.0	0.59	0.42	±0.08	±0.025
Customary	0.0555	12.7	0.0232	....	±0.2	±0.0010
1-mm. Sieve:						
Metric	1.00	7.0	0.43	0.43	±0.15	±0.020
Customary	0.0394	17.8	0.0169	....	±0.4	±0.0008
0.85-mm. Sieve:						
Metric	0.85	8.0	0.40	0.47	±0.2	±0.015
Customary	0.0335	20.3	0.0157	....	±0.5	±0.0006
0.71-mm. Sieve:						
Metric	0.71	9.0	0.40	0.56	±0.3	±0.012
Customary	0.0280	22.9	0.0157	....	±0.75	±0.0005
0.59-mm. Sieve:						
Metric	0.59	10.0	0.41	0.69	±0.4	±0.012
Customary	0.0232	25.4	0.0161	....	±1.0	±0.0005
0.5-mm. Sieve:						
Metric	0.50	12.0	0.33	0.66	±0.4	±0.012
Customary	0.0197	30.5	0.0130	....	±1.0	±0.0005
0.42-mm. Sieve:						
Metric	0.42	14.0	0.29	0.69	±0.6	±0.010
Customary	0.0165	35.6	0.0114	....	±1.5	±0.0004
0.36-mm. Sieve:						
Metric	0.36	16.0	0.26	0.72	±0.6	±0.010
Customary	0.0142	40.6	0.0102	....	±1.5	±0.0004

TABLE 1.—*Standard Screen Scale.*—(*Continued*)

	Opening	Mesh	Wire Diam.	Ratio	Tolerances	
				Wire Diam. to Opening	Mesh	Diam.
0.29-mm. Sieve:						
Metric	0.29	20.0	0.21	0.72	±0.8	±0.010
Customary	0.0114	50.8	0.0083	....	±2.0	±0.0004
0.25-mm. Sieve:						
Metric	0.25	23.0	0.185	0.74	±1.0	±0.008
Customary	0.0098	58.4	0.0073	....	±3.0	±0.0003
0.21-mm. Sieve:						
Metric	0.21	27.0	0.16	0.76	±1.0	±0.008
Customary	0.0083	68.6	0.0063	....	±3.0	±0.0003
0.17-mm. Sieve:						
Metric	0.17	31.0	0.15	0.88	±1.0	±0.008
Customary	0.0067	78.7	0.0059	....	±3.0	±0.0003
0.14-mm. Sieve:						
Metric	0.14	39.0	0.116	0.83	±1.0	±0.008
Customary	0.0055	99.1	0.0046	....	±3.0	±0.0003
0.125-mm. Sieve:						
Metric	0.125	47.0	0.089	0.71	±1.5	±0.008
Customary	0.0049	119.4	0.0035	....	±4.0	±0.0003
0.105-mm. Sieve:						
Metric	0.105	59.0	0.064	0.61	±2.0	±0.008
Customary	0.0041	149.9	0.0025	....	±5.0	±0.0003
0.088-mm. Sieve:						
Metric	0.088	67.0	0.061	0.69	±2.5	±0.005
Customary	0.0035	170.2	0.0024	....	±6.0	±0.0002
0.074-mm. Sieve:						
Metric	0.074	79.0	0.053	0.72	±3.0	±0.005
Customary	0.0029	200.7	0.0021	....	±8.0	±0.0002
0.062-mm. Sieve:						
Metric	0.062	98.0	0.040	0.65	±3.5	±0.005
Customary	0.0024	248.9	0.0016	....	±9.0	±0.0002
0.052-mm. Sieve:						
Metric	0.052	110.0	0.039	0.72	±4.0	±0.004
Customary	0.0021	279.4	0.0015	....	±10.0	±0.00015
0.044-mm. Sieve:						
Metric	0.044	127.0	0.035	0.80	±5.0	±0.004
Customary	0.0017	323.0	0.0014	....	±12.0	±0.00015

technical, and engineering societies and committees, and by branches of National, State, and Municipal Governments as a part of their specifications for materials and methods of test; also that it be used by private firms who have need of standard sieves.

This screen scale is essentially metric. The sieve having an opening of 1 mm. is the basic one, and the sieves above and below this in the series are related to it by using in general the square root of 2, or 1.4142, or

the fourth root of 2, or 1.1892, as the ratio of the width of one opening to the next smaller opening. The first ratio is used for openings between 1 mm. and 8 mm. while the fourth root of 2 is used as the ratio for openings below 1 mm. to give more sieves in that part of the scale. The series has been made large enough, it is hoped, to meet the needs of all industries. Some industries may have occasion to use all the sieves in a certain section of the series and none of the others, while in other industries it may be desirable to use only certain sieves selected from the whole range of the series. In making such selections it is recommended that this be done on some systematic plan as, for example, the selection of every other sieve or of every fourth one in the series below 1 mm. opening and every other sieve above 1 mm., in which case the ratio of each opening to the next smaller one would be as 2 to 1.

Because of the wide range of openings in sieves now manufactured which are possible with a given number of meshes of wire per unit length by the use of wires of different diameters, and the consequent confusion and uncertainty which arises in designating sieves by the number of meshes per unit length, the sieves of this series have been designated by the width of the opening in millimeters as, for example, a 1.41-mm. sieve, or a 0.36-mm. sieve. It is urgently recommended that all users of sieves in the future designate these standards sieves in this way and that the manufacturers mark and list the sieves in this manner rather than by the meshes per inch.

In the designation and certification of the sieves the metric units will be used by the Bureau of Standards. In Table 1, however, are also given the equivalents of these metric quantities in inches in order that the series may be more readily related to work previously done. It is, of course, immaterial whether units of the metric system, or of the customary system, or of any other system, are used in the manufacture of the sieves, provided they are within the tolerances.

To meet the need for sieves of this series at the present time, a temporary provision has been made in the specifications for the acceptance of sieves of slightly different mesh and wire diameter than that called for in the screen scale, provided the resultant opening is the same as the nominal opening within a small range. This will make possible the use of a number of sieves now on the market in which the ratio of wire diameter to opening is only slightly different from that of the screen scale. This provision will be withdrawn when conditions are such that the manufacturers of sieves can furnish sieves made more exactly in accordance with the specifications.

The Bureau of Standards hereby announces that it will test sieves of this series to determine whether they conform to specifications given below. This test will consist of the examination of the mesh of both the warp and shoot wires of the cloth to ascertain whether it comes within

the tolerances allowed; also measurements of the diameter of wires in each direction to determine the average diameter, and a measurement of any large openings to ascertain whether they exceed the limits given in these specifications; also an examination of the sieve to discover any imperfections of the sieve which may seriously affect its sieving value. Sieves which pass the specifications will be stamped with the seal of this Bureau and will be given an identification number. A certificate will be furnished for each sieve that passes the requirements.

For sieves which fail to meet the specifications, reports will be rendered showing wherein the sieve was not up to the standard.

A fee of $2 per sieve will be charged for the test of the sieves when submitted singly. For from 2 to 9 sieves submitted at one time the fee will be $1.50 per sieve. For lots of 10 or more the fee will be $1 per sieve. Only half of the above fees will be charged for such sieves as may be rejected for exceeding the tolerances of mesh, in which case the wire diameter will not be measured.

In Table 1, in the first column headed "Openings," is given the width of the opening (on the first line in millimeters, on the second line in inches), for each sieve. In the second column, headed "Mesh," on the first line is given the number of meshes per linear centimeter, and on the second line the equivalent number of meshes per linear inch. In the third column, headed "Wire Diam.," is given on the first line the diameter of wire in millimeters, and on the second line its equivalent in inches. In the fourth column, headed "Ratio Wire Diam. to Opening," is given the ratio of the wire diameter to the width of the opening between wires. In the fifth and sixth columns, headed "Tolerances," are given the tolerances for these sieves mentioned in the specifications below. These tolerances will, for testing purposes, be used essentially in the metric dimension, but on the second line in each case is given the equivalent in inches in order that the tolerances may be compared readily with those in previous use. The tolerance in the fifth column is that for the meshes per centimeter and per inch, respectively, and in the sixth column the tolerances for wire diameter in millimeters and inches, on the first and second lines respectively.

In Table 2 is given a list showing the dimensions of sieves now on the market which would most nearly meet the specifications and tolerances of the Standard Screen Scale. The headings of the columns of this table are self-explanatory. Where the dimensions of more than one sieve are shown for a given sieve of the screen scale, one set of dimensions is that of one manufacturer, and the other that of another. In some cases the third set, if one is given, is made by two or more manufacturers.

In Table 3 is given a list of sieves between 1-mm. opening and 8-mm. opening which would be interpolated in the series of the Standard Screen Scale if the fourth root of 2 or 1.1892, were used as the ratio of successive

TABLE 2

Sieves now on the market which would most nearly meet the tolerances of the Standard Screen Scale.

	Opening in Mm.	Opening in Inches	Meshes per Inch	Wire Diam. in Inches	Wire Diam. in Mm.
8-mm. Sieve.......	8.13	0.320	2.5	0.080	2.03
	8.05	0.317	2.5	0.083	2.11
5.66-mm. Sieve	5.66	0.223	3.5	0.063	1.60
	5.61	0.221	3.5	0.065	1.65
4-mm. Sieve.......	4.04	0.159	5.0	0.041	1.04
	4.06	0.160	5.0	0.040	1.02
2.83-mm. Sieve....	2.82	0.111	7.0	0.032	0.81
	2.82	0.111	7.0	0.0315	0.80
2-mm. Sieve.......	1.96	0.077	10.0	0.023	0.58
	2.03	0.080	10.0	0.020	0.51
	2.03	0.080	10.0	0.0205	0.52
1.41-mm. Sieve....	1.40	0.055	12.0	0.028	0.71
	1.42	0.056	12.0	0.027	0.69
1-mm. Sieve.......	1.01	0.0396	18.0	0.016	0.41
	0.99	0.0391	18.0	0.0165	0.42
0.85-mm. Sieve....	0.85	0.0335	20.0	0.0165	0.42
	0.86	0.0340	20.0	0.016	0.41
0.71-mm. Sieve....	0.72	0.0285	22.0	0.017	0.43
	0.70	0.0275	22.0	0.018	0.46
	0.74	0.0290	22.0	0.0165	0.42
0.59-mm. Sieve ...	0.58	0.0230	26.0	0.0155	0.39
	0.60	0.0235	26.0	0.015	0.38
0.5-mm. Sieve.....	0.50	0.0198	30.0	0.0135	0.34
	0.50	0.0196	30.0	0.01375	0.35
0.42-mm. Sieve....	0.42	0.0166	35.0	0.012	0.30
	0.42	0.0164	35.0	0.01225	0.31
0.36-mm. Sieve....	0.36	0.0140	40.0	0.011	0.28
	0.35	0.01375	40.0	0.01125	0.29
	0.37	0.01475	40.0	0.01025	0.26
0.29-mm. Sieve....	0.28	0.0110	50.0	0.009	0.23
0.25-mm. Sieve....	0.25	0.0097	60.0	0.007	0.18
	0.26	0.0102	60.0	0.0065	0.17
	0.23	0.0092	60.0	0.0075	0.19
0.21-mm. Sieve....	0.19	0.0073	70.0	0.007	0.18
	0.20	0.0078	70.0	0.0065	0.17
0.17-mm. Sieve....	0.17	0.0068	80.0	0.00575	0.15
0.14-mm. Sieve....	0.14	0.0055	100.0	0.0045	0.114
0.125-mm. Sieve...	0.117	0.0046	120.0	0.0037	0.094
	0.119	0.0047	120.0	0.0036	0.091
0.105-mm. Sieve...	0.104	0.0041	150.0	0.0026	0.066
	0.094	0.0037	150.0	0.0030	0.076
0.088-mm. Sieve...	0.089	0.0035	170.0	0.0024	0.061
	0.084	0.0033	170.0	0.0026	0.066
0.074-mm. Sieve...	0.074	0.0029	200.0	0.0021	0.053
0.062-mm. Sieve...	0.061	0.0024	250.0	0.0016	0.041
	0.058	0.0023	250.0	0.0017	0.043
0.052-mm. Sieve...	0.051	0.0020	280.0	0.0016	0.041
0.044-mm. Sieve...	0.041	0.0016	325.0	0.0015	0.038
	0.041	0.0016	330.0	0.0014	0.036

sieves throughout the series. Suitable meshes and wire diameters to give these openings are also given, together with the tolerances under which such sieves would be tested if used. These sieves have not been included in the Standard Screen Scale, as it is believed to be unnecessary to have so many sieves in this part of the scale. This list is given separately, however, in case any organization in selecting sieves systematically from the Standard Screen Scale in the series of openings less than 1 mm. finds it desirable to use any of these interpolated sieves above 1 mm. in carrying out a systematic plan of selection of sieves. In case any organization or firm should adopt any of these six sieves under such circumstances, the Bureau of Standards will test and certify them in accordance with the dimensions given herewith.

TABLE 3

Additional sieves which would be interpolated between the 8-mm. opening and the 1-mm. opening of the Standard Screen Scale by the use of the fourth root of 2, or 1.1892, as the ratio of the successive sieve-openings.

	Opening	Mesh	Wire Diam.	Ratio Wire Diam. to Opening	Tolerances	
					Mesh	Diam.
6.72-mm. Sieve:						
Metric............	6.72	1.2	1.61	0.24	±0.01	±0.08
Customary........	0.265	3.05	0.063	....	±0.03	±0.003
4.76-mm. Sieve:						
Metric............	4.76	1.6	1.49	0.31	±0.02	±0.05
Customary........	0.187	4.1	0.059	....	±0.05	±0.002
3.36-mm. Sieve:						
Metric............	3.36	2.4	0.81	0.24	±0.02	±0.05
Customary........	0.132	6.1	0.032	....	±0.05	±0.002
2.38-mm. Sieve:						
Metric............	2.38	3.15	0.79	0.33	±0.04	±0.05
Customary........	0.094	8.0	0.031	....	±0.1	±0.002
1.68-mm. Sieve:						
Metric............	1.68	4.0	0.82	0.49	±0.04	±0.025
Customary........	0.067	10.2	0.032	....	±0.1	±0.0010
1.19-mm. Sieve:						
Metric............	1.19	6.0	0.48	0.40	±0.1	±0.020
Customary........	0.0468	15.2	0.0189	....	±0.25	±0.0008

In Table 4 is given an extension of the metric series beyond the 8-mm. opening using the square root of 2, or 1.4142, as the ratio of successive openings. A suggested diameter of wire for use in making such sieves is also given. No tolerances for these sieves are given, however, as it is not proposed to test such sieves at the Bureau of Standards, since the user of the sieves could ordinarily make such tests of them as are necessary with sufficient accuracy. Such sieves would generally be made

up into sieves of larger diameter than those of the Standard Screen Scale and would usually be made of iron or steel wire.

TABLE 4

Showing the sieves which an extension of the Standard Screen Scale from an 8-mm. opening to a 128-mm. opening would comprise, using the square root of 2, or 1.4142, as the ratio of successive openings.

	Opening	Wire Diameter
128-mm. Sieve:		
Metric	128.0	9.5
Customary	5.04	0.375
90.5-mm. Sieve:		
Metric	90.5	9.5
Customary	3.56	0.375
64-mm. Sieve:		
Metric	64.0	6.4
Customary	2.52	0.25
45.3-mm. Sieve:		
Metric	45.3	5.26
Customary	1.78	0.207
32-mm. Sieve:		
Metric	32.0	4.85
Customary	1.26	0.192
22.6-mm. Sieve:		
Metric	22.6	4.11
Customary	0.891	0.162
16-mm. Sieve:		
Metric	16.0	3.05
Customary	0.630	0.120
11.3-mm. Sieve:		
Metric	11.3	2.67
Customary	0.445	0.105

In the specifications given below the diameter or other dimensions of the sieve frames are not given, with the idea that any organization or firm in adopting these specifications will decide upon the size of sieve frame that best meets its needs. For purposes of uniformity and interchangeability of sieves, pans, and covers, it is recommended that sieves be purchased in diameters of either 20 cm., 15 cm., or 10 cm. (7.87 in., 5.91 in., or 3.94 in.). These are the outside diameters of the bottom of the sieve or the inside diameter of the top of the sieve.

SPECIFICATIONS FOR SIEVES OF THE STANDARD SCREEN SCALE

Wire cloth for standard sieves shall be woven (not twilled, except that the cloth of 0.062-mm., the 0.052-mm., and the 0.044-mm. sieve,

may be twilled until further notice), from brass, bronze, or other suitable wire and mounted on the frames without distortion. To prevent the material being sieved from catching in the joint between the cloth and the frame, the joint shall be smoothly filled with solder, or so made that the material will not catch.

The number of wires per centimeter of the cloth of any given sieve of the Standard Screen Scale shall be that shown in the accompanying Table 1 in the second column, headed "Mesh," and the number of wires in any whole centimeter shall not differ from this amount by more than the tolerance given in the 5th column, that headed "Mesh" under the heading "Tolerances." No opening between adjacent parallel wires shall be greater than the nominal width of opening for that sieve by more than the following amounts:

10 per cent. of the nominal width of opening for the 8-mm. to 1-mm. sieve, inclusive.

25 per cent. of the nominal width of opening for the 0.85-mm. to the 0.29-mm. sieve, inclusive.

40 per cent. of the nominal width of opening for the 0.25-mm. to the 0.125-mm. sieve, inclusive.

60 per cent. of the nominal width of opening for the 0.105-mm. to the 0.044-mm. sieve, inclusive.

The diameters of the wires of the cloth of any given sieve shall be that shown in the third column of Table 1 headed "Wire Diam." and the average diameter of the wires in either direction shall not differ from the specified diameter by more than the tolerance given in the last column of Table 1, that under "Tolerances" headed "Diam."

The Bureau of Standards also reserves the right to reject sieves for obvious imperfections in the sieve cloth or its mounting, as, for example, punctured, loose or wavy cloth, imperfections in soldering, etc.

Until further notice, to permit the use of sieves now on the market which have slightly different mesh and wire diameters from that specified above, sieves will be certified as satisfactory if the measurements of mesh and wire diameters show the resulting average width of opening to be within 4 per cent. of the nominal opening of a given sieve, and the ratio of wire diameter to opening of the sieve in question is within 0.03 of that given in Table 1 in the column headed "Ratio Wire Diam. to Opening" for the 8-mm. to the 2-mm. sieves, inclusive, and within 0.06 of the ratio given for sieves of smaller openings than 2 mm.

A Uniform Sizing Diagram from Different Screen Standards

BY JOHN RANDALL,* BOULDER, COLO.

(St. Louis Meeting, October, 1917)

It is a fair assumption that the main purpose of any diagram is to present facts to the eye in more convenient form than they could be tabulated in figures, and this implies that a screen diagram should set forth in a convenient and familiar form certain facts as to the character of the material screened regardless of the screen standards used, and it seems particularly desirable on account of the well-known differences in the size of openings of different laboratory screens having the same nominal mesh.

In making screen-sizing tests it has been customary first to consider the largest size, going progressively to the smallest, and this seems to have led to the practice of placing the smallest size at the right-hand side of the diagram and measuring toward the left. This is not altogether convenient and might be changed to advantage, but for illustration I have preserved that feature and present the method just as I have used it for over 2 years. Direct plotting in which the intervals on the diagram are proportional to the sizes of the openings, or to the arithmetical difference between them, seems at first sight more natural and is extremely useful for certain purposes, but it makes the diagram very crowded in the smaller screen sizes while the larger ones occupy unnecessary space, making it less convenient for most purposes, and for that reason the method here discussed is the well-known one based on the ratio between openings, and is plotted in logarithms instead of the direct values of the screen opening. Metric coördinate paper is convenient, although not necessary; it is recommended, as the use of the metric scale gives a diagram of convenient size, but any scale of equal parts may be used.

In order to make any screen diagram translatable in terms of any standard screen size, it is necessary to have noted on the size scale of the diagram all points corresponding to the openings of the various screens in question, instead of a single series of points as with a single screen. In drawing the diagram assume 0.001 in. (corresponding approximately to 0.025 mm.) to be the smallest size to be considered and fix this as the zero point at the lower right-hand corner of the diagram. This value is selected because the values are to be plotted in logarithms, and the logarithm of 1 is zero. Then lay off to the left for the position of each screen

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opening a distance on the scale equal to the logarithm of the opening. If the screen openings are given in millimeters instead of inches, a corresponding change must be made in the horizontal scale of the diagram. The zero point of the diagram corresponds to 0.025 mm. and since the mantissa of the logarithm of 0.025 is approximately 4, the logarithmic values for the millimeter scale can be made directly comparable with the inch scale by subtracting 4 from the first figure of the mantissa of the millimeter logarithm. After the subtraction of 4 the screen openings can be located on the scale as before and their position will coincide with that of openings of the same size given in inches. This being the case, it is immaterial what unit of measurement is used in designating the screen openings, since it is very easy to transfer directly from one scale to the other. This transfer is of course not absolutely accurate but it is sufficiently accurate for the purposes of the comparison.

In Table 1 are given the screen openings and logarithm values for four different sets of laboratory screens, the logarithms being rounded off to not more than two decimal places for convenience in plotting, and the characteristics being omitted. The rounding off of the logarithms makes the ratio exact in the case of the screens having a common ratio of increase of the size of the opening, a condition not otherwise capable of exact expression decimally, and introduces no appreciable error. The different screens here tabulated in part are: (1) The Tyler screens, based on the opening of a 200-mesh screen standardized by the U. S. Bureau of Standards, only each alternate size being here listed, the lineal dimensions of the openings having an ascending geometric ratio of 2; ratio with all the sizes in would be $\sqrt{2}$; (2) the I. M. M. screens, having the sanction of the Institution of Mining and Metallurgy, London, and almost universally used throughout the British Empire. The screens here shown approximate for practical purposes a ratio of 1.5874, or $\sqrt[3]{4}$ for the ratio of increase. Stadler (Royal School of Mines, So. Kensington, London) has proposed that these screens be manufactured in the following sizes in order to make the ratio more nearly exact, but as now made the approximation to a common ratio is close:

Screen Openings Proposed by Stadler

Nominal Mesh	Diameter of Opening in Inches
12	0.03937
20	0.02480
30	0.01562
50	0.00984
80	0.00620
120	0.00391
200	0.00246

When the I. M. M. screen standard was first adopted, there seems to have been no attempt to secure a common ratio between openings, as

the set originally contained a number of odd intermediate sizes, but the importance of a common ratio is now well recognized in England and is secured with substantial accuracy by using only a part of the original set.

3. The third set of screens here listed is by a well-known American maker. They have no common ratio between openings, but are considerably used in this country.

4. The fourth set is one lately recommended by the U. S. Bureau of Standards after much study and conference with makers and users of screens. The 0.062-mm. screen is almost identical with the I. M. M. 200-mesh screen and for practical purposes these screens coincide with other screens at a number of points.

To locate on the diagram (Fig. 1) the screens enumerated, we will refer to the logarithms in Table 1. The Tyler 200-mesh, since log. 29 =

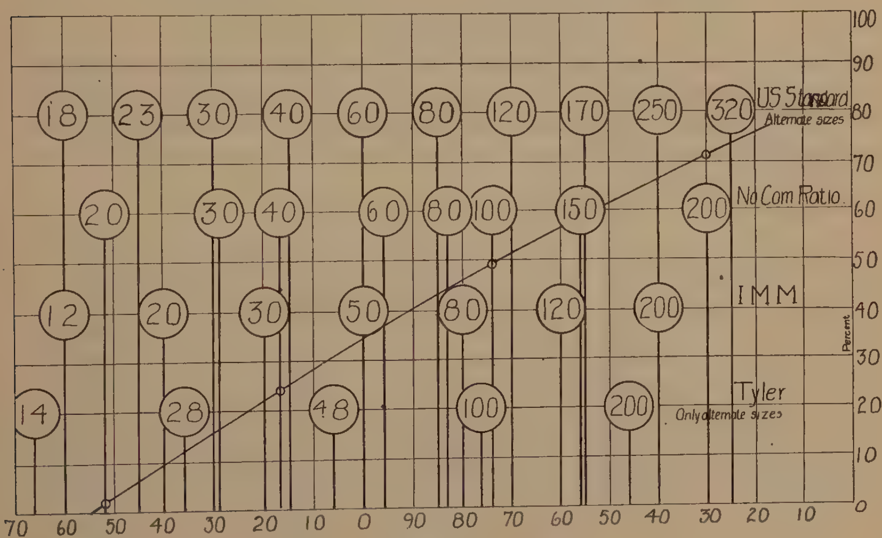

FIG. 1.

46, will be 46 scale units from the right index, and each succeeding screen will be 30 units farther to the left. In like manner, the 200-mesh I. M. M. screen will be at 40 and the 200-mesh screen of the set having no common ratio will be at 30 from the right. Dropping 4 from the first figure of the mantissa, we find that the logarithm, of the dimensions given in millimeters, exactly checks with the points already located, and we may proceed to locate the points for the fourth set. It will be noticed that the errors introduced to make the logarithms in exact arithmetical progression are slight. For instance, in the logarithm column for the new U. S. standard there is an error amounting to 0.412 scale unit at the last screen, which theoretically has an opening of 0.0625 mm., and this error

is thrown to the other side of the diagram by using the even figure 40 for placing the plotting scale instead of the more nearly correct 40.4834, the logarithm of the exact millimeter equivalent of 0.001 in. This, however, is considerably within the limit of tolerance allowed. It will be noticed that the indices of the logarithms change at each 100 mm. of the scale along the A-axis and that same distance along the B-axis of the diagram seems a most convenient range for the percentages, 0 to 100, of the material screened. This proportion between the dimensions along the two axes of the diagram is now much used and leaves nothing to be desired.

It is hardly necessary to add that the most common method of plotting is to locate the curve at the points along the B-axis at which all the material remains on a given screen, thus making the diagram cumulative. The curve drawn for illustration is from a stamp-battery product and only four screens were used.

It is probably too much to hope that everybody will eventually use the same screen standard, but the adoption of a plan giving the same curve for similar materials regardless of screen standards would be a welcome improvement. It is hoped that these suggestions will at least be helpful to those who might desire to change to a better screen standard and at the same time have a convenient means of comparing former work.

Table 1.—*Screen Openings*

Tyler Screens (W. S. Tyler Co., Cleveland)

Nominal Mesh	Opening in Inches	Logarithm of Opening Rounded Off	Opening in Millimeters	Logarithm (Metric)
8	0.0928	0.96	2.362	0.360
14	0.0464	0.66	1.168	0.060
28	0.0232	0.36	0.589	0.760
48	0.0116	0.06	0.295	0.046
100	0.0058	0.76	0.147	0.160
200	0.0029	0.46	0.074	0.860

I. M. M. Screens (Inst. Mining & Metallurgy, London)

Nominal Mesh	Opening in Inches	Logarithm of Opening Rounded Off	Opening in Millimeters	Logarithm (Metric)
12	0.0416	0.60	1.056	0.00
20	0.0250	0.40	0.635	0.80
30	0.0166	0.20	0.421	0.60
50	0.0100	0.00	0.254	0.40
80	0.0062	0.80	0.157	0.20
120	0.0042	0.60	0.107	0.00
200	0.0025	0.40	0.063	0.80

TABLE 1.—*Screen Openings.*—(*Continued*)

Screens Having no Common Ratio

Nominal Mesh	Opening in Inches	Logarithm of Opening Rounded Off	Opening in Millimeters	Logarithm (Metric)
10	0.0799	0.90		
20	0.0335	0.52		
30	0.0195	0.29		
40	0.0147	0.17		
60	0.0091	0.96		
80	0.0067	0.83		
100	0.0055	0.74		
150	0.0036	0.56		
200	0.0020	0.30		

Screens Recommended by U. S. Bureau of Standards

Approximate Mesh				
18			1.000	0.000
20			0.850	0.925
23			0.710	0.850
25			0.590	0.775
30			0.500	0.700
35			0.420	0.625
40			0.360	0.550
50			0.290	0.475
60			0.250	0.400
70			0.210	0.325
80			0.170	0.250
100			0.140	0.175
120			0.125	0.100
150			0.105	0.025
170			0.088	0.950
200			0.074	0.875
250			0.062	0.800

The Effect of Anti-friction Bearings on the Haulage of a Coal Mine

BY P. B. LIEBERMANN,* NEWARK, N. J.

(St. Louis Meeting, October, 1917)

1. *Haulage Tests on Coal-mine Cars*

THE haulage of coal from the face to the tipple is an important enough link in the production of coal to deserve its full share of study and care.

In order to obtain a better understanding of mine haulage conditions, a series of tests and investigations has been made which promises to hasten the improvement of present conditions.

For hauling coal through the mine and on the surface, locomotives are required, the size and power consumption of which depend on the resistance to motion offered by the mine cars. This resistance is principally caused by gravity and bearing friction. The effect of gravity can be reduced to a minimum by a suitable layout of the haulageways; that is, by arranging the grades in favor of the loaded trips, and by fixing the percentage of grade to that value which gives the lowest power consumption for loaded and empty trains. The bearing friction, which is considerably greater than is generally realized, can be easily reduced to a negligible quantity by the installation of a suitable anti-friction bearing that will stand the rough use and abuse of coal-mine service.

The present investigation will deal with flexible roller bearings in comparison with ordinary plain bearings.

A number of tests for determining train resistance have been run during the last 2 years both in the bituminous and anthracite coal regions. For all these tests the dynamometer car described in a previous paper was used. This car is equipped with recording instruments and is entirely automatic in its operation. It is coupled between the locomotive and train, and records the following quantities: drawbar pull in pounds, speed in miles per hour, and a time interval, usually 5 sec.; the paper travel is proportional to the distance covered by the car. Opposite these records the profile of the road is drawn in by hand so that full particulars

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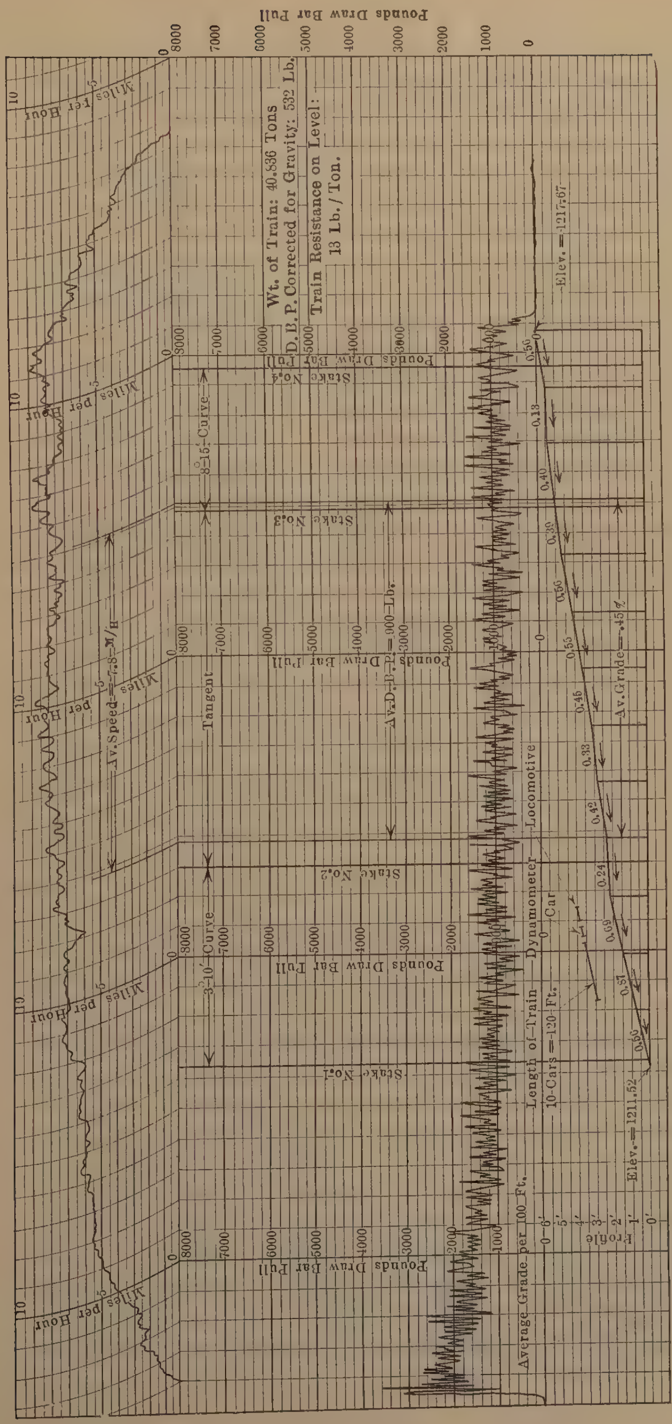

FIG. 1.—TEST ON 10 LOADED FLEXIBLE ROLLER-BEARING MINE CARS.

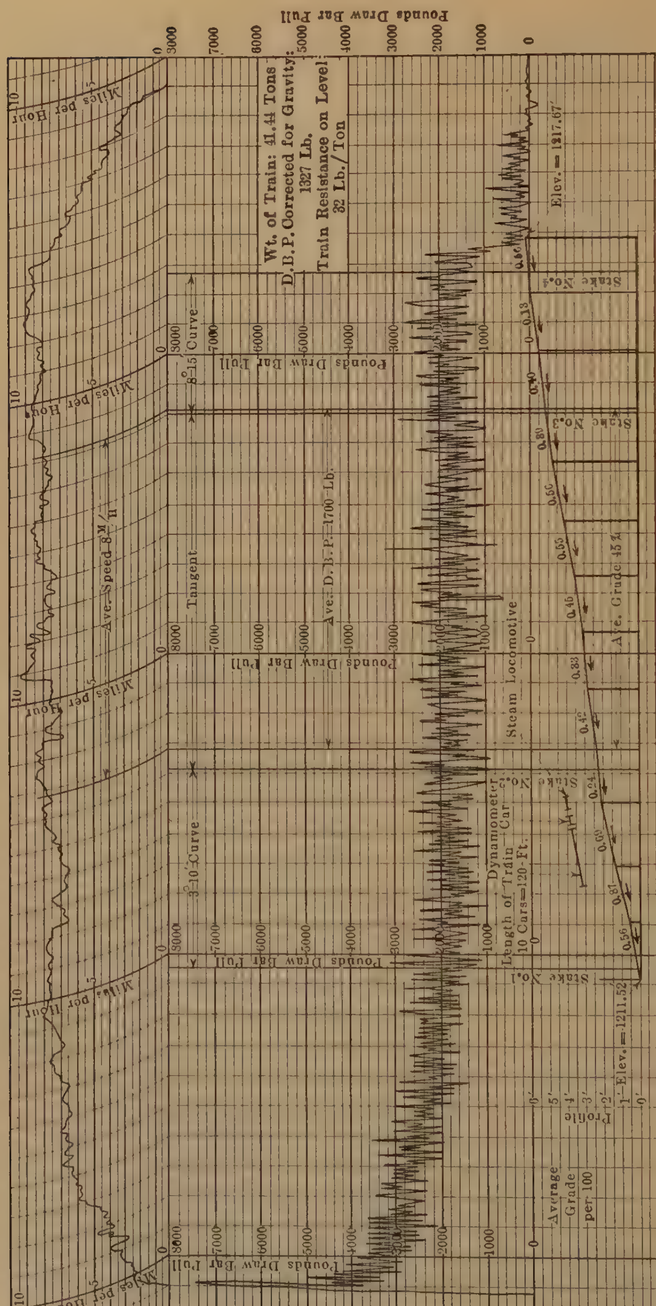

FIG. 2.—TEST ON 10 LOADED PLAIN-BEARING MINE CARS.

regarding train resistance for any grade or piece of track can be read from the chart at a glance. As an illustration, two charts are shown in Fig. 1 and 2 that were obtained at a comparative test made Nov. 17 to 20, 1916, at the D. & H. Co.'s Coalbrook colliery at Carbondale, Pa. The following types of mine cars were used:

Item No.	Type of Bearing	Location of Bearing	Diameter of Journal, Inches	Length of Bearing, Inches	Track Gage, Inches	Diameter of Wheel, Inches	Wheel Base, Inches	Drawbar to Top of Rail, Inches
1	Roller	Outside box	$2\frac{7}{8}$	4	30	16	36	14
2	Plain	Outside box	$2\frac{7}{8}$	4	30	16	36	14

These were wooden cars built by the D. & H. Co. in its own shops. They were identical in every respect except bearings. All wheels were loose and were held on the axle between a shrunk-on collar on the one side and the bearing box on the other side. All cars had side bumpers. When the test was made the cars had seen enough actual service to insure well run-in bearings. For lubrication a high-grade liquid grease was used compounded according to the D. & H. Co.'s own specifications. The plain bearings were self-lubricating by means of a felt pad in an oil reservoir; the roller bearings were self-lubricating due to their inherent hollow helical construction. No bearings received any overhauling or special treatment for any of the tests.

Each one of the various tests was run first with a train of cars having one type of bearing and was immediately followed by another test with a train of the same number of cars but with the other type of bearing and on the same piece of track and under identical conditions. This particular series of tests was witnessed by a large number of representative anthracite mine operators and engineers. The record charts are self-explanatory.

From the many tests conducted up to the present time, sufficient data have been collected to obtain a definite idea as to train resistance and its relation to speed.

An average value of these data can be expressed in the form of an equation. Any formula for train resistance consists of three terms as follows:

$$R = A + BS + CS^2$$

in which

R = train resistance in pounds per ton on straight level track.

$$\left.\begin{array}{l} A = \text{friction constant relating to bearing and rolling friction.} \\ B = \text{friction constant relating to speed.} \\ C = \text{friction constant relating to air resistance.} \\ S = \text{speed in miles per hour.} \end{array}\right\} \text{determined by experiment.}$$

On account of the comparatively low speeds encountered in mine service, the air friction is insignificant, and therefore the last term of the above formula, CS^2, can be dropped. The test data were found to meet the following equations:

$R_p = 26 + 0.45S$ = train resistance of loaded plain-bearing cars in pounds per ton.

$R_r = 9 + 0.45S$ = train resistance of loaded flexible roller-bearing cars in pounds per ton.

These formulas apply to average mine conditions, with clean tracks, with cars of from 2 to 4-ton capacity, and with wheels not below 16 in. in diameter.

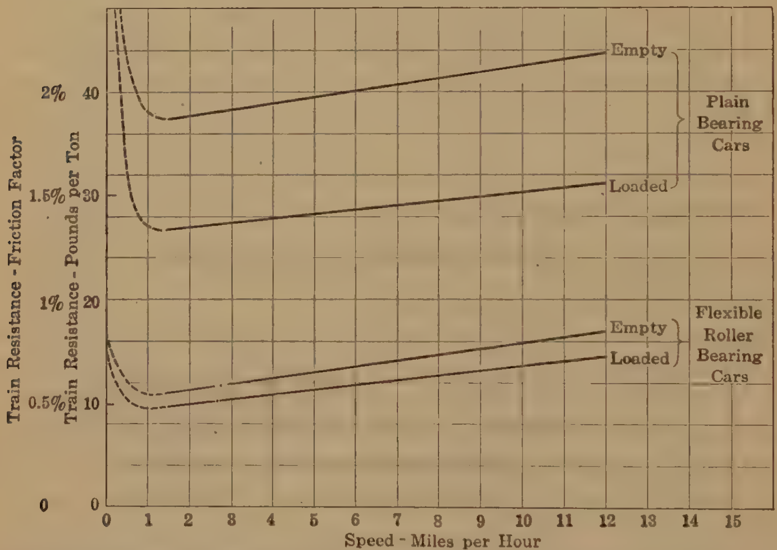

FIG. 3.—RELATION BETWEEN TRAIN RESISTANCE AND SPEED OF COAL-MINE CARS (AVERAGE FIGURES).

Tracks covered with coal dust were found to cause an increase in the train resistance of about 30 per cent. for either type of bearing. It should be remembered that this is the figure obtained with the dynamometer car behind the locomotive and that the locomotive, by rolling down the dust and loose coal particles, lessens to a large extent their effect on the train of cars behind it. The electrical input into the locomotive will, of course, be considerably increased over the 30 per cent., but this additional loss was not measured during these tests.

Tests made with empty cars showed that the train resistance per ton is about 40 per cent. higher for plain-bearing cars and about 15 per cent. higher for the roller-bearing cars than would be obtained from the formula for loaded cars.

Fig. 3 shows graphically the relation between train resistance and speed of loaded and empty cars. The train resistance is given both in pounds per ton and as friction factor. The friction factor is simply the percentage of the train weight required to keep the train in motion at any given speed.

If W_L = the weight of a train of loaded cars in tons and W_E = the weight of a train of empty cars in tons, the total train resistance in pounds on straight level track will be:

$$F_L = W_L \times R_L \text{ and } F_E = W_E \times R_E$$

When running on a grade the effect of gravity is taken into consideration as follows:

$$F_L = (W_L \times R_L) \pm (W_L \times G \times 20) \text{ and } F_E = (W_E \times R_E) \pm (W_E \times G \times 20)$$

G is the grade in per cent., while 20 is the added drawbar pull in pounds per ton per 1 per cent. of grade.

The grade also affects the tractive effort of the locomotive by the grade percentage of its own weight. If M = the weight of the locomotive in tons, then, for a loaded train of cars,

$$F_L \pm (M \times G \times 20) = (W_L \times R_L) \mp (W_L \times G \times 20)$$

and for a train of empty cars,

$$F_E \mp (M \times G \times 20) = (W_E \times R_E) \pm (W_E \times G \times 20)$$

The symbols have the following meaning:

F_L = total train resistance of loaded cars in pounds.
F_E = total train resistance of empty cars in pounds.
W_L = weight of a train of loaded cars in tons.
W_E = weight of a train of empty cars in tons.
R_L = train resistance of loaded cars in pounds per ton.
R_E = train resistance of empty cars in pounds per ton.
M = weight of locomotive in tons.
G = grade in per cent.
20 = gravity force in pounds per ton per 1 per cent. of grade.

2. *Test Data Commercially Applied*

The following comparisons are based on the ideal case of two mines, one equipped throughout with plain-bearing cars and the other with roller-bearing cars, each mine being laid out to the theoretical grade that will make the locomotive effort required to pull a train of loaded cars down grade equal to the effort required to pull the same train of empties against the grade. Such an example can easily be altered to

apply to any individual mine by simply making the necessary change of those items which differ from the example.

A 1,000-ton-per-day mine will be considered based on the viewpoint of reducing the cost of mining 1,000 tons of coal a day.

Some mine operators are also interested to know the quickest method of development or increase of output of their mines by equipping their cars with flexible roller bearings with no increased overhead except the additional wages required for the extra miners. An increased output is naturally limited by the length of landings, the accommodation of extra miners, the capacity of the gathering and haulageways, the capacity of cage and tipple, etc. There are too many different possibilities to make the working out of an example possible, but, fortunately, every mining operator will be able to determine for his individual case the possible increase of output by utilizing the figures and test data described above.

The following is a description of the 1,000-ton-per-day mine:

Output of coal per day	1,000 tons
Capacity of each mine car	3 tons
Number of car loads to supply 1,000 tons	333⅓
Weight of an empty car	2,500 lb.
Weight of a loaded car	8,500 lb.
Average distance per car per trip loaded	5 miles
Average distance per car per trip empty	5 miles
Average speed of trains	6 miles per hour
Working hours per day	8
Time for a round trip, hours	2
Round trips per day per train	4

To find the theoretical grade that will result in equalized locomotive effort in pulling a loaded train down grade and bringing the same train of empty cars up the grade, that is, $F_L = F_E = F$, the last two equations can be written as follows:

$$\frac{F - (M \times G \times 20)}{(W_E \times R_E) + (W_E \times G \times 20)} = \frac{F + (M \times G \times 20)}{(W_L \times R_L) - (W_L \times G \times 20)}$$

it then follows:

$$G = \frac{(W_L \times R_L) - (W_E \times R_E)}{20(W_E + W_L + 2M)}$$

In these equations the designations given below are used:

F = drawbar pull of locomotive on level in pounds.
M = weight of locomotive in tons.
G = percentage of the desired grade.
W_E = weight of a train of empty cars in tons.
W_L = weight of a train of loaded cars in tons.
R_E = train resistance of empty cars in pounds per ton.
R_L = train resistance of loaded cars in pounds per ton.
20 = gravity force in pounds per ton per 1 per cent. of grade.

Substituting for the train resistance the figures found by test for the speed of 6 miles per hour, the data shown in Table 1 are obtained:

TABLE 1

	Plain-bearing Cars	Roller-bearing Cars
Train resistance of loaded cars, pounds per ton	28.6	11.6
Train resistance of empty cars, pounds per ton	40.0	13.6
Grade for equalized drawbar pull, per cent	0.61	0.284
Number of cars per trip	42	42
Weight of coal per train, tons	126	126
Train resistance, loaded cars out, pounds	2,930	1,069
Train resistance, empty cars in, pounds	2,740	1,027
Train resistance, starting (estimate), pounds	4,000	1,200
Necessary weight of locomotive, tons	8	3
Number of locomotives	2	2
Foot-pounds per train per round trip	146,800,000	54,100,000
Kilowatt-hours per train per round trip	55.3	20.4
Kilowatt-hours all trips per day	443	163
Foot-pounds per locomotive per round trip	8,430,000	3,160,000
Kilowatt-hours per locomotive per round trip	3.180	1.195
Kilowatt-hours of two locomotives per day	25.40	9.55
Total kilowatt- hours per day of cars and locomotive	468.40	172.55
Total cost of power per day at 2 c. per kilowatt-hour at locomotive*	$9.36	$3.45
Total cost of power per year of 300 days	$2,810.00	$1,035.00
Lubrication material and labor per car per year as found in actual service	$1.73	$0.29
Lubrication 164 cars per year	$284.00	$47.50
Total cost of power and lubrication per year	$3,094.00	$1,082.50
Saving	....	$2,011.50

* The power cost of 2 c. per kilowatt-hour at the locomotive corresponds to a charge of approximately 1 c. per kilowatt-hour at the switchboard. This is due to the losses taking place in transformers, converters, transmission line, locomotive motor, rails, rail bonding, etc.

A clear saving of $2,011.50 per year has been accomplished through the application of flexible roller-bearings. These roller-bearing cars are necessarily more expensive than the ordinary plain-bearing cars. The 164 cars in the above example would require an additional investment of about $20 per car, or $3,280 based on prices prevailing at the present time. This amount is balanced by the saving in the size of the locomotives. Each one of the 8-ton locomotives costs about $1,000 more than one of the 3-ton locomotives, resulting in a saving of about $2,000. The 8-ton locomotive would require for the track rails weighing 30 lb. per yard, while the 3-ton locomotive could get along with 20-lb. rails. The difference in cost amounts to about $3,500 for the example based on $40 per ton and 5 miles of track.

The last item alone has wiped out the extra cost of the roller-bearing cars. On top of this should be figured the saving accomplished in the electrical transmission line, transformers, power house, reduced derailments, reduced wear and tear on cars and tracks, reduced size of repair-shop, etc.

Inasmuch as it is possible only in new mines and in a limited number of existing mines to arrange the haulage road for an ideal grade, the above example shall be supplemented with another example of a mine having a prevailing grade of, say, 3 per cent. in favor of load, and still another example will be given based on a prevailing grade of 3 per cent. against load.

Based on the same output, car weights, number of trips, etc., as before, the figures for the 3 per cent. grade in favor of load turn out as shown in Table 2.

TABLE 2

	Plain-bearing Cars	Roller-bearing Cars
Train resistance of loaded cars, pounds per ton	28.6	11.6
Train resistance of empty cars, pounds per ton	40.0	13.6
Grade against empty, per cent	3	3
Number of cars per trip	28	28
Weight of coal per train, tons	84	84
Train resistance going in, pounds	3,500	2,575
Train resistance starting, going in, pounds	4,200	2,660
Train resistance going out, pounds	−3,740	−5,760
Necessary weight of locomotive, tons	13	8
Number of locomotives	3	3
Foot-pounds per train per round trip	92,500,000	68,000,000
Kilowatt-hours per train per round trip	34.9	25.6
Kilowatt-hours all trips per day	419	307
Foot-pounds per locomotive per round trip	27,500,000	16,900,000
Kilowatt-hours per locomotive per round trip	10.35	6.37
Kilowatt-hours of three locomotives per day	124.0	76.5
Total kilowatt-hours per day of locomotive and train	543.0	383.5
Total cost of power per day at 2 c. per kilowatt-hour at locomotive	$10.86	$7.67
Total cost of power per year of 300 days	$3,260	$2,300
Lubrication, material and labor per car per year	$1.73	$0.29
Lubrication 164 cars per year	$284.00	$47.50
Total cost of power and lubrication per year	$3,544.00	$2,347.50
Saving	....	$1,196.50

To the saving of $1,196.50 should be added the saving due to the reduced size of locomotives and to the reduction in the weight of the rails. Each one of the 13-ton locomotives costs about $1,200 more than one of the 8-ton locomotives, resulting in a saving of $3,600. The 13-ton

locomotive would require rails weighing 50 lb. per yard against the 8-ton locomotives using 30-lb. rails which accounts for a saving of $7,000.

The figures for the 3 per cent. grade against load are shown in Table 3.

TABLE 3

	Plain-bearing Cars	Roller-bearing Cars
Train resistance of loaded cars, pounds per ton	28.6	11.6
Train resistance of empty cars, pounds per ton	40.0	13.6
Grade against load, per cent	3	3
Number of cars per trip	14	14
Weight of coal per train	42	42
Train resistance going out, pounds	5,270	4,260
Train resistance starting, going out, pounds	7,460	4,520
Train resistance going in, pounds	−350	−812
Necessary weight of locomotives, tons	25	15
Number of locomotives	6	6
Foot-pounds per train per round trip	278,000,000	225,000,000
Kilowatt-hours per train per round trip	104.5	85.0
Kilowatt-hours all trains per day	1,255	1,020
Foot-pounds per locomotive per round trip	52,800,000	31,600,000
Kilowatt-hours per locomotive per round trip	19.9	11.9
Kilowatt-hours of six locomotives per day	119.5	71.5
Total kilowatt-hours per day of locomotives and trains	1,375	1,092
Total cost of power per day at 2 c. per kilowatt-hour at locomotive	$27.50	$21.80
Total cost of power per year of 300 days	$8,250	$6,540
Lubrication, material and labor per car per year	$1.73	$0.29
Lubrication 164 cars per year	$284.00	$47.50
Total cost of power and lubrication per year	$8,534.00	$6,587.50
Saving	....	$1,946.50

Due to the reduction in the size of locomotives from 25 tons to 15 tons, a saving of at least $1,500 per locomotive, or $9,000 for the six locomotives, has been accomplished.

The lighter locomotives need 50-lb. rails against the 70-lb. rails of the heavier locomotives. This difference in weight accounts for a saving of about $7,000.

In determining the size of locomotives, no attention has been paid to the size of the tunnels. If any such locomotive should be of too large a size, then two smaller locomotives would have to be substituted, each locomotive pulling half the number of cars that could have been hauled by the bigger locomotive. The two smaller locomotives would cost more than one big locomotive. Besides this, there is the expense of the motorman and helper for the extra train. It is obvious that this handicap is particularly great with plain-bearing cars.

The main items are tabulated again for easy comparison:

SUMMARY

Ideal Grade:	
Saving in investment for locomotives	$2,000.00
Saving in investment for rails	3,500.00
Total saving in investment	5,500.00
Investment for roller bearings	3,280.00
Net saving in investment	2,220.00
Yearly saving in power and lubrication	$2,011.50
3 *Per Cent. Against Empties:*	
Saving in investment for locomotives	$3,600.00
Saving in investment for rails	7,000.00
Total saving in investment	10,600.00
Investment for roller bearings	3,280.00
Net saving in investment	7,320.00
Yearly saving in power and lubrication	$1,196.00
3 *Per Cent. Against Load:*	
Saving in investment for locomotives	$9,000.00
Saving in investment for rails	7,000.00
Total saving in investment	16,000.00
Investment for roller bearings	3,280.00
Net saving in investment	12,720.00
Yearly saving in power and lubrication	$1,946.50

For obvious reasons, these examples had to be reduced to the simplest possible conditions, and therefore the hauling in of timbers and supplies and the hauling out of rock were not taken into consideration in either case and no allowance has been made for the increased train resistance on curves, it being assumed that only curves of liberal radius were employed and that the grade was sufficiently eased up at the curves to compensate for the increased drawbar pull.

These items and others, the acceleration of the trains, for instance, will of course increase the total power consumption in each case, but they will not materially affect the relative proportion of the three examples.

The examples bring out forcibly the importance of proper grade. It has been clearly established that while considerable economy can be obtained through the introduction of roller bearings on existing grades, yet the maximum saving in the total expenditures for haulage can be obtained only by a proper arrangement of grades as shown in the first example. When heavy grades occur they are in most cases only of short length and with roller-bearing cars they can be taken flying, that is, the momentum of the moving train will carry the train over such grades.

It is safe to say that the flexible roller bearing will do more for the mine car than it has done for the modern automobile, because, besides its durability and certainty of operation, there are other features which

make it particularly attractive and profitable. To begin with, there is the saving in power which accounts for a smaller power bill, smaller locomotives, smaller transmission line and power house, lighter rails, less expensive rail bondings, less wear and tear on cars and tracks with fewer derailments, greater safety of operation, and steady and uninterrupted production. It will be noticed that only such items as directly affect the haulage cost and can be expressed in dollars and cents have been figured. There are other features, however, which cannot readily be figured, such as the effect on mules of easily running cars, where cars are gathered by mules, or the effect on the miners where cars have to be moved by hand. This is important in itself, as the human element more and more demands the elimination of all useless drudgery. The effect is very pronounced, too, when storage-battery locomotives are used.

The merits of storage-battery locomotives for gathering purposes and for short hauls are generally appreciated. Their battery capacity is necessarily limited and it is highly desirable that one charge of the battery be sufficient for one day's run. The two things affecting the battery capacity are grade and car journal friction. The grade quickly limits the use of storage-battery locomotives, while bearing friction places a serious handicap on their productiveness. A glance at the curves showing relative train resistance and a comparison of the kilowatt-hours given in the above examples will prove that a storage-battery locomotive with a single charge per day will be able to haul considerably more coal when flexible roller-bearing cars are used in place of plain-bearing cars. By the use of these figures it should be easy to figure out the possible production of any individual mine when full particulars regarding grade, length of haul, etc., are known. Not to be overlooked should be the heavy current discharges when starting a train of plain-bearing cars. These high starting peaks seriously affect the capacity and life of a storage battery. A locomotive pulling roller-bearing cars will not be subjected to such peak loads; on the contrary, it will pick up faster and consequently gather faster.

Discussion

P. B. Liebermann.—As mentioned in my paper, a special investigation was made on the effect of dirt on the track, for the express purpose of educating mine operators to the advantage of keeping their tracks clean.

As to the benefit of clean tracks to the saving in battery consumption, that can be easily determined if the rate of discharge is known under existing conditions, that is, on dirty tracks. The tests have shown that tracks covered with coal dust cause an increase in draw-bar pull of about 30 per cent. The battery discharge table will show the capacity corresponding to the new rate of discharge and will thereby disclose the possible saving in battery consumption.

The difference between dirty tracks and clean tracks will be found large enough to have an appreciable effect on the size of the storage-battery locomotive, and for a locomotive of any given size it may mean the difference between one charge and two charges per working day. While speaking of storage-battery locomotives, it might be well to remember that the factors determining the life of a battery are the number of charges and discharges and the peak loads. With clean tracks and flexible roller-bearing cars, only one charge per day will be necessary in the majority of cases, while at the same time the starting peaks will be negligible compared with the starting peaks occurring on dirty tracks or when ordinary plain-bearing cars are used.

In this connection, it is well also to remember that through my tests I have shown that the frictional resistance of empty cars with flexible roller bearings is only 10 per cent. greater than that of loaded cars, while with plain bearings the frictional resistance of empty cars increases 40 per cent., so that in reality it requires more power to move an empty plain-bearing car than it does to move a loaded flexible roller-bearing car.

Frank F. Jorgensen, Gillespie, Ill.—I would like to ask Mr. Liebermann, what is the highest speed at which he has tested these roller-bearing cars?

P. B. Liebermann.—The highest speed at which tests were made was 15 miles per hour on the surface and 6 miles per hour below ground. The last figure could not be exceeded because it is the legal limit.

Thomas G. Clagett, Bluefield, West Virginia.—I would like to ask Mr. Liebermann, what is the maximum grade on which a saving in power can be effected by using roller-bearing cars?

P. B. Liebermann.—If the grades in a mine are only of short length, they can be taken flying with the least possible effect on the power consumption. It is only when grades are of considerable length that their effect has to be studied, and it has been found that a pronounced saving in power consumption is possible up to grades of about 6 per cent. It is obvious, however, that a power saving occurs on any grade, but the steeper the grade the more the bearing friction becomes overshadowed by the effect of gravity; consequently, no great increase in train length or production can be expected when there is a continuous steep grade against the loaded cars, but with grades not exceeding 5 or 6 per cent., it is easy to deliver greater tonnage owing to the easy running qualities of roller-bearing cars.

Referring to the power saving on various grades, the paper brings out the fact that this is almost constant on grades against the load. It should be remembered that the saving shown includes that of both power and lubrication, but the saving in lubricant is a constant factor for any grade.

Coal Wastage*

BY FRANCIS S. PEABODY,† PH. B., CHICAGO, ILL.

(St. Louis Meeting, October, 1917)

THIS paper will not be a technical paper, because, although I have been in the business of mining and selling coal for 30 odd years, I am neither a mining engineer nor a practical miner. If I digress from time to time from my thesis, I must be forgiven by the expert engineers and highly technical gentlemen for whose proceedings this article is written.

Waste of the wonderful store of power that the Creator placed at our disposal appears to have been the theory and the basis of our methods of extracting this power—waste from the time the coal is mined to the time it is consumed. Let us consider the prime causes of this wastage which has been the heritage of the coal mining industry.

In the early days—about 150 years after "Stone Cole" was discovered in Illinois by Joliet and Marquette, and Father Hennepin noted a "cole" mine on his map—the mode of mining prevalent among farmer landowners, on whose property coal was discovered near the surface or outcropping on the hillsides, was to enter the seam by a shallow shaft or drift into the hillside and remove the coal by wedging it down until a large room was left. The roof without support would fall and our primitive coal operator would then sink another shaft or drive another drift. In these early operations only the large lumps were taken and all small pieces and screenings were left to be covered by the fall of the roof—in fact this condition prevailed in Illinois up to about 1885.

Gradually the method of mining changed from the open chamber, to a single entry with rooms turned to the right and left. Later, as the demand for coal increased and the necessity for larger mines became evident, the "room and pillar" system grew to be regular practice.

The room and pillar system of mining consisted of starting away from the shaft bottom or drift mouth, with a pair of entries usually 12 to 15 ft. wide and about 30 ft. center to center, one entry being used for the air intake and the other for the return air and haulage; from each entry rooms or chambers were turned, narrow at the mouth and widening

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to between 20 and 30 ft., these rooms being driven to a depth of about 200 ft. from the neck or mouth.

Practically all present-day methods and systems of mining bituminous coal in this country, except "longwall," are based on this system, and it is surprising to note how little improvement has been made over the old method even in what is considered the most approved system of today, the "panel" system.

Using the panel system the operator starts away from the shaft or drift mouth with a pair, or sometimes three entries; but instead of turning rooms off the main entries, "stub" or "panel" entries are turned. These panel entries are driven between 1,000 and 1,200 ft., and from them are turned rooms as the entries *advance*.

It is not my intention to dwell on the merits of our present methods, but rather to show their demerits.

As previously explained, the rooms in both the room and pillar and the panel systems are driven while the entries are also advancing, the first room reaching its limit of 200 or 250 ft., while the second room will be a little shorter, and so on successively up to the face of the entry where the last room will be just started. In Illinois mines of this type, the "room pillars," coal between the rooms, and the "chain pillars," coal between the entries, are left in the mine, generally through inability to remove them on account of the "gob" or refuse which has been placed alongside them while cleaning the coal taken from the room, and because of the fall of slate and roof resulting from insufficient and temporary timbering.

Nor is the coal left in the ground our only loss. The pillars prevent the overlying strata from subsiding evenly, in most cases breaking through to the surface, thereby spoiling the surface drainage, frequently leaving a previously level surface rolling and full of "sink" holes.

We know by visible evidence the conditions on the surface resulting from this uneven settlement; then what greater damage must be done to the thinner seams of coal and other minerals overlying the seam mined, and at present not considered valuable. Thus we are not only wasting our heritage but we are placing almost insurmountable difficulties for our successors to overcome to work the thinner seams.

As we study the old laborious methods of our fathers and compare them with present-day practice, we realize we have not improved coal mining methods at a pace equivalent with other industries. True, we have machines for mining and modern high-speed hoists at our mines producing 5,000 or 6,000 tons of coal per day, but our percentage of recovery is little, if any, higher than in early practice.

Some mining reforms have been forced into the industry by the lessors of coal lands. About the year 1890, automatic stokers began to come into general use, thereby creating a market for the cheaper grades of coal screenings, etc., which up to this time had been thrown away and left

in the mines. It is estimated that probably 57,000,000 tons of screenings had been wasted in Illinois.

With all credit to present-day coal operators, let me remark that most, if not all, mines are started with the view of recovering the chain pillar coal, at least, on the second mining, or as the mine retreats after having reached its boundaries and just prior to abandonment. I am afraid, however, it is mostly a case of "the spirit is willing but the flesh is weak," for I know of only one mine about to be abandoned which worked for nearly 2 years before abandonment recovering pillar coal and recovered approximately 350,000 tons out of possibly 4,000,000 tons left in the ground.

Another, and the most deplorable condition created by present haphazard methods, is the danger to life and limb. Our miners produce a great many more tons of coal per year per man employed than they do abroad, because we have fewer supervisors, and therefore naturally fewer men are employed per ton produced.

In Germany, there is a supervisor for every 15 or 20 men. In this country if we have one pit boss to 150 men we consider it enough; we must have closer supervision to get better extraction results and reduced loss of life and limb.

Foreign reports of 1912—the later reports being deficient on account of the war—show the loss of life in coal mines in Belgium to be 1 man for every 1,000 employed; in Germany, 1.54; in Great Britain, 1.17; in the United States, 3.35. Our wastage of life, considering these figures, cannot but be apparent. Our losses in life are more than three times as much as in the coal mines of Belgium where thin seams are all that is left and these are operated at 2,500 to 3,000 ft. below the surface and with a recovery of about 97 per cent. of the entire seam.

I have endeavored to show that the coal industry is beset by all manner of waste, waste of natural resources, waste of the human element and waste of capital, and we do not seem to realize how dearly future generations must pay for it.

It would be far better if a situation could be created in the near future with strong governmental control, preferably through the medium of the Federal Trade Commission, so that the bituminous coal industry could be thoroughly regulated with respect to the operation of present properties, so that all may operate on a reasonable basis, returning a fair percentage of recovery, with regulations that will insure the best conditions for the safety of life and limb, and so founded that the operator will be assured of a reasonable return on his capital invested.

Such regulation of the coal industry when it does come must begin at the bottom; the industry must be regulated from every standpoint. The governing commission must be assured that the prospective operator owns, or controls, sufficient coal land to permit a mine large enough

to produce a sufficient tonnage to return the investment in the surface plant and non-movable machinery, etc.—in other words, to wipe out the capital accounts.

The sinking of the shafts must be regulated. Our present law in Illinois regarding the sinking of shafts is adequate in its provision for the protection of life, and I believe is fair to the operator.

A system of mining whereby all, or at least 97 per cent., of the coal in the ground must be recovered, would have to be provided and rigidly enforced; a system providing for the protection of the now valueless thinner seams of coal and also for the protection of the other minerals, sandstones, shales and other rocks, which are now of no conceivable value, but may in future years be discovered to be a very important factor in some industry yet undeveloped.

All this regulation would necessarily involve much time and study and would gradually be revised as new conditions were met. Objections would be raised; no doubt many attempts would be made to prove the early acts unconstitutional; no doubt claims would be made, that only those controlling large amounts of capital could enter the business, and such would undoubtedly be a fact.

Let me say that the coal business is not a business for small capital; that is one of its greatest difficulties today. If we want a charter for a coal company now, we can get several dummy directors and pay a lawyer $50 or $100 and secure a charter for a full-fledged coal company, with the rights to sink shafts and produce coal, and incidentally waste 45 to 50 per cent. of the natural resources in the ground.

In Germany, where wages are much lower than in this country where machinery is less expensive, the investment in the coal business per ton of the annual production, not including any sums spent for mineral rights, for the government owns it all, is $2.50. It is a law, in this war-besieged country, that all the coal must be mined. In the States of Illinois and Indiana the investment per ton of the annual production is about $1.46 and we are mining only about half of the coal in the ground. When we consider the greater money value in Germany, it is apparent that coal mining in Germany is limited to those who are able to do business in a large way and with business-like methods.

I do not know of, nor would I attempt to specify, any particular system whereby all the coal can be recovered, but I do know that if the "retreating" system or something similar to it were adopted and so regulated that all coal operators must meet this standard, it would be a very desirable advance over our present methods.

In the retreating system, as it is talked of among coal men, main entries are driven to the boundaries of the coal property after which stub or panel entries are driven. The main body of the coal is extracted as the mine retreats to the shaft. To do this it is necessary that the mine

be sunk and worked for a number of years, probably averaging between 10 and 15, at a loss.

The surface plant would be fully equipped at a cost of approximately $300,000, taking the average mine of today as a standard. The coal rights, either owned or leased, would necessitate a carrying charge of say $25,000 to $30,000 per year. Until the boundary is reached by the main entries the output would be small, probably no more than 200 tons per day in the first year and about 1,500 tons per day the year the boundary is reached. From then on the tonnage would rapidly increase and the mine be on a paying basis immediately.

Every ton of coal taken from the entries would undoubtedly cost the operator not less than $2.50 to $3 per ton to produce and would probably sell, judging from present market conditions, for $1.50 per ton, leaving a deficit of $1 to $1.50 per ton produced, to be charged to the capital account. I do not wish to infer that our present entry coal costs this much to produce, but with the system outlined it would also be a part of the proposed scheme of things that all supports, such as we now call "timbering," would be of a permanent and possibly a recoverable nature. Our mine tracks would be laid of far heavier steel than used at present and all items of operation would be of a far more permanent nature than is now required.

Assuming we opened a mine according to this ideal standard, carried it through for a number of years until the entries reached the limit, all work done being of a permanent nature, with the losses on production capitalized up to the time when production reached a stage sufficient to put the mine on a paying basis, our investment, according to my general figures, would amount to $500,000. Our mines naturally being fewer in number than at present, our operations would be steady and our ideal mine would produce possibly 2,000,000 tons of coal per year. Then taking into consideration our sinking fund for all causes, we would have to be guaranteed not less than 20 c. per ton net profit to realize an equitable return on our investment.

In Germany, where regulations similar to those I speak of are in effect, persons entering the coal business must show unquestionable evidence that they are able financially to stand the enormous investment called for.

It is not unreasonable to assume that the investing public and the capital interests would not bear the same ill feelings toward the coal industry they now do, if they could be assured the business was so regulated that the possibilities of loss were minimized. It would not be any more difficult to call upon them for large sums to be invested in operating property than it is at present to secure capital for investment in coal lands.

I venture to say that many operators in the business today, if com-

pelled to keep their costs in a standard commercial manner, charging just and fair depreciation on their coal lands and their plants, would find that they are not making any profit; however, I maintain that it would not be against the interests of the public to exact a profit, for we would be saving the natural resources for the years to come, which, under our present method of extravagance, will be so minimized in the future that five and six times the profit I mentioned will be regarded as just.

By government control or regulation of our methods of mining and the natural following of the greater permanency of the work we do, our fatalities among coal mining labor would naturally decrease.

During the first 3 months of 1916, 259 men were killed in the United States through falls of slate and rock in our coal mines; or approximately two men lost their lives from this one avoidable cause, for every million tons of coal produced. This would be particularly safeguarded by our better and more permanent roof supports. With our roof supports more firmly installed, consequently fewer falls, fewer electric wires would be damaged and we could save a proportion of the 18 men killed in the first 3 months of 1916 by electric shocks. The pockets in the roof of the entries from which the loose rock has fallen would not be present to be a gathering place for gas, which caused the death of 82 men in the United States in the first 3 months of 1916.

We have many laws on our statute books which I freely say are good laws. Many have been passed to guard the safety of our workmen; very few to protect our natural inherited wealth. We have the Sherman Anti-Trust Law, which most of us have an idea means that competitors cannot combine for any purpose whatever, if the result of that combination is to restrain fair competition or if such combining results in the creation of a monopoly. In England this is common law; in Australia they have a similar law called the Australian Industries Preservation Law, which I believe is a far more pleasant term than Anti-Trust. Our own Supreme Court has interpreted the Sherman Law, "that people cannot get together and combine to create a monopoly or restraint of trade, when it is against the interest of the public to do so." The Australian Law says: "when it is against the interests of the public, you cannot combine."

Unfortunately, as yet there has been no case before our Supreme Court in which they could say the restraint would be unreasonable, so we do not know what facts would constitute a reasonable restraint, or a reasonable monopoly under our laws. In Australia they have had a case affecting coal companies, in which it was shown that there was ruinous competition between two coal fields, and the operators, to remedy conditions, got together and entered into contracts which would raise the price of their product. It was shown that the prosperity of their towns which were dependent upon the coal industry was seriously threatened and in the

operation of the two agreements the selling price of coal at the mines was increased 40 per cent. The House of Lords or the Privy Council in rendering their decision said "That it was impossible to disregard the interests of those who are engaged in production and distribution, and further that it can never be of interest to the consumers if any article of consumption should receive fair remuneration for the capital employed and the labor expended."

There can be no doubt in our minds that these agreements were entered into to raise the price of coal and to preclude competition. However, it was proved that the prevailing prices previous to negotiations of the agreement were disastrously low, owing to the cut-throat competition. Where these conditions prevail the less remunerative collieries will be closed down; there will be a great loss of capital, miners will be thrown out of employment, less coal will be produced and prices will consequently rise until it becomes possible to reopen the closed collieries.

The consumers will lose in the long run if the mine operators do not make a fair profit or the miners do not receive a fair wage; therefore, in the opinion of the English Privy Council, the mere intention of an agreement to raise prices does not always prove the intention to injure the public. To prove an intention to injure the public by raising the prices, the intention to charge excessive or unreasonable prices must be apparent. I, therefore, believe it is to the best interest of all, not only those in the coal industry but also those who have dealings with the coal industry, to advocate strongly the enactment of a law providing for combinations and agreements of the kind which will permit producers of natural resources to produce and market their products under a uniform cost-accounting system, and a uniformly regulated manner of production, safeguarding the natural resources of the earth from wanton waste, and returning to the men in the industry an amount commensurate to the value of their services to society. When I say the men in the industry, I mean the men who work with their hands, the men who are charged with the executive management, and the men who furnish the necessary capital and credit.

Discussion

The Chairman (Carl Scholz, Chicago, Ill.).—Coal wastage is undoubtedly one of the most prominent topics before the coal operators of the United States today. Even those who have been in the business only 25 years, like myself, know that the exhaustion of coal is something that should and must be curbed. If it is not, before long we shall discover that the great coal resources of the country are not what we thought they were; we have been so accustomed to work at the best and thickest coal alone, that it will be difficult for us to realize one

of these days—and that day is not far distant—that we are no longer the rich nation in coal that we thought we were.

In this connection I feel obliged to attach a little blame to the Geological Survey for misleading the people. There are several members here who assisted in that work, and I am going to ask them to defend themselves. The Geological Survey deals with coal seams of much less thickness than we could now expect to mine, particularly with the present high wages, excepting at a prohibitive cost. The Geological Survey, in its estimates, does not discriminate between thick and thin seams, but simply estimates our resources at so many hundred thousand billion tons of coal. I believe they include seams from 15 to 18 in. thick, but you all know that a vein below 3.5 or 4 ft., is not considered economical to mine at this time.

R. V. Norris, Wilkes-Barre, Pa.—Mr. Peabody has evidently been thinking almost entirely of Illinois conditions, with which I regret to say I am absolutely unfamiliar. In some West Virginia mines the total recovery at present is considerably over 90 per cent. and extractions as high as 97 or 98 per cent. are claimed.

The most serious point in Mr. Peabody's paper is one that I have been hammering on for a great many years; it applies particularly to the anthracite region, where the Government has attacked the ownership of coal by the great railroad companies and has in some cases forced the sale of their interests to individual operators. Railroad control, which is so much deprecated by the public and by the newspapers, has an influence on conservation that is stupendous. The railroads have been and are mining anthracite coal for the ultimate tonnage for their roads. They have been willing to mine anything that would pay the ultimate cost to market, even though possibly not paying the cost of the actual mining itself. How far this may affect the ultimate recovery was shown by an examination I made a few years ago of the properties of one of the large anthracite interests. I was directed to report the tonnage and value, both as a railroad proposition and as an individual enterprise. The results were rather startling. Considering the amount the individual would mine, while operating for the greatest profit, there was only about 60 per cent. of the tonnage that the railroad would mine. On the other hand, the individual's prospective profit was approximately double that of the railroad, with only 60 per cent. of the quantity.

The particular difficulty encountered in the anthracite region and in many bituminous districts is the interstratification of small beds with the large. It is manifestly to the advantage of an operator to mine the largest beds and leave the small beds to the future. If the large beds could be mined without injury to the small ones, it would be desirable to do so, from every point of view, not merely that of the

operator; but unfortunately, while mining the large beds, the smaller ones are frequently ruined, involving serious loss of coal.

It is very easy to criticise former methods of mining, as to the loss of coal, which we all deplore. But the fact remains that the profits on coal were so small that the operators could barely live by mining the very best of it and the cheapest; they could hardly be expected to mine coal for the benefit of posterity at a loss to their own pocketbooks, which would have been the case, if they had attempted to extract all the coal in the past.

The remedy seems to me to be along the lines suggested by Mr. Peabody; the forming of combinations and the fixing of a price at a point where economical mining becomes possible. It is a matter of finance. No corporation and no individual is going to mine coal at a loss. The last census returns analyzed by Mr. E. W. Parker showed that the entire bituminous region had made less than 1 per cent. on the actual capital invested, with no allowance for depreciation. Now that is not business, and the attacks of the Government on the operators are, it seems to me, thoroughly unjustified.

Mr. Peabody dwelt on the larger rate of accidents per man employed. Would it not be fairer to figure the accident rate per ton of coal produced, rather than per man employed, because of our much more intensive methods? The output is certainly measured in tonnage, not men, and the figures given by Mr. Peabody would be very different if computed as accidents per ton rather than per employee.

J. A. Udden, Austin, Tex.—As to the thickness of coal that it pays to mine at the present time, this varies in different parts of the country. I recall when miners were working coal seams in Kansas that were not more than about 20 in. thick; and at present a part of the seam which is worked in the Thurber mine, in Texas, is less than that. I believe we have other coal seams in the northern part of Texas that it pays to mine at the present time, although they are probably not more than 20 in. thick.

Edwin Ludlow, Lansford, Pa.—I can speak only of the anthracite region. With improved breaker practice and better jigs, the coal wastage has been reduced to an almost negligible quantity; the tests of refuse now being sent out from the breakers on the prepared sizes show that not more than 2 per cent. of coal is being lost in the present jigging practice, as against 10 to 20 per cent., and as high as 30 per cent., in the earlier breakers.

Incidentally, the coal wastage of the past has been one of the greatest benefits to the operators of the present day, as the recovery of the coal formerly wasted in the refuse banks is one of the large industries of the anthracite region, and has been of especial value at this time when the

demand for coal is in excess of the capacity of the mines to produce; with improved jigs and machinery, the old culm banks have been rehandled and the coal made available for use.

In the original preparation of anthracite, everything smaller than nut was thrown away, and the separation of the nut and larger sizes was very imperfect. With improved methods of burning anthracite, the smaller sizes have gradually increased in value, enabling the operator to save more and more of the output of his mine, until four sizes of buckwheat are now being made, the smallest size, No. 4, being coal that will pass through a $\frac{1}{16}$-in. mesh and over a $\frac{1}{32}$-in. This coal, while difficult to use directly on the grate bars, is found to be very valuable mixed with bituminous slack, as the coking qualities of the bituminous prevents the loss of the fine coal, either through the grate bars or by being blown away by the draft, and the high percentage of fixed carbon increases the heat value of the bituminous.

With improved grate bars, especially of the automatic stoker type, No. 3 buckwheat, made through a $\frac{1}{8}$-in. mesh and over a $\frac{1}{16}$-in., has been found capable of producing 200 per cent. boiler rating at the large electric plants, where it is now used almost exclusively as a fuel.

The fine dust which now passes through a $\frac{1}{32}$-in. mesh is being refined and used in the manufacture of briquettes, using 7 per cent. of 180° oil. The egg-shaped briquette is well adapted for domestic purposes, being between the sizes of nut and stove coal, and is smokeless, odorless and weather-proof, while it stores and handles fully as well, if not better, than anthracite coal.

In the methods of underground mining, much improvement has been made, not only in the methods of working the larger seams, running up in some cases to 100 ft. in thickness, but also by the adoption of coal-cutting machines and face conveyors in the thin seams, from 2 to 4 ft.; these were formerly considered not thick enough for profitable work, but they are now being mined in large quantities, with good financial results.

H. H. Stoek, Urbana, Ill.—I wish to emphasize two of the points Mr. Norris has brought out.

In the first place, percentage of extraction is purely an economic question. I have for many years resented the slurs often made against American coal-mining engineers, that they were not attaining as high a percentage as in Europe. In western Pennsylvania they are getting at least 90 per cent. of the coal in the ground, and have done it in other places, when it has paid to do so. But coal men are not in business primarily for the good of the community. One factor that discourages economy is the extremely low price at which much of the coal has been bought. It is difficult to impress upon a man the importance of saving something,

when he has paid maybe only a tenth of a cent for it; much of the coal in Illinois has been bought at from 0.1 to 1 c. per ton in the ground. I heard one operator, who bought land a great many years ago, say, "What is the use of including in the present cost the price that I paid for that coal, so long ago I have forgotten about it?"

A great many people are deceiving themselves as to the percentage of coal extraction because many of the published figures are based merely on the thickness that is being mined, not on the actual thickness of the coal seam. For instance, in mining a 10-ft. bed, 2 ft. of top coal may be left; but the calculations of extraction that you generally see are based on the 8 ft. mined instead of the 10 ft. actually in the ground.

Another factor has been the extreme ease with which a mine could be opened. In Illinois, 10 years ago, there were nearly a thousand operating mines. At present, in spite of the increased output, there are several hundred mines fewer, which means that a large number of small mines, which could produce coal only under the most favorable conditions, have had to close down, and this will make for higher extraction.

Robert W. Hunt, Chicago, Ill.—I do not mine coal, but I feel very deeply on this conservation movement, and as our paternal Government has taken up the question of coal regulation, I hope, personally, that its activities will not cease until it reaches to the very bottom of the proposition.

Now, to my mind, it is criminal to throw away coal; I am looking at it from the humanitarian's point of view. If a mine cannot be operated profitably while extracting all the coal in its ground, that coal ought to be let alone until the time comes when it will pay, or until a method is developed by which it can be mined profitably. Price regulation is going to play an important part in that matter, I believe.

We think we know that no more coal is being created and therefore every ounce of it ought to be conserved. The operators are not to blame for past practices, economic conditions have been to blame, and those conditions ought to be changed. It has been said that we are not interested in posterity. I do not believe that. We owe a duty to posterity, and the conservation of that which cannot be reproduced is a great part of that duty.

Eli T. Conner, Scranton, Pa.—The operator of a generation ago was confronted by conditions of the market, of transportation, of cost of production, and other factors, which, unless we study history carefully, we do not always fully appreciate. All engineers who have examined former mining practices in this country have been appalled at what seem to us like very wasteful methods, particularly underground. This applies in the anthracite as well as in the bituminous region. But when we of

this day criticise our forefathers for wasteful methods, it is well to reflect on the difficulties under which they labored.

There is a limit to which we can approach in the ultimate recovery of coal. I think that the aim of all engineers and operators should be 100 per cent. recovery. Of course that is impossible, but we should try to attain it, and wherever possible we should leave the remainder, which cannot now be profitably recovered, in such condition that future generations may enjoy what has been left by this generation.

T. H. Clagett, Bluefield, W. Va.—Mr. Peabody's paper should not be construed as applying to the coal-mining industry throughout the country, since there are many mining districts in which very great improvement has been made over former methods and in which wasteful practices are now the exception, not the rule. In the Pocahontas coal field of West Virginia, efforts of the operators to correct such wasteful practices as existed have met with marked success. Areas that had been temporarily abandoned before room pillars were mined have been reopened, refuse due to the cleaning of the coal and falls of slate has been removed where necessary, and all such pillars are being recovered. In some mines the work is confined almost entirely to entry driving and the recovery of room pillars with a view of completing all such pillar work before room driving is continued.

The plans followed in new work admit of many modifications in detail but in general require that main entry chain and barrier pillars shall be of such size that they can be successfully removed at the proper time, and room development is so arranged as to permit continuous mining from the time the room is begun until the room pillar has been completely removed, resulting in concentration of workings, economy of operation and the recovery of a greater percentage of the coal. The uniform and continuous advance of the line of roof fracture following systematic removal of pillars minimizes the damage to coal seams lying above, and to the surface. Such complete mining can be accomplished as the workings advance to the boundary of the property as well as retreating from the boundary, with every opportunity for a large recovery and as little loss as would have been afforded if mining had been confined to entries until the boundary was reached. When the mining advancing is confined to one-half the area, the remaining area being mined retreating, a uniform output may be maintained during the entire life of the property. Many operators are following these methods consistently with very satisfactory results.

I have information in detail regarding some 90 mines worked by 42 independent operating companies, the first of these mines having been opened in 1883, the others having been opened from time to time to the present. While the average recovery of coal from the area mined over

annually rarely exceeded 75 per cent. prior to the year 1900, the average recovery from all of the mines from the beginning of operations to and including the year 1916 was 87.5 per cent. of the coal from the area mined over, 42 per cent. of this area being pillars; and for the 6 months ending Dec. 31, 1916, the average recovery was 90 per cent. of the coal from the area mined over, 48 per cent. of this area being pillars.

F. W. Sperr, Houghton, Mich.—To my mind, the percentage of extraction in mining is not by any means altogether an economic question, but very largely, if not altogether so, an engineering problem to the solution of which we should devote ourselves assiduously. I take it to be the function of the engineer to show how the proposition can be made economically desirable.

In Ohio, in the early days, the best recovery was scarcely 10 per cent. with their single-entry diggings; but, as the demand for coal increased, extraction was gradually advanced to 40 per cent. and for many years this was considered good work. Then, along in the early 80's, a certain corporation owning large tracts of coal land, and leasing to operators, was employing an engineer in another department who became interested in the fact that the company was getting only a small percentage of the maximum royalty from its coal lands. The result, in short, was the establishment of a new relationship between the land owners and the operators. The former, as the party of the first part, were to keep up monthly surveys of the mines and to make the annual maps required by the State. The latter, as the party of the second part, were merely required to work the mines in such workmanlike manner as to secure the highest percentage of extraction. Then, in case the party of the second part failed to perform his part of the contract, in the estimation of the engineer, the lease was forfeited and all rights, improvements, and equipment reverted to the land owners. In a few years extraction had advanced to 80 per cent., and it was not long before not only the operators on the lands of this particular company, but also their neighbors both on leased lands and on their own lands, were getting 90 to 95 per cent. and nothing under 96 per cent. was considered good.

But, in some way, the influence of the depression of '93 and following years seems to have spoiled all the good accomplished. I was down there again a few years ago, and was told they had reverted to about 50 per cent. extraction; and the reason given was "They are begging us to take leases now on any kind of a proposition."

The main point is that in the interval while extraction was being increased from 40 up to 96 per cent., the price of coal went down—not up—and the operators and engineers learned to work together to produce coal cheaper, with improved methods, for the benefit of everybody from the producer to the consumer. Hence, I say, it is largely an engineering

proposition—you must not blame it on the economist altogether if our mineral resources are in large part destroyed.

FRANK F. JORGENSEN, Gillespie, Ill.—It is not always possible to avoid leaving much coal in the ground. In our mines we have to leave at least 50 per cent. to support the surface. If we take out more, by pulling the pillars and letting the ground squeeze, there is absolutely no limit to the price we have to pay for the damage in spite of the fact that in our part of the country the land is not worth a great deal for farming purposes. For instance, we squeezed an area of not more than a quarter of an acre, and at the trial the jury awarded the owner $1500 for it; the coal we took out from under that quarter of an acre would not pay for the court costs. I cannot agree with the previous speaker, that it is an engineering problem in our field.

H. H. STOEK.—It may interest this meeting to know that our Chairman, until a few months ago, was operating mines in northwestern Illinois which yielded the highest percentage of extraction yet obtained in the middle West. He may be willing to tell us about that.

CHAIRMAN SCHOLZ.—What Mr. Stoek says is true. We have operated a mine in Illinois for the last 10 years, from which 97 per cent. of the coal has been extracted. Adjacent to us is a mine which extracted only 50 per cent. The difference in that case was simply, as Mr. Sperr has rightly indicated, a question of engineering. Our workings were laid out in a professional manner, and were kept that way by the perseverance of our mine management. Our manager, Mr. Lee, carried his lines right to the face, so that we have been able to draw all of our pillars, and by actual weight and measurement have obtained 97 per cent. of our coal. The mine maps are in the hands of the Geological Survey of Illinois. Our cover was very thin, but we had very valuable farm land overlying the mine. Geological conditions were favorable, because there was a heavy rock overlying the coal, which broke over a considerable area.

R. D. HALL, New York, N. Y.—The main difficulty in Illinois and Indiana, which States probably waste as much coal today as any other mining regions of this country, is the fact that the wage scale and other operating conditions are based on previous bad practice, which the union is endeavoring to perpetuate. The law requires the operator to put in frequent crosscuts. This prevents the leaving of large pillars and makes it hard to get out a good percentage of the coal. Of course, if your cover is thin, you may be able to extract a high percentage by careful, systematic mining; but systematic mining will not help you when your cover is heavy and frequent crosscuts are demanded, for which class of work you have to pay quite a large additional price.

In West Virginia, the operators manage to get their rooms driven narrow and their crosscuts long, and thus they are enabled to leave good

pillars; their percentage of first recovery is small, but their ultimate extraction is quite large. This is because their wage scale and the law permit such methods of extraction.

The high price of coal land, as has been suggested, has much to do with the saving of coal in the Connellsville region and in West Virginia, but that is not the only reason. In those non-union districts it has been possible to adopt certain plans, to put them into effect, and to carry them out thoroughly, and so by a willing or unwilling coöperation between the men and the management, the coal has been mined by the best possible methods. I do not believe that the success in the Connellsville and West Virginia fields is attributable entirely to a higher intelligence among the operators and engineers. The success is largely due to the fact that the existing labor and legal conditions made it possible to adopt improved methods and to carry them out successfully.

The coal producer is not the only one who wastes coal; the consumer is every bit as wasteful, but I do not know any reason why the producer of coal should be requested to exercise any more care than the user of coal. It is always a debatable question how much time, labor and material we should be allowed to waste in the effort to economize. Moreover, the coal producer has just as much right and reason to be wasteful as the man who burns the coal prodigally under a boiler, or keeps his electric light blazing all the time, or uses electricity for advertising in a big city. If the Government is going to restrict the coal producers so closely, it ought also to restrain the consumption of that precious coal so that it will not be wasted. I think that we shall find it difficult to accomplish this, as somehow the liberties of the consumer always seem so big and those of the producer always seem so small.

E. A. Holbrook, Urbana, Ill.—Another possible source of economy will be found in the saving of iron pyrite found in coal. As this matter comes up before the War Minerals Committee, I shall only refer to it briefly. The sulphur occurring in coal seams has been a total waste to the present time. Probably 50 to 90 per cent. of the sulphur encountered in the average coal mine does not come to the surface. The supply of pyrite in this country has heretofore come largely from Spain, and was consumed for the manufacture of sulphuric acid along the Atlantic seaboard. The present freight conditions are compelling us to look for our pyrite in America. The objection to pyrite recovered from coal seams for acid manufacture is that the presence of much carbon discolors the acid. If the pyrite, however, could be concentrated up to 40 per cent. sulphur—pure pyrite containing 53 per cent. sulphur—it should sell for at least $8 per ton, or practically four times as much as coal. I have a feeling that in the central West, in connection with the coal industry, we ought to develop a permanent pyrite-producing industry.

Steam-shovel Mining of Bituminous Coal*

BY H. H. STOEK,† B. S., E. M., URBANA, ILL.

(St. Louis Meeting, October, 1917)

Conditions Favorable to Steam-shovel Mining

The fundamental reasons underlying the choice of a method of mining a coal seam are safety of operation, cheapness of producing the coal and the character of the product as a saleable article.

From the standpoint of safety, open or strip coal mining is always preferable, as many of the dangers inherent to underground work are absent, such as explosions of gas and dust, falls of roof and coal, and also to a great extent those due to haulage and the use of explosives, thus eliminating three of the chief causes of accident from which about 75 per cent. of the underground accidents in coal mines occur. The conditions under which coal is stripped also make the work much safer than ordinary quarry work, as the depth of the pit is usually comparatively small.

From the standpoint of low mining cost, there is much misapprehension and many of the costs given are misleading in that they do not take into account such items as the destruction of the surface for future use, the full cost of the removal of the overburden, depreciation, amortization, coal depletion, loss of time through floods and from bad weather in winter, etc. Coal is no doubt often put on the cars at less cost by stripping than by underground mining, but by no means always so, as is too generally assumed. In some cases coal must be obtained by stripping or not at all, because the small amount of overburden renders underground mining impossible.

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* This paper was prepared at the suggestion of the chairman of the Coal and Coke Committee, and at the outset the writer wishes to acknowledge the coöperation of a number of members of the Institute and others in furnishing much of the data which he has attempted to edit and correlate. Wherever possible due credit is given in the text.

Especial assistance was furnished by Eugene McAuliffe who had begun a paper on the Kansas stripping fields for the St. Louis meeting and who very kindly offered to incorporate his data in the more general paper.

† Professor of Mining Engineering, University of Illinois.

The cost of opening up a strip mine is unusually heavy. According to J. A. Swanberg of the Carbon Hill Mining Co., the amortization of this charge sometimes amounts to as much as 15 c. per ton as against 3 to 5 c. in many underground operations.

The condition of the marketable product in competition with coal mined by the ordinary methods is a subject that has not received great consideration and is one upon which there will probably be a wide difference of opinion.

C. E. Lesher in the U. S. Geological Survey, *Mineral Resources* for 1915, says:

"Stripped coal does not, however (except locally), bring so high a price in the market as other coal from the same bed, mining from under deeper cover. The average value per ton for the pit coal from Illinois was 10 c. and for Indiana 20 c. per ton less than the average for the State. The difference in Kansas was 8 c.; Missouri, 7 c.; and Ohio, 17 c. The Oklahoma pit coal averaged 7 c. higher than the output of the State."

For the approximate values of pit coal for 1915, see Table 2.

Dr. F. C. Honnold, Secretary of the Illinois Coal Operators' Association and an operator of both stripping and underground mines suggests that a deduction of 15 c. per ton must be made for the inferior quality of stripped coal because it is obtained near the outcrop. In the Kansas field, Eugene McAuliffe says:

"Coal found under cover of less than 15 ft. (4.5 m.) in thickness is usually soft and low in volatile and sulphur content, high in fixed carbon and moisture, and is not adapted for steam-making purposes, but is consumed by the zinc smelters for fluxing purposes, being sold under the trade name Dead Coal, on account of its inert qualities shown when attempts are made to burn it."

It is the opinion of those connected with the boiler plant of the University of Illinois, where large quantities of Illinois stripped coal have been used, that it requires a larger quantity of this coal to produce a given amount of heat, than of coal from the same region produced by underground mining.

The evidence on this point for other districts is scanty and a discussion of this phase of the subject will be welcomed. If the stripped coal has been subject to weathering, it would naturally be expected to be of slightly lower grade than coal in the same bed found under heavier cover, so that weathering could not take place, but unless weathered, there is no reason why stripped coal should differ from any other coal. In other words, the deterioration is due to the method of occurrence and has no relation to the method of mining.

The conditions under which bituminous coal has been stripped to date are very different from those that exist in connection with stripping of anthracite; indeed, the conditions have been almost the opposite, for in

the anthracite region the beds have been thick and often steeply inclined, while in the bituminous region the beds have been usually flat, are in many cases quite thin, and seldom over 6 ft. (1.8 m.) thick. In connection with anthracite, the dumping area is usually ample and the destruction of the surface for farming purposes does not need to be considered, while some of the bituminous strippings are in very fertile farming country.

The geological conditions that render stripping possible may be briefly given as a bed of sufficient thickness, at not too great a depth, and covered by soft or comparatively soft material. The thickness of coal that is minable is determined by the cost of mining, the quality of the coal, and the price obtainable, which will vary with market conditions. At the present time beds varying from 20 in. to 7 or 8 ft. (0.5 to 2 m.) are being stripped. The thickness of overburden removable depends upon the machinery available and at present the maximum seems to be between

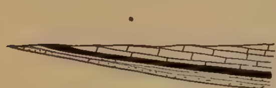

Fig. 1.

Fig. 2.

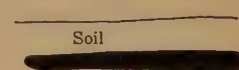

Fig. 3.

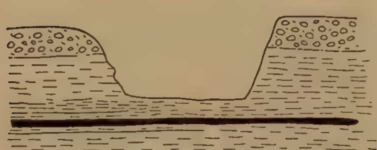

Fig. 4.

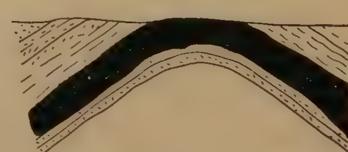

Fig. 5.

Figs. 1 to 5.—Geological occurrences.

40 and 50 ft. (12 and 16.7 m.), but no one can say that steam shovels have reached their maximum development. Comparatively hard shales are easily handled by modern steam shovels, but solid rock, such as limestone occurring in the overburden and requiring blasting, seems to offer a difficulty that in a number of instances has rendered stripping unprofitable.

The geological occurrences of beds that render stripping possible are:

1. A slightly inclined outcropping bed under a comparatively flat surface, Fig. 1. This is the condition in the southeastern Kansas field, where, as noted elsewhere in this paper by Mr. McAuliffe, there are two seams, one varying from 33 to 42 in. (0.8 to 1 m.) in thickness and the other 22 to 25 in. (0.55 to 0.6 m.) in thickness with 65 ft. (19.8 m.) of material between the seams. The dip is small and the cover varies from 8 to 40 ft. (2.5 to 12 m.) of soft slate, shale, clay, and a soft fissured sandstone.

Fig. 2 shows a similar condition where the coal bed may be flat or slightly inclined and where the amount of stripping possible is determined by theslope of the hill.

According to J. H. Sinclair, in the *Colliery Engineer* for June, 1913, there is a bed of coal 10 ft. thick at Tofield, Canada, occurring under a rich prairie loam on a dry sub-soil, which is sometimes from 10 to 15 ft. (3 to 4.5 m.) deep (Fig. 3). The coal area is estimated at 20,000 acres which will give 170,000,000 tons available for stripping.

Fig. 4 shows a condition found in eastern Illinois (Danville district), where a horizontal coal bed was originally overlain by a considerable thickness of shale and other rock, but small areas have been eroded by streams so as to leave a stripping area surrounded by bluffs.

There are no published records of thick steeply dipping bituminous beds or anticlinal outcrops where stripping is being carried on and such as are common in the anthracite strippings (Fig. 5), but in southern Illinois, near Duquoin, there is said to be a considerable stripping area where the No. 6 coal comes near the surface as a result of the so-called Duquoin anticline, which is more properly a monocline, as only one side is inclined to any extent. A small stripping operation has been carried on in this locality for some years and extensive ones are projected for the near future. The coal is from 6 to 7 ft. thick and occurs under cover varying from 11 to 41 ft. on the stripping area in question.

Distribution of Bituminous Strippings by States and Geological Conditions That Have Rendered Stripping Possible

To determine the area in which stripping is now being carried on, or areas that are available for such stripping, and the geological conditions that render stripping possible, letters were sent to a number of State geologists, heads of mining departments and individual operators, and the following is a digest of the replies received:

Alabama

E. A. Smith, State Geologist, says:

"A good many years ago this kind of work was done on some considerable scale in Tuscaloosa County, where the conditions were favorable, as follows: The occurrence of coal in the Brookwood group high upon the hills with slight cover and the distribution of the group of the Warrior Basin coals may be seen in the Alabama reports on Warrior Basin Coals."

C. H. Nesbitt, Chief Mine Inspector, says:

"There are two stripping operations in Walker County, one located at Carbon Hill on the Jasper seam, being stripped by the Brookside Pratt Mining Co. and another in Walker County, 10 miles (16 km.) north of Jasper, operated by the Sunlight Mining Co. on the Jefferson seam. The geological conditions which render stripping possible are a flat seam with approximately 30 ft. (9 m.) of cover and not much sand rock, the roof being principally slate at Carbon Hill. At Jasper there is sandstone. I can hardly estimate the area being stripped and the coal tonnage available."

The 1915 Mine Inspectors Annual Report gives the output of the Sunlight mine as 1803 tons and 400 tons used at the mines. The mine worked only 40 days during 1915.

J. B. Baddil, of the Sunlight Mining Co., writes on April 17, 1917, that the Sunlight Mining Co. is using a 250-B Bucyrus revolving shovel with a 6-yd. dipper. The company has stripped about 30 acres and has from 7000 to 10,000 acres of coal land available. The thickness of the seam averages 26 in. and the overburden about 25 ft.

Colorado

R. D. George, State Geologist, says:

"There is no stripping of bituminous coal in this State at the present time. A few years ago there was some done in Routt County and a little in Gunnison. I think that was discontinued several years ago. In the instances referred to the coal was overlain by a very much disintegrated shale and in places by stream detritus. None of this stripping was done with a view to production of coal on a large commercial scale, but for local use."

Illinois

There are four different stripping areas in Illinois, as follows:

Danville District.—West of Danville, Vermilion County, and near the Indiana border is the district in which so much of the development of the modern steam shovel has been carried on as noted on page 526.

The No. 7 coal seam, although usually too deep to be stripped, in this district in a number of instances now has an overburden of 15 to 45 ft. (4.5 to 13.7 m.) due to the fact that streams have eroded valleys and it is in these valleys that the strippings are mainly located. As the coal goes under the hills surrounding these valleys the mining operation is usually a combination of a stripping pit and an underground mine worked by the room and pillar method. The coal is from 5½ to 6 ft. (1.67 to 1.82 m.) thick and yields about 10,000 tons per acre. In this district are a number of brick plants and at a number of them the shale overlying the coal is also mined with the steam shovel and utilized in the manufacture of brick, the underlying coal furnishing fuel for the plant. The brick companies are not generally shippers of coal and mine only for their own use and most of the stripping in the district for coal to be shipped is done by Messrs. Hartshorn, under the several names of the Carbon Hill Coal Co., Missionfield Coal Co., Two Rivers Coal Co. These companies own three shovels, two of which are at present operating, and the third of which will soon be put into operation. These are large Marion shovels, two being type 270, one having a 5-yd. dipper and 90-ft. boom, one an 8-yd. dipper and an 80-ft. boom, and one (type 250)

a 65-ft. boom and 3½-yd. dipper. One of these is at present operating in the Two Rivers mine, another at the Carbon Hill mine, and the third at the new Mission mine.

The stripping plant of the Western Brick Co. in the Danville district is of particular interest on account of the variety of operations carried on. The glacial surface till, varying from 5 to 10 or more feet in thickness, is first hydraulicked, and then the shale from 20 to 30 ft. (6 to 9 m.) thick is mined with a standard Bucyrus No. 75 steam shovel and used for manufacture of brick, and finally the underlying coal, 5 to 6 ft. (1.5 to 1.8 m.) thick is mined and loaded by hand and used for fuel purposes at the brick plant. The coal is shot loose by charging 1½ lb. (0.68 kg.) of powder in 2½-in. (63.5-mm.) vertical holes, a row of which is drilled 6 to 8 ft. back from the face and about 12 ft. apart. The cost of stripping under these conditions runs up to 40 or 50 c. per ton of coal on the cars.[1]

The Western Brick Co. also has a Marion model No. 250 shovel operating at its No. 3 plant south of Danville.

Southern Illinois.—Near Marion, Williamson County, the new Enterprise Coal Co. has about 60 acres (24 ha.) of stripping land covered by 18½ ft. (5.64 m.) of cover above the No. 6 coal. A Marion 250 shovel with 3½-yd. (3.2-m.) dipper and 75-ft. (22.9-m.) boom is used for the stripping and a model 36 shovel with 1½-yd. (1.37-m.) dipper is used for loading. The stripping area is an isolated patch of coal and although it is near the outcrop the conditions are not typical outcrop conditions.

Near Duquoin, Perry County, there was for several years a small stripping property worked by a very primitive shovel, which is now not operated. There is, however, considerable stripping activity in connection with lands north and east of Duquoin and in a short time two of the largest type modern shovels will be operating. Messrs. G. W. Dowell and Wm. LaFont of Duquoin estimate that there are some 600 acres in the neighborhood of Duquoin underlain by 7 ft. (2.2 m.) of the No. 6 coal, at a depth of from 11 to 41 ft. (3.4 to 12.5 m.). The outcrop of No. 6 coal westward from Duquoin has been quite thoroughly prospected, according to Messrs. Dowell and LaFont, but no available stripping area has thus far been discovered.

Near Millstadt, east of St. Louis, the Northern Coal Co. had a Marion shovel, type 250—75-ft. boom and 3½-yd. dipper. (Now abandoned.)

Indiana

M. Scollard, Mine Inspector, says that there were in Indiana, September, 1916, ten stripping plants, with capacities of 2000 to 20,000 tons

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[1] S. O. Andros: Coal Mining Practice in District VIII (Danville). Illinois Coal Mining Investigation, *Bulletin* 2 (1914), 47.

per month. These plants are described by Mr. Scollard and the operating companies, as follows:

Globe Mining Co., near Staunton, mining No. 3 coal 5 to 7 ft. (1.5 to 2.2 m.) thick with 14 to 40 ft. (4.3 to 12 m.) of overburden. Coal is loaded with a small shovel; plant has capacity of 1000 tons per day.

Warren Coal Co., near Hymera, is mining No. 7 seam, 5-ft. (1.5-m.) thick overburden with soil and soft shale. The floor is soft, output is 600 tons per day. One 5-yd. (4.5-m.) Marion shovel is used for overburden and one 2-yd. (1.8-m.) loading shovel for the coal.

Forschner Coal Co., near Linton, is mining No. 4 seam, 5 ft. (1.5-m.) thick with 20 to 30 ft. (6 to 9 m.) of overburden consisting of 18 ft. (5.5 m.) of clay, 12 ft. (3.6 m.) of soft shale and with fire-clay floor. Capacity of 800 tons per day; 60 acres (24 ha.) have been stripped with 50 acres available. A 250 Marion shovel is used, with 3½-yd. (3.2-m.) dipper, and a model 31 is used for loading coal.

Carbon Mining Co. at Carbon is mining the upper Brazil block coal 4 ft. (1.22 m.) thick, 14 to 25 ft. (4.3 to 7.6 m.) of overburden. In opening the pit, a Class 14 Bucyrus drag line was used and this is now used for hoisting the coal in boxes from the pit; also used for mining 7 to 12 ft. of underlying fire clay. The output of coal is 4000 tons per month. Ten acres have been stripped to date and 20 remain to be stripped. The pit is drained through bore-holes into mine workings 60 to 70 ft. below the strip pit.

J. L. Conover & Co., Carbon, is working block coal 4 to 4½ ft. (1.21 to 1.37 m.) thick with 9 ft. (2.8 m.) of surface clay and 9 ft. of shale overburden. There are 8 to 11 ft. of good fire clay under the coal. The relation between the coal stripping and the fire-clay industry is well described by Mr. Conover as follows:

"At present we are not doing any stripping but are loading the clay from under the coal which we have stripped. We originally started stripping with a 70-C Bucyrus steam shovel, which is a railroad contractor's shovel, and loaded the overburden in Western 4-yd. (3.6-m.) dump cars and hauled the dirt to a spoil dump with 18-ton Davenport dinkies. However, we found this method too expensive. We have contracted the stripping of our coal. The contractor is to strip the coal with a drag line, load the coal on 4-yd. cars in the pit and deliver these cars to the bottom of an incline where they are pulled up to a tipple with a cable and where they dump automatically. We have stripped about 1 acre and have about 10 acres to strip. There are about 20 ft. (6 m.) of overburden, half blue shale and half yellow clay. We grind 7 ft. of fire clay that is found under the coal and ship it to foundries to use in molding and laying brick. Our fire-clay business is by far the best part of our stripping operation."

The Sunlight Coal Co. of Booneville, Ind., is mining the No. 5 seam, 7 ft. (2.2 m.) thick with 20 to 25 ft. of overburden.

The Linton Coal Co. near Linton has one 5-yd. Marion shovel and one small loading shovel. Other companies in Indiana are the Inland

Coal Co. at Janesville, the Richards Coal Co. at Terre Haute and the Eppert Coal Co. at Coal Bluff.

The Linton Fourth Vein Coal Co. at Linton is stripping 12 to 14 ft. (3.6 to 4.3 m.) of shale and clay soil with a 225-B Bucyrus shovel having an 85-ft. (25.9-m.) boom, a 62-ft. (18.9-m.) dipper handle and a 7-yd. (6.4-m.) dipper. This shovel can move 8000 to 10,000 yd. of overburden in two 8-hr. shifts. The coal is loaded into cars with a 35-B Bucyrus shovel which loads from 1100 to 1300 tons in 8 hr.

Iowa

James N. Lees, Assistant State Geologist reports that he does not know of any stripping being done in Iowa.

Kansas

The following account of the southeast Kansas-Missouri area and operations has been furnished by Eugene McAuliffe:

"What is known as the Southeastern Kansas Coal Field is located in Crawford and Cherokee Counties, Kansas, and Barton County, Missouri. Fig. 6 shows the eastern crop line and the western edge of the coal, where the cover is about 260 ft. (79 m.) deep and where erosion has replaced the coal with sand stone. An arrow indicates the dip which is N. 69° 58′ W., as calculated by me from levels covering a large number of drill holes.

"Two seams of coal in this field are being stripped (Fig. 7); the lower or Weir City-Pittsburg, varies from 42 in. in thickness in the south end of the field to 33 in. in the north end." (According to H. A. Buehler, State Geologist of Missouri, these coals occur in the Cherokee formation of the lower coal measures.)

"My estimate of the strip coal in this territory and confined to this seam, as originally found in place and now remaining, is substantially as follows:

Original area in Kansas	10,320 acres	
Original area in Missouri	11,680 acres	
Total		22,000 acres
Removed to date in Kansas	5,920 acres	
Removed to date in Missouri	2,400 acres	
Total		8,320 acres
Remaining to strip		13,680 acres
Tonnage available for stripping Weir City-Pittsburg seam		68,500,080 tons

"The upper seam, sometimes known as the Lightning Creek seam, while generally present over the entire district, is found of sufficient thickness to warrant stripping only over a limited area. This seam is 65 ft. (19.8 m.) above the Weir City-Pittsburg seam, and where it is now being stripped ranges from 20 to 25 in. (0.5 to 0.6 m.) in thickness. My calculations indicate about 2640 (1068.37 ha.) acres of this seam available for stripping, totaling 8,680,000 tons. In calculating the tonnage of the

lower or Weir City-Pittsburg seam, I have deducted 10 per cent. for faults and horsebacks, which are very numerous in that seam; the Lightning Creek seam being reasonably continuous as to thickness.

"The cover above the coal in the Kansas field ranges from 8 to 40 ft. (2.5 to 12 m.) of soft slate, soft shale, and yellow clay. From 5 to 6 ft. (1.5 to 1.8 m.) of soft fissured

FIG. 6.

sandstone is sometimes found and frequently the shovels move this without blasting, in other cases explosives are required to loosen it. It is not uncommon to strip from 30 to 35 ft. (9 to 10.6 m.) above the coal.

"In the northeast corner of the map in the hatched area there are three shovels working; the coal seam going back under a flat hill. There are 45 shovels now working in the southeastern Kansas field, the majority being of a modern type. Some of these shovels have a 90-ft. (27.5-m.) boom and the dipper handles vary from 45 to 55 ft.

(13 to 16 m.) in length, the dipper in some cases being 6½ yd. (5.9 m.) capacity. One shovel is working at Browington, Henry County, Missouri, but the area of stripping coal is there very limited."

Kentucky

J. B. Hoeing, State Geologist, reports that he knows of no stripping being done in Kentucky at present. Some was done in Laurel County years ago, as is noted later under the development of the steam shovel.

Michigan

R. C. Allen, Director, Michigan Geological and Biological Survey, reports that no coal is being stripped in Michigan at present. A small amount of coal was stripped in Williamston, Ingham County, about 2

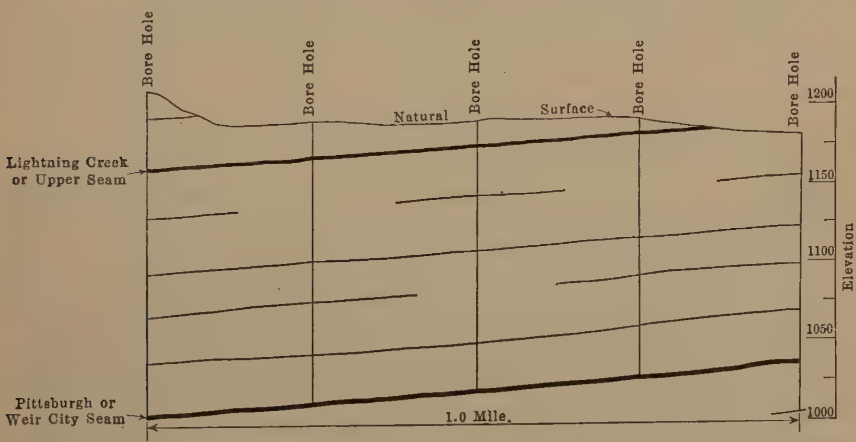

FIG. 7.—SEAMS STRIPPED IN KANSAS.

years ago, but on a very small scale. At Grand Ledge, Eaton County, a small amount of coal is obtained in connection with the winning of clay. Excepting for these operations at Williamston and Grand Ledge, the conditions in Michigan are not favorable for stripping, as the coal basin is generally deeply covered by drift and the coal is from 50 to 250 ft. (15 to 76 m.) in depth.

Ohio

J. A. Bownocker, State Geologist of Ohio, reports that stripping is being carried on in an extensive way in the No. 8 or Pittsburgh coal in eastern Ohio. There are not more than a half dozen plants in operation at the present time, since the country is so hilly that the areas where stripping is possible are not large. There is also a plant on the border of Stark and Tuscarawas Counties on which the lower Kittanning bed is

being stripped. Ohio has about a dozen coal beds that are being mined which are persistent across the southeastern part of the State. At many places they may be mined by stripping and the areas though large would form only a small part of the entire coal field. No figures are available giving the amount of coal obtained by stripping.

S. A. Taylor, of Pittsburgh, who is interested in Ohio stripping propositions, says that the location of the Ohio stripping is at present in Harrison and Jefferson Counties in eastern Ohio in the No. 8 field largely. There are five or six different companies operating in this district. The property of the Kehota company with which he is connected is located in Perry County in the Hocking district. He says:

"We have tested quite a number of properties in Hocking and Perry Counties, as well as in others, but have not found any available, excepting two in Perry County. It is possible that others will be discovered by drills now working."

It is rumored that a number of shovels have been purchased recently to operate in Ohio.

Pennsylvania

In regard to southwestern Pennsylvania, S. A. Taylor says:

"There is some coal in western Pennsylvania along the Ohio River, bordering West Virginia, which may be stripped. While the overburden is light, it is composed largely of rock and I fear that the cost of stripping may be prohibitive. An attempt to strip in this section will soon be started by John Bell and George Flynn of Pittsburgh. As to stripping the Upper Freeport coal, I do not think this is feasible, as this coal is capped by Mahoning sandstone which is usually a massive sandstone of considerable thickness and unless locations were found where the rock was eroded, leaving the thin black slate immediately over the coal, I do not think it would be a commercial stripping proposition."

Richard R. Hice, State Geologist, says:

"I think there is quite a little area of the Pittsburgh bed in Washington and Allegheny Counties that can be mined by stripping and, indeed, only mined in this manner. At present most of this is too distant from transportation. The possibility of stripping this coal is due to the fact that the general rise of the rocks to the north has brought the Pittsburgh coal to daylight. It is now found close to the top of the hills with but little cover. The area lies outside of the glaciated region. It is possible that the Pittsburgh bed can be stripped farther east and I know of no reason why there should not be places where considerable areas of the Allegheny coals might also be mined in this manner, but we have no exact information on this subject. I understand there is a prospect of mining some Upper Freeport coal by stripping."

Joseph Knapper, Mine Inspector, Philipsburg, Pennsylvania, reports that at Karthous an operation belonging to the Black Diamond Coal Co. owned by Rembrandt Peale was abandoned by some Italian miners who contracted to deliver coal from about 10 acres at a price which should compete with pick mining. They worked for about 6 weeks, removing the cover, which varies from a few inches to 6 ft. (1.8 m.) and used only

pick and shovel. At that time they had uncovered about 200 tons of coal, but most of it was smut which was not fit for use. They attempted to collect days' wages from the coal company, but the suit that was brought was soon dropped, and the attempt at stripping ceased.

Oklahoma

C. W. Shannon, Director, Oklahoma Geological Survey, says:

"Considerable stripping has been done near Coalgate and Lehigh in the southern part of the field, but the greater part of this has been done by teams and consequently no very large quantity has been removed. Some of the companies have been considering the steam-shovel proposition in the outcrops of these districts. However, this has not been undertaken. The coal of this vicinity is from 3 to 4 ft. (0.9 to 1.2 m.) thick and the overburden is of such character that it can readily be removed by the use of a steam shovel. Other districts in the State in which coal is being removed by stripping are near Collinsville, in Rogers County, and near Broken Arrow in Tulsa County, the mining, however, being chiefly across the county line in Wagoner County.

"Practically all of the coal in the State lies at a considerable angle and of course outcrops over long and sinuous courses."

Bulletin 22 of the Oklahoma Geological Survey, page 72, under the heading "Mining Coal by Strip Pits" says:

"In northeastern Oklahoma and southeastern Kansas there are several thin beds of coal covering large areas. Until very recently these coals had not received very favorable notice. These deposits consist of almost continuous beds and are from 1 ft. 6 in. to 2 ft. (0.46 to 0.6 m.) in thickness. The coal is of extra good quality for shallow coal. These beds are covered with an overburden of shale and clay from less than 10 to 30 ft. (3 to 9 m.) in thickness. Some of these deposits, as well as narrower strips along the outcrop of the principal beds in the main part of the field, have been worked in the past by stripping with horses and scrapers, but this method was not found practicable where the covering is more than 10 ft. (3 m.). During the past few years steam-shovel operations have begun in these areas and a marked development has taken place. Two shovels of the largest type are operating in the Oklahoma field, one near Lehigh and the other at Henryetta."

The Handbook of the Natural Resources of Oklahoma, page 13, says:

"In the northern end of the coal field are several beds of coal averaging from 1 ft. 6 in., to 3 ft. 6 in. (0.46 to 1.06 m.). These coals dip but slightly and are not far from the surface. Mining operations are being carried on in these coals at Bluejacket, Catale, Collinsville, Dawson, Tulsa, Broken Arrow, and Pryor. The coal mined is of a good quality, but is softer than the deeper coals. At Collinsville and Broken Arrow the coal is being stripped with a steam shovel and this method of securing coal has proved to be very satisfactory.

"There are many miles along the outcrops in various parts of the field where a little coal has been stripped in former years. The stripping was formerly done by teams. There are large areas throughout the coal region where steam-shovel stripping may be successfully carried on. The thickness of the surface which can be economically removed depends on the thickness and quality of the coal, but in most cases 25 to 50 ft. (7.6 to 15 m.) can be removed."

Tennessee

A. H. Purdue, State Geologist, reports that there is no coal mined by stripping in Tennessee.

West Virginia

I. C. White, State Geologist, reports that so far as he knows there are no stripping mines in that State, due to its very hilly character.

Development of the Steam Shovel for Strip Mining

In looking into the history of steam-shovel mining, it was found that Grant Holmes of Danville, Ill., for many years closely associated with the manufacture of steam shovels and with the development of the Missionfield Mining Co. near Danville, had gathered a large amount of data upon the development of the steam shovel in connection with coal mining. These data he has most kindly given permission to have incorporated into the present paper and, for practically all of the following historical section, credit is due Mr. Holmes.

In 1866, Messrs. Kirkland, Blankeney, and Graves opened a strip mine at Grape Creek south of Danville, Ill. In 1875, the late Michael Kelly opened a similar mine at Hungry Hollow, also near Danville, and a number of similar operations were begun at about the same time in this same district. The coal was usually exposed in long pits, a section along one edge of the field being first plowed, the surface removed with scrapers, and the overburden piled in a long mound overlooking the pit. This work was usually done in the summer and the coal was mined and hauled out during the winter by wagons. The following summer a block parallel to the one previously mined out was stripped, the waste being dumped into the abandoned cut. In 1876 and 1877, J. N. Hodges and A. J. Armil began stripping near Pittsburg, Kan., and depending upon the experience of Mr. Hodges in using steam shovels for building railroads in Ohio, they decided, after stripping coal with teams and scrapers for a while, that the work would be better done with a steam shovel. In 1877 they rented an Otis steam shovel and this is the first record of steam-shovel coal stripping. In regard to this pioneer machine, Mr. Hodges says:

> "We operated the shovel for about 1 year very successfully and I sold my interests to engage in other business, but the shovel was operated very successfully for about 3 years, when it was returned to the owners. We had land with from 8 to 12 ft., (2.4 to 3.6 m.) overburden. This with good management could be handled very well, but coal at this depth was limited and the boom of the shovel was too short to do deeper work and waste the overburden far enough away to uncover a pit of coal of sufficient width to handle economically." Mr. Hodges expressed the opinion that a larger shovel could be successfully used for such work, but the manufacturers said that such a shovel was not practicable.

The second steam-shovel experience was that of the Consolidated Coal Co. of St. Louis, which owned a stripping proposition in the river bottom, known as Missionfield, near Danville, Ill. The overburden did not exceed 35 ft. (10.6 m.) and was often only 10 to 15 ft. (3 to 4.5 m.) deep, while the coal bed was 6 ft. (1.8 m.) thick. Wright and Wallace of Lafayette, Ind., dredge contractors, accepted the contract to uncover a fixed amount of coal daily for the Consolidated Coal Co., under the supervision of J. L. Swanberg and Louis Stockett of the Consolidated Coal Co. They accepted the opinion of Hodges and Armil, that steam shovels had not been developed to a size suitable for stripping coal economically and, being dredgemen, Wright and Wallace purchased from the Marion Steam Shovel Co. of Marion, Ohio, a dredge minus the hull, which they erected on a wooden frame (Fig. 8), supported on wheels, thus forming a dry-land dredge built entirely of wood. The boom was 50 ft. (15 m.) long and a single-cylinder vertical steam engine furnished

Fig. 8.—Wright and Wallace shovel.

Fig. 9.—Dredge and stacker.

power to hoist the ¾-yd. (0.68-m.) dipper and to swing the boom. With this machine, 400 cu. yd. (305 cu. m.) of overburden per day was a great record. The only means of propelling the dredge was by block and tackle; hence the moving of the outfit was a slow process, particularly on curves, as the wheels were rigidly fastened to the frame. This dredge moved forward in an elliptical spiral path and, as the curves became short, jacks were used to skid the dredge around the sharp curves. The widest cuts possible with the dredge were 20 ft., and on account of the limited dumping range waste banks could not be placed far enough away to prevent the dirt from rolling down and covering up the face of the coal. Because of this gradual covering of the coal, it was necessary for the coal miners to follow immediately after the stripper and also to cut an entry in the coal before it could be loaded. The slowness of operation, the narrowness of the cut, and the annual flooding of the field from the neighboring stream induced the contractors to place a second shovel

in an effort to keep their agreement with the Consolidated Coal Co This second dredge or shovel was similar to the original one, but larger, having a 1¼-yd. (0.38-m.) dipper. Soon after, a third similar machine was purchased with a 1½ yd. (0.46-m.) dipper, 65-ft. (19.8-m.) boom, and two vertical steam engines. In 1888, an unforeseen complication arose in the nature of a strike of the coal miners, thus shutting down the coal-producing part of the operation. As this contingency had not been provided for in the contract, the stripping contractors continued to operate the dredges or shovels during the strike, and a point was soon reached where the shovels had made a complete round of the workings and were starting on a second cut, depositing the overburden upon the coal which had been stripped on the preceding round. To overcome this, the coal company bought the dredges and the contract, and when the strike was finally settled the stripping work was resumed and the three machines operated for 2 years by the Consolidated Coal Co. Apparently no money was made and owing to the heavy expense and the slow rate of operation and to the fact that the machines were about worn out, not having been built to stand such heavy work, and after many fruitless attempts to repair them, in 1890 they were abandoned.

The next development in steam-shovel mining was an adaptation of a scraper line used in the production of clay for ballast burning in Kansas and Missouri. This gumbo or sticky clay was burned in trenches in which alternating layers of coal and clay were placed. In the digging of the trenches the Butler Brothers (Henry G. and William) who were the ballast burners for the Rock Island Railroad and who previously came from Kenosha, Wis., used drag lines and a cable-manipulated scraper bucket.

The Consolidated Coal Co., after investigating this drag-line operation, made a contract with Butler Brothers to finish the stripping at Mission field. They began work in 1890 with three drag lines with bucket capacities of ¾, ⅞ and 1 yd. respectively. The drag or scraper buckets were steel boxes with one open end and with teeth on the bottom edge. A cable passing through a sheave wheel on top of the drag served as an adjustable track on which the buckets moved forward and backward and by means of which it was raised and lowered. By means of ropes fastened to the two ends, the buckets were moved backward and forward and by means of another cable the latch was tripped so that the bucket swung downward to dump the load. These various cables were strung along a horizontal boom 80 ft. long, which boom was suspended from a vertical frame or gantry by ropes. Two vertical boilers supplied power for a two-cylinder engine geared to three drums which were controlled by clutches. The machine was self-propelled in either direction by means of gearing operated from the same engine.

These drag lines worked on top of the bank and near the edge the

horizontal boom extended over the cut. The machines worked with accuracy and speed and three dumps per minute was not an unusual record. As the boom could not swing, the whole machine was moved to keep the scraper supplied with digging material, and this required much track-laying of an expensive nature to prevent the drag line from sinking into the soft overburden. With these drag lines, a rectangular cut was made about 20 ft. (6 m.) wide and a ¼ mile (0.4 km.) long, the boom being of sufficient length to permit the waste bank to be built far enough from the coal face to prevent burying the face by sliding spoil. When the end of the cut was reached, the machine was moved back about 20 ft. from the edge of the bank and then was worked back in the reverse direction from the previous cut, the spoil being deposited in the cut from which the coal had meanwhile been quarried. These first operated on what is known as the upper bottom of Missionfield, where Wright and Wallace had previously operated. The stripping was shallow and the overburden contained no hard material, so that the operation was carried on with ease and speed and an output of 1000 tons of coal a day was obtained. The available easy stripping ground in this section did not last very long, as much of the coal had been mined by Wright and Wallace in their previous work, and because of increased overburden, unexpected appearance of hard blue shale above the coal, and bad floods, the Butler Brothers[2] moved two of the excavators to what is known as lower Missionfield, the third machine being abandoned. In this new field, the overburden was light and rapid work was made on a 40-acre tract of coal. The machines operated in tandem, one stripping, and the other, loading the coal directly into railrod cars which ran into the pit, making one of the first examples of machine coal mining. In the center of the field, however, shale and sandstone were encountered and the depth of overburden increased. The work progressed more slowly and it became necessary to drill and blast the hard material, thus increasing the cost of production. Butler Brothers' contract with the coal company was for a certain price per ton delivered on coal cars and, as the result of the difficulties in digging and the trouble with the miners who struck for an 8-hr. day and increased wages, the drag lines were abandoned.

Butler Brothers still believed in this type of excavator and thought that a larger one would be successful. Financed by the Consolidated Coal Co., they began the erection of a large drag line which should handle deep stripping, shale, soapstone, and coal and in 1900 this large machine was completed at a cost of $30,000. It was supported on three 10-in. (0.25-m.) axles, each 22 ft. (6.7 m.) long, the center axle being geared to the engine and the end axles connected to the center one by driving rods. The horizontal boom was 135 ft. (41 m.) long and was supported by a vertical gantry 60 ft. (18 m.) high. To balance this, the rear end of the

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[2] H. H. Stoek: Dredging for Coal. *Mines and Minerals* (August, 1901), 5.

machine was loaded with 25 tons of iron, but later, as the result of a fire, the horizontal boom was shortened by 20 ft. (6 m.) and then the difficulty was to keep the machine from tipping backward. Three buckets were used, each holding about 2 cu. yd. (1.5 cu. m.), one intended for dirt, one for rock and the third for coal. The difficulty in operating this machine was its immense weight which caused both rails and ties to be buried in the ground, especially those near the edge of the bank, with the resultant constant fear that the whole machine would slide into the pit. The rock bucket, which was later fitted with a knife to cut into the hard material, was not successful, as it could not be held into the cut, and drilling and blasting, at a heavy expense, was necessary. The loading of the coal with the machine was entirely successful and 25 gondola cars could be filled a day. After 2 years of unsuccessful effort to keep to their part of the contract, Butler Brothers gave up the undertaking, but 16 of their employees leased the equipment and, as a coöperative venture, under the name of the Salt Fork Co., operated with fair success in a section where the coal was not covered with hard material. In 1904 this enterprise failed, due partly to there being too many coöperators.

In 1903, Mr. Donovan and associates with a Schenable drag line operated in the Middle Fork bottom of Missionfield in a neighboring valley to the one the Salt Fork Co. was operating. This drag line, though much smaller than those of the Butler Brothers, revolved in a complete circle, and for 9 months this machine operated, but, due to overflow from the adjoining river, the waste banks slid down so badly that no coal could be taken out. The outfit and property were bought by Edward Gray who organized the Gray Coal Co. but after 5 months of unsuccessful operation he also gave up.

Mr. Gray was next made manager of the Salt Fork Coöperative Co., but in less than a year's time the coöperators again took over the proposition into their own hands and for 2 years attempted to carry on the work. The occurrence of the hard soapstone, excessive repair bills, and floods from the nearby river, resulted in the failure of this enterprise for the third time. Thus, after a period of 9 years and as a result of the efforts listed, only 7 or 8 acres (2.8 to 3.2 ha.) had been stripped, at a cost of approximately $100,000.

In 1907, under conditions similar to those in the Missionfield, a stripping proposition was started near Lilly, Ky., in the Robinson Creek bottom where Jellicoe coal was 28 in. (0.71 m.) thick with only 6 to 8 ft. (1.8 to 2.5 m.) of overburden. Two shovels were used, one for stripping, the other for mining, built by the Vulcan Steam Shovel Co. of Toledo, Ohio. The stripping shovel had a 2½-yd. (2.3 m.) dipper and a 28-ft. (8.5-m.) boom mounted on railroad trucks. The mining shovel known as the Vulcan "Little Giant" had a 1¼-yd. (1.2-m.) dipper, a 22-ft. (6.7-m.) boom and was mounted on traction wheels. After 2 years'

trial the work was discontinued, the principal trouble being that water caused the waste banks to slide into the pit.

The Standard Block Coal Co. tried to operate a Monighan drag line in this district, but unsuccessfully, and after 1 years' work the attempt was given up.

Thus, in both Missionfield and Robinson Creek bottom, steam shovels were first used and then drag lines. A combination of these two types was tried by the Danville Brick Co. of Danville, Ill. in a tract adjacent to its brick yards, which contained about 4 acres of coal under 25 ft. (7.6 m.) of cover. A small Vulcan steam shovel worked on top of the coal and dumped the spoil at one side. Moving parallel with the shovel, a small drag line mounted on top of the nearest waste bank picked up the spoil from the shovel and deposited it in front of the drag line, thus laying the foundation for its own tracks and also building the waste bank. As this coal was intended for use in the brick plant of the operating company, the question of reasonable profit did not enter in so long as the coal could be produced below the current market prices. This piece of coal was uncovered during the fall and winter of 1912–13.

About 6 miles southeast of the Missionfield is a stripping area called the Carbon Hill district, containing about 70 acres (28 ha.) underlain by No. 7 coal 6 ft. (1.8 m.) thick, with an overburden of about 40 ft. (12 m.) of sand, gravel, loam and a heavy bed of shale. This tract was purchased by the Consumers Coal Co. of Danville in 1904, who contracted with George W. Prutsman, an excavation contractor, to strip the area. The machine used was built by the Bellefontaine Foundry & Machine Co. of Bellefontaine, Ohio, according to the joint ideas of Mr. Prutsman and George E. Turner, who were familiar with the failures of the drag line. They adopted the steam shovel, but instead of using a long-boom shovel, as Wright and Wallace had done, they incorporated a belt conveyor to handle the spoil dug with a short-boom shovel. This shovel could dig rapidly and deposit the waste in a hopper (Fig. 9) which fed the conveyor belt (Fig. 10), operating at right angles to the direction of digging. The shovel was operated by four two-cylinder engines, one for hoisting the 2-yd. (1.8-m.) dipper, a second for crowding it, a third for swinging a 35-ft. (10.6-m.) boom, and the fourth for operating the conveyor, which was 105 ft. (32 m.) long. The machine was built of wood and was mounted on four four-wheel trucks, thus giving a four-point suspension, no attempt being made to keep the frame level. It was propelled by block and tackle. The machine could only dig forward, hence the circular plan of operation was used. The work was begun in 1904, the surface being stripped in the summer and coal loaded out in winter. After several attempts to get the field in shape for circular operation, the plan was abandoned on account of the irregular shape of the coal area and the great depth of overburden in certain places. To overcome

this the work was opened in parallel cuts, which necessitated leaving the coal in the cut after one cut of the spoil had been completed, until the excavator could be moved back to the starting point for a second cut. This moving back usually took 2 weeks and the parallel cuts were not straight but had many inside turns in them; that is, turns in which the dumping arc is smaller than the digging arc. The end of the conveyor boom was thus practically a center about which the machine traveled during these turns and consequently the waste bank would build up so as to clog the conveyor. The conveyor was lengthened to 147 ft. (44.8 m.) but this did not obviate the difficulty; the bills for repairs were heavy, especially for the conveyor belts, and after 2 years, Mr. Prutsman gave up the contract. Backed by other men who formed the Coal Production Co., he took a new contract which in a few months also stopped for lack of funds. The Consumers Coal Co. then decided to

Fig. 10.—Stacker and spoil bank.

Fig. 11.—Drag line loading coal.

operate the stripping itself and the shovel was completely rebuilt, steel being substituted for wood. However, in spite of a better shovel and careful management, in 1913 this company quit business without having made a profit. This type of machine was therefore abandoned as a theoretical success but as a practical failure.

The combination of shovel for digging and conveyor for depositing of the spoil seems to be a good one. Morton E. Pugh, who spent a fortune and lifetime with conveying-belt strippers, took out his first patents on a machine that moved and dug sideways, the conveyor dumping behind. His later patents show that he separated the excavator and the conveyor into two independent machines, the steam shovel being substituted for his excavator and a crusher being added to the conveyor for sizing the waste before dropping it on the belt, the two machines being operated on parallel tracks. Mr. Pugh also planned to use a revolving steam shovel, so that the conveyor could be on the same track behind the shovel, but several attempts at such operation made in Iowa were unsuccessful.

The Carbon Hill stripping area is now being operated by the Carbon Hill Coal Co., with a revolving 5-yd. (4.5-m.) dipper shovel.

The Revolving Stripping Shovel

In 1909, the Missionfield Coal Co., of which W. G. Hartshorn was president, leased the stripping equipment described above as belonging to the Consolidated Coal Co., and after 2 months' trial operation bought the property with the equipment described. Easy stripping, the absence of floods and labor trouble, permitted a good profit to be made with the drag line for about 16 months, but then repairs became excessive and the equipment was abandoned. Those connected with the Missionfield Coal Co., although recognizing the faults in the Prutsman machine, believed that the ultimate stripping machine must combine the dumping range and the digging power over the conveyor steam shovel with an ability to dig and dump in any direction; that is, it should be a revolving shovel. They believed that the appliances built along these lines had failed through being too small to stand up to the work, and that the new type of stripping machine must be of greatly increased size, but although several of the steam-shovel manufacturers were already making small revolving shovels and locomotive cranes, none seemed willing to undertake the construction of a large revolving shovel.

In 1910, the Vulcan company built two revolving shovels for stripping purposes with 1½ yd. (1.37-m.) dippers and with booms about 50 ft. (15 m.) long. Messrs. Patrick Dirky and Joseph Stevenson each purchased one of these machines for use near Pittsburg, Kan. The shovels were badly proportioned in the swinging parts and constant breakdowns resulted. While these original shovels have been operated by a number of different men since that time, and are still being operated, they have never been very successful.

The Brown company of Cleveland, Ohio, next modified a locomotive crane so as to form a revolving steam shovel with a 2-yd. (1.8-m.) dipper and a 50-ft. boom, and in 1910 this machine was put to work in the Mission field. In 6 months the remains were sent back to the factory, as the machine was too lightly constructed, but it was evident that a start had been made in the right direction and after 2 years of argument, the Marion Steam Shovel Co. began the construction of a revolving shovel with the dimensions and according to the ideas of Messrs. Grant Holmes and W. G. Hartshorn of Danville, Ill. The machine had a 3½-yd. (3.2-m.) dipper, 40-ft. (12-m.) handle, 65-ft. (19.8-m.) boom, and weighed 150 tons, making the largest shovel in the world up to that time. A unique feature was the hydraulic compensating truck which was patented by Mr. Holmes and which is now used on all shovels of the Marion type, by means of which the shovel frame is kept level when the apparatus is

moved over irregular tracks. In 1911, this big shovel, known as Model 250, began work in the Missionfield, where it continued until the field was stripped completely, toward the latter part of the fall of 1915, thus completing the work of stripping the field where the early experiments of the Butlers and Brownings had been carried on. After 4 years of successful and profitable operation, the shovel was in good condition and repairs had been very slight. From the Missionfield it was moved to an adjoining stripping basin owned by the same interests, where it is still in operation. The operation of this shovel was watched with interest and, as soon as it was a demonstrated success, others were built. The large-size revolving shovel has now become the feature in coal-stripping operations.

Drag lines are used in a few places and their general construction is similar to that of the steam shovel in so far as the operating devices are concerned.

Steam-shovel Construction

A steam shovel should be able to work under severe conditions, as nearly as possible continuously, and with a minimum of breakdowns, as any shovel delay usually means a delay along the entire line of operation. The principal points to be considered in comparing shovels are:

Pounds of pull at the dipper.	Ease of movement.
Height of the lift.	Repairs.
Distance at which material can be deposited.	Cost of operation.
Capacity of bucket.	Cost of repairs.
Weight of shovel.	Investment cost.

The largest shovels used in stripping for bituminous coal are either of the Marion model 271 or 300, or the Bucyrus 150-B, 175-B, 225-B. Fig. 12 and 13 show photographs of the latest of each of the largest types. These are described in great detail in the excellent catalogs of the two companies noted and only a few distinguishing points need here be given. They are designed to strip overburden from approximately horizontal beds of coal and other deposits where a great reach is needed, so that the spoil need not be hauled away in cars, but can be deposited either upon the original ground level or in the excavation from which the coal has been removed. As the entire superstructure revolves the shovel can work back and forth along the face by simply shifting the track and without turning the shovel. The booms vary from 60 to 90 ft. (18 to 27 m.) in length[3] and the dippers from 2½ to 8 yd. (2.29 to 7.3

[3] 8-yd. dipper Marion has an 80-ft. boom.
6-yd. dipper Marion has a 90-ft. boom.
150-B Bucyrus has a 60-ft. boom.
225-B Bucyrus has an 85-ft. boom.

m.). For a given model, the larger the dipper the shorter the boom, to prevent overturning. In order to keep the platform horizontal when the shovel is moved, the Marion shovel is supported upon a compensating

FIG. 12.—MARION SHOVEL, MODEL 300, PITTSBURG, KANS.

FIG. 13—BUCYRUS 225-B SHOVEL, CARNEY CHEROKEE COAL CO., KANSAS.

hydraulic device patented by Mr. Holmes of Danville, and with the Bucyrus shovel the same end is accomplished by a three-point support with screw jacks. The Bucyrus shovels have a split dipper handle which

straddles the boom (Fig. 13), but the Marion has the dipper handle solid and working between the two parts of the boom. The later types of shovel have a very heavy built-up steel boom. The shovels are operated either by steam or electricity and the makers rate the capacity of the larger sizes as from 1500 to 5000 cu. yd. (1147 to 3823 cu. m.) per 9 hr. The larger shovels remove an overburden up to 45 to 50 ft. (13.7 to 15 m.) deep and make a cut from 40 to 110 ft. (12 to 33.5 m.) wide, depending on the depth of overburden, length of boom, and length of dipper handle. The width of cut varies inversely with the depth in order to provide suitable space for the waste. The catalogs of the several shovel makers contain tables of relative depths and widths of cut of shovels of different dimensions.

Shovels should be able to load cars standing on a track not less than 6 ft. above the shovel track and to do this the distance from the shovel track rail to the lowest point of the dipper door when open should be at least 14 ft. or more for advantageous working.

After material has been loosened by the steam shovel, it occupies from

Table 1

Company Manufacturing Shovel	Type	Boom	Handle	Dipper	Where Operated and Number							
					Ala.	Ill.	Ind.	Kan.	Mo.	Ohio	Okla.	Total
Bucyrus....	150-B	68	38	2½	1	...	1	5	1	...	....	8
Bucyrus....	175-B	75	48	3½	...	...	...	8	1	1	....	10
Bucyrus....	175-B	85	58	3½	...	...	1	...	...	...	....	1
Bucyrus....	225-B	80	58	6	1	...	2	3	1	5	....	12
		75	58	6	...	...	...	2	...	...	1	3
		80	58	7	...	...	...	1	...	...	....	1
		80	54	7	...	...	...	5	...	3	....	8
Marion.....	211	45	....	2½	...	...	1	...	...	...	....	1
Marion.....	250	65	....	3½	...	1	2	7	1	...	....	11
Marion.....	250	75	....	3½	...	2	1	1	2	...	....	6
Marion.....	251	75	....	3½	...	...	1	2	2	...	....	5
Marion.....	251	75	....	4	...	...	...	...	1	...	....	1
Marion.....	252	75	....	3½	...	...	...	1	...	...	....	1
Marion.....	270	90	....	5	...	1	...	2	...	...	....	3
Marion.....	270	80	....	8	...	1	1	...	...	1	....	3
Marion.....	271	90	....	5	...	...	3	1	1	1	....	6
Marion.....	271*	90	....	6	...	...	...	...	...	1	....	1
Marion.....	271	80	....	8	...	...	...	1	...	...	....	1
Marion.....	300*	80	....	8	...	...	2	1	2	2	....	7
Marion.....	300*	90	....	6	...	...	...	...	...	3	....	3
					2	5	15	40	12	17	1	92

* Electrically operated.

25 to 50 per cent. more space than when solid and the waste bank will take up considerably more space than the original solid bank. To provide ample space for mining the coal, the bottom of the waste bank should be kept at least 10 ft. away from the coal face. If part of the overburden dug by the shovel is harder than the main portion and therefore does not run as readily as the main spoil bank, this harder material may be deposited nearest the coal face and thus serve as a retaining wall to hold back the more easily running spoil. Table 1, based mainly on lists furnished by the Bucyrus and Marion companies, shows the distribution of the steam shovels in the United States, July 1, 1917.

Steam-shovel Operation

The statistics of steam-shovel operations are somewhat meager. A summary of those collected by the United States Geological Survey for 1915 is shown in Table 2. Data regarding the depth of cover and yardage handled were published in the report of this series for 1914.

TABLE 2.—*Coal Recovered from Steam-shovel Strip Pits in* 1915

State	Number of Shovels	Quantity of Coal Mined (Net Tons)	Average Tonnage per Man		Average Value per Ton
			Per Day	Per Year	
Alabama	1	(*a*)	....	......	.....
Illinois	10	455,195	8.7	1,979	$1.00
Indiana	22	638,220	8.3	1,474	0.90
Kansas	23	780,787	5.3	1,169	1.58
Missouri	20	655,670	4.4	870	1.66
Ohio	7	273,263	7.0	1,763	0.91
Oklahoma	4	28,484	4.3	274	2.08
Total bituminous (except Alabama)	87	2,831,619	6.0	1,208	$1.29
Pennsylvania, anthracite	57	1,121,603	....	....	.....
Grand total (except Alabama)	144	3,953,222	....	....	.....

(*a*) Only one steam shovel was in use in Alabama, and the extent of its operations cannot be revealed.

Steam shovels operate either in a circular path beginning at the outside of the area and gradually working inward spirally until an area or island is left in the center about which the shovel cannot be economically moved (Fig. 14).[4]

This method is used mainly by the old-type fixed shovels to avoid turning the shovel, but with the modern revolving shovel this necessity

[4] The illustrations for the methods of operating steam shovels are furnished by Grant Holmes of Danville, Ill.

is obviated and such shovels make first a "thorough cut" in a straight line across the property (Fig. 15), dumping the spoil on the surface, and

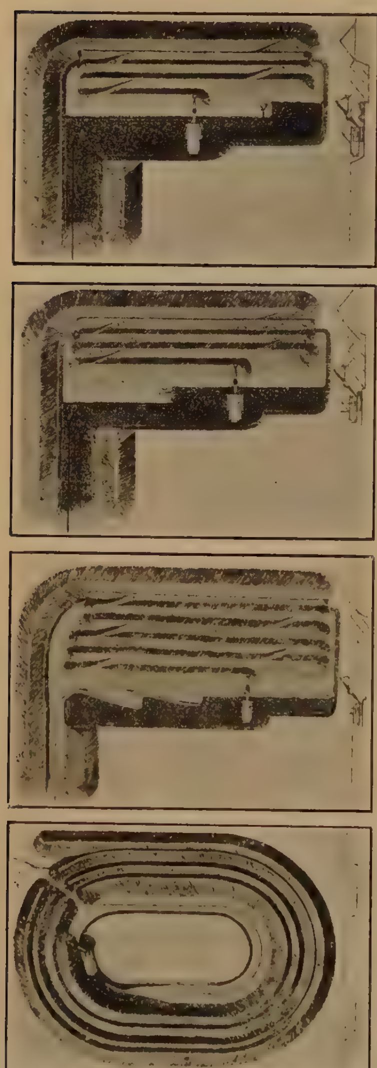

Fig. 14.—Spiral. Fig. 15.—Straight. Fig. 16.—Modified. Fig. 17.—Modified.

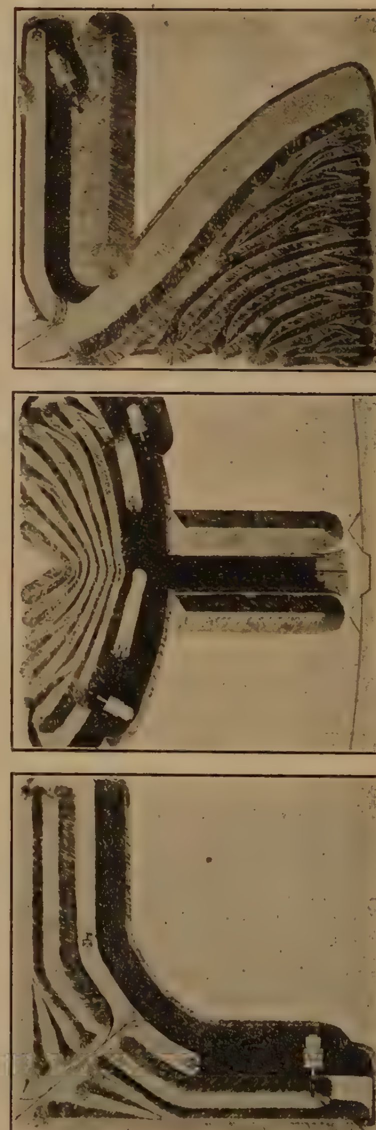

Fig. 18.—Corner opening. Fig. 19.—Center opening. Fig. 20.—Corner opening.

Fig. 14 to 20.—Various methods of steam-shovel operation.

after reaching the boundary return on a parallel cut, dumping the spoil in the excavation from which the coal has been meanwhile removed, either by hand or by a smaller steam shovel.

The details of the parallel cut method can be modified as shown by Fig. 15 to 17, and as necessitated by the shape of the property, drainage, depth of overburden, etc.

Fig. 16 and 17 show two such modifications of the ordinary parallel system. The first cut is made close to one of the property lines and extends across the property. At the far end cuts are made at right angles extending to the other property line. When the shovel reaches the far property line it reverses its direction and returns parallel to the previous cut. The cross ridges shown in Fig. 15 to 17 are made when the shovel turns into the bank to begin the return cut. After the shovel has dug in the proper distance in beginning the cut, it digs back to uncover the triangular piece left in opening up the cut. In Fig. 16 the small loading shovel is shown following the digging shovel and there is a curve in the loading track laid on top of the coal. This curve must be continually moved forward as the loading shovel advances and is, therefore, a constant source of expense and trouble. In Fig. 17 the digging shovel follows the loading shovel; the loading track is continuous along the coal face and therefore does not require changing.

Fig. 18 shows a stripping field opened from the corner and having a diagonal haulway. The shovel starts from the upper left-hand corner and makes the first cut along one property line, depositing the spoil away from the boundary line. It then returns to the starting corner and on its next cut dumps the spoil from this cutting and about one-half of the waste bank from the thorough-cut against line so as to prevent caving along the line. Then returning again to the original starting corner, a second thorough-cut is made at right angles to the first, a gap being left between the ends of the spoil bank at the starting corner to provide for future haulage. In order to avoid right-angle turns with the shovel, wedge-shaped cuts are taken off from the starting corner as indicated and soon a continuous face is formed, giving easy turns for the shovel. With such an arrangement two small shovels can be used, both loading coal or, if there is an underlying fire clay, as sometimes occurs, the second shovel can load fire clay. If the underlying material is of value, a larger output of coal can be obtained by using two stripping shovels and two loading shovels, but in this case it would not be necessary to provide for the small wedge-shaped cuts, as neither of the shovels would have to then make a right-angle turn.

Fig. 19 shows a somewhat similar method of opening up by running the thorough-cut through the center of the property and across its full width. At the far end of this thorough-cut the shovel cuts in opposite directions, but by first making a number of short cuts a continuous face is produced as shown.

Fig. 20 shows a method intended to accomplish the same result as that illustrated by Fig. 18. From one corner a thorough-cut is made

along one property line, the overburden being thrown inward and away from the property line. When the opposite boundary is reached the shovel makes alternate short and long cuts in order that it may be kept working along a line 45° to the boundary, dumping being toward the boundary line. Another thorough-cut is made along the boundary at right angles to the first and a similar procedure followed, thus giving a fan-shaped area and insuring that spoil banks are left against each property line to protect them from caving.

Mining and Loading the Coal

The thin layer of dirt left on the coal by the shovel is removed with hand shovels and brooms and vertical holes are then drilled in the coal

Fig. 21—Marion model 36 loading shovel.

by hand augers or power drills at intervals of about 6 ft. and a few feet back from the face. The Jackhamer drill with a dull drill bit is used in the Danville field. The Clipper Blast Hole drill is also used for this purpose. Wet holes are dried out by firing one-fourth of a stick of dynamite in the hole. The coal is shot by using about a quart of black powder in each hole, and a keg of powder gives about 110 tons of loose coal in the Kansas field.

At small operations the coal is still shoveled into pit cars by hand, but small revolving steam shovels are rapidly replacing hand labor for loading wherever conditions permit their use.

The loading shovels used are shown in Fig. 21 and 22. Fig. 21 shows Marion model 36 mounted on caterpillar traction trucks, and equipped with a special 2-yd. dipper.

The Bucyrus company has two types of loading shovels. Until recently the one mostly used was type 35-B revolving shovel, equipped with a slightly longer boom and handle than the standard 35-B shovel so that it could load battleship cars. The Bucyrus company has lately put out a new machine known as 27-D coal loader, designed especially for the work in the Kansas coal fields where the coal beds are seldom 3 ft. in thickness and are badly cut up by intrusions of fire clay or horse-backs. To operate in these conditions, it is necessary to have a machine with a horizontal thrust and a vertical lift, so that by means of the hori-

FIG. 22.—BUCYRUS MODEL 36-B LOADING SHOVEL.

zontal thrust the coal is separated near the cleavage lines and not broken up, and by the vertical lift the coal can be excavated close up to the horse-back without mixing the clay with the coal. The radial action of the ordinary loading shovel is said to break the coal badly and make it difficult to separate the coal from the refuse material when the horse-back is encountered. The shovel illustrated in Fig. 23 should handle 600 tons per shift of 9 hr. under ordinary conditions, and under conditions pre-vailing in Kansas where 10 to 15 per cent. of the area consists of horse-backs, it should handle about 400 tons, although 700 tons per day have been handled under Kansas conditions.

These small shovels run on a track, or have a caterpillar traction mounting that enables them to run on the bottom of the pit without track. They load the coal into gondolas (Fig. 22) or into pit cars (Fig. 23), which are hauled away by mules or small locomotives to the tipple or

to the foot of an incline leading to the tipple up which the cars are hauled by a rope.

In some of the Kansas coal there are too many "horse-backs" to make the use of a small steam shovel practicable, but by hand shoveling the coal is loaded into cars or boxes and these are lifted by a crane to the track on the surface. The crane is set on the surface close to the edge of the pit and propels itself along the surface so as to keep up with the miners, close to the steam shovel. In some cases cars are lifted out of the pit by a derrick which is set up on the coal. In order to avoid wear and tear on the cars, the crane handles loose car bodies, lifting the loaded bodies to their running gears on the surface. A small steam locomotive then

Fig. 23.—Bucyrus 27-D loading shovel.

handles the cars to the tipple where a derrick raises and dumps the bodies at the tipple.

Strip-pit Drainage

Strip pits often have much water to contend with. Sometimes there is an outlet to lower ground so that the pit becomes self-draining, but, in general, pumping is resorted to, electric centrifugal pumps being commonly used. In case of severe storms, the pits may become flooded to such an extent as to close the operation until the water is pumped out.

Strippings located in bottoms near to rivers are particularly subject to overflow at time of high water and frequently must be protected by dikes thrown up along the entire river front. When water comes into the bed by seepage and from springs, a large main tile is laid in the first cut parallel to the direction of cutting and below the coal. Openings in this carry water away from the first cut, and laterals attached to these

openings pass under the spoil bank deposited in the cut out to the face of the coal.

Strip mines located above underground mines may be drained through bore-holes into the under mine.

Costs

C. E. Lesher in the U. S. Geological Survey Report for 1915 says:

"That coal can be mined with steam shovels more cheaply than by underground methods is indicated by the fact that the average daily output per employee in the stripped pit operations is twice as great as the average for the respective States. In other words the labor cost is about one-half as great."

Reference has already been made in the introduction to the fact that too often such items as repairs, amortization, delays and expense incident to floods are overlooked in calculating the cost per ton of coal produced by strip mining.

In *The Colliery Engineer* for March, 1913, Barry Scobee gives the cost of removing an average of 17 ft. of soil in the Kansas field as 5 to 6 c. per cubic yard of dirt removed, the overburden consisting of 6 ft. of soil, 6 ft. of shale or soapstone and thin blue shale to the coal.

The steam-shovel crew consists of:

1 craneman,
1 oiler,
1 fireman,
1 coal shoveler.

In the pit there are usually three men.

The crew of an electrically operated shovel consists of:

1 engineer,
1 oiler,

and the pit crew of two or three men, thus decreasing the labor expense for a shovel so operated. There is also no expense for coal or water which must be compensated for by the cost of power.

The Marion company says "Strip costs as given to us are not very reliable, but we should say that 40 to 80 c. per ton of coal mined would be a fair estimate varying with conditions."

The Bucyrus company gives the following estimates of costs for its different types of shovels:

175-B Steam Shovel.

1 engineer	per month,	$155.00	Coal uncovered per day at 2000 cu. yd. stripping on coal 3′–0″ thick, being a ton of coal under each square yard of surface.
1 craneman	per month,	100.00	
1 fireman	per month,	75.00	
4 pitmen	per month,	260.00	
1 watchman	per month,	75.00	
140 tons coal	per month,	140.00	15′ deep × 85′–0″ wide 400 tons
Water	per month,	50.00	18′ deep × 80′–0″ wide 334 tons
Oil, waste, pack'g.	per month,	30.00	21′ deep × 80′–0″ wide 287 tons
Repairs and upkeep	per month,	200.00	24′ deep × 80′–0″ wide 250 tons
Share of supt.	per month,	75.00	27′ deep × 75′–0″ wide 222 tons
Incidental	per month,	150.00	30′ deep × 70′–0″ wide 199 tons
Interest on $28,000	per month,	140.00	33′ deep × 60′–0″ wide 182 tons
		$1,450.00	35′ deep × 50′–0″ wide 171 tons

At the rate of stripping 40,000 cu. yd. per month at an expense of $1450 it makes the rate 3.6 c. per cubic yard.

We submit herewith an estimate on the expense per ton of stripping, taking out the coal, hauling it to the tipple, screening it and delivering it on railroad cars, our 175-B shovel to handle 24 ft. in depth of overburden at the rate of 2000 cu. yd. per day of 9 hr., uncovering 250 tons of coal per day, or 5000 tons per month of coal 3 ft. thick, as follows:

175-B Shovel				Expense per Month
Steam shovel	5,000 tons,	$0.29	per ton	$1,450.00
Blasting	5,000 tons,	0.05	per ton	250.00
Loading	5,000 tons,	0.14	per ton	700.00
Hauling	5,000 tons,	0.07	per ton	350.00
Tipple and track	5,000 tons,	0.07	per ton	350.00
Superintendent	5,000 tons,	0.03	per ton	150.00
Repairs	5,000 tons,	0.06	per ton	300.00
Incidental	5,000 tons,	0.03	per ton	150.00
Interest on $40,000	5,000 tons,	0.04	per ton	200.00
Total		0.78	per ton	$3,900.00

The expense of operating the 225-B shovel per month in Kansas and the amount of coal it will uncover in different depths of stripping at the rate of 4000 cu. yd. of stripping per day or 80,000 cu. yd. per month are here given as follows:

225-B Steam Shovel

1 engineer	per month,	$155.00	Coal uncovered per day at 4000 cu. yd. stripping on coal 3′–0″ thick being a ton of coal under each square yard of surface.
1 craneman	per month,	100.00	
1 fireman	per month,	78.00	
4 pitmen	per month,	270.00	
1 oiler	per month,	68.00	
1 watchman	per month,	75.00	15′–0″ deep × 115′–0″ wide 798 tons
150 tons coal	per month,	150.00	18′–0″ deep × 110′–0″ wide 669 tons
Water	per month,	80.00	21′–0″ deep × 106′–0″ wide 574 tons
Oil, waste, pack'g	per month,	50.00	24′–0″ deep × 100′–0″ wide 499 tons
Repair and upkeep	per month,	290.00	27′–0″ deep × 95′–0″ wide 444 tons
Share of supt.	per month,	100.00	30′–0″ deep × 90′–0″ wide 400 tons
Incidental	per month,	189.00	33′–0″ deep × 84′–0″ wide 364 tons
Interest on $39,000	per month,	195.00	36′–0″ deep × 75′–0″ wide 333 tons
		$1,800.00	39′–0″ deep × 65′–0″ wide 310 tons

At the rate of stripping 80,000 cu. yd. per month at an expense of $1800 the cost would be 2¼ c. per cubic yard.

We submit herewith an estimate on the expense per ton of stripping, loading the coal by hand into 2-ton tram cars, hauling it to the tipple, screening it and delivering it on railroad cars, our 225-B shovel to handle 24 ft. in depth of overburden at the rate of 4000 cu. yd. per day of 9 hr. uncovering say 500 tons of coal per day 3 ft. thick or 10,000 tons per month as follows:

225-B Steam Shovel				Expense per Month
Steam shovel	10,000 tons,	$0.18	per ton	$1,800.00
Blasting	10,000 tons,	0.05	per ton	500.00
Loading	10,000 tons,	0.14½	per ton	1,450.00
Hauling	10,000 tons,	0.08	per ton	800.00
Tipple and track	10,000 tons,	0.08	per ton	800.00
Superintendent	10,000 tons,	0.02	per ton	200.00
Repairs	10,000 tons,	0.04½	per ton	450.00
Incidental	10,000 tons,	0.03	per ton	300.00
Interest	10,000 tons,	0.03	per ton	300.00
Total		$0.66	per ton	$6,600.00

The expense of operating our 150-B shovel per month in coal stripping in Kansas and the amount of coal it will uncover in different depths of stripping at the rate of 1500 cu. yd. of stripping per day or 30,000 cu. yd. per month, are here given as follows:

150-B Shovel

1 engineer	per month,	$155.00	Coal uncovered per day at 1500 cu. yd. stripping on coal 3′–0″ thick, being a ton of coal under each square yard at surface.
1 craneman	per month,	100.00	
1 fireman	per month,	75.00	
3 pitmen	per month,	190.00	
1 watchman	per month,	75.00	
70 tons coal	per month,	70.00	
140,000 gal. water	per month,	50.00	15 ft. deep × 70′–0″ wide 300 tons
Oil, waste, pack'g	per month,	30.00	17 ft. deep × 65′–0″ wide 266 tons
Repairs and upkeep	per month,	100.00	18 ft. deep × 60′–0″ wide 250 tons
Share of supt.	per month,	75.00	21 ft. deep × 55′–0″ wide 217 tons
Incidental expense	per month,	100.00	25 ft. deep × 50′–0″ wide 187 tons
Interest on $20,000	per month,	100.00	27 ft. deep × 45′–0″ wide 166 tons
		$1,125.00	

At the rate of stripping 30,000 cu. yd. per month at an expense of $1125 makes the rate at 3¾ c. per cubic yard.

We submit herewith an estimate on the expense per ton of stripping, taking out the coal, hauling it to the tipple, screening it and delivering it on railroad cars, our 150-B revolving shovel to handle 21 ft. in depth of overburden at the rate of 1500 cu. yd. per day of 9 hr., uncovering 200 tons of coal per day or 4000 tons per month of coal 3 ft. thick as follows:

150-B Shovel			Expenses per Month
Steam shovel	4,000 tons,	$0.28 per ton	$1,125.00
Blasting	4,000 tons,	0.05 per ton	200.00
Loading	4,000 tons,	0.14 per ton	560.00
Hauling	4,000 tons,	0.07 per ton	280.00
Tipple and track	4,000 tons,	0.07 per ton	280.00
Superintendent	4,000 tons,	0.03¾ per ton	150.00
Repairs	4,000 tons,	0.04 per ton	160.00
Incidental	4,000 tons,	0.03½ per ton	140.00
Interest on $16,000	4,000 tons,	0.02 per ton	80.00
Total		$0.74¼ per ton	$2,970.00

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Coal Stripping in Illinois. *Colliery Engineer* (September, 1915), 69.

Wm. C. Perry: Perry-McMahan Coal Company's Stripping. *Excavating Engineer* (June, 1915), **11**, 333–34. Ills. Stripping from 18 to 35 ft. of overburden consisting of very hard slate or fire clay and earth.

Frank H. Kneeland: Large Stripping Operation. *Coal Age* (Sept. 25, 1915), **8**, 497–501. Ills. At Ebervale, Pa. A bed of coal which was abandoned many years ago is being in large measure reclaimed by stripping.

H. S. Mikesell: Coal Stripping in the West. *Coal* (Jan. 22, 1916), **9**, 164. Short note on stripping mines, and their locality.

D. C. Helms: Mining the Mammoth Vein with Steam Shovels. *Coal Age* (February, 1916), **9**, 322–24. Ills. A description of the method pursued at the Nesquehoning Colliery.

Anonymous: Coal Stripping Rush Run, Ohio. *Coal Age* (Jan. 22, 1916), **9**, 161–2. Ills. Description of the general method employed and the equipment required.

Steam Shovel Coal Stripping in the Danville District. *Coal Age* (Mar. 11, 1916), **9**, 448–452. Ills.

Mining with Electric Shovels. *Coal Age* (Mar. 25, 1916), **9**, 532–3.. Pinney Fork Coal Co. is the first to adopt coal stripping by electricity.

Wilbur G. Burroughs: Coal Stripping with an Electric Shovel. *Coal Age* (Feb., 1917), **11**, 304–306. Ills.

General Reference. *Handbook of Steam Shovel Work, The Excavating Engineer* Milwaukee, Wis. The Bucyrus Co., Milwaukee, Wis.

Discussion

J. B. Warriner, Lansford, Pa. (written discussion*).—The figures on costs given at the end of this paper are of particular interest and it is unfortunate that they necessarily cover such limited periods, as figures

* Received Oct. 4, 1917.

giving the costs over a period of at least a year would be of much greater value; for instance, it seems hardly probable that the performances credited to the various shovels for the months for which the costs are given would represent the average of the performances of these machines over a longer period. It would seem to me that the maximum of 4000 cu. yd. (3058 cu. m.) per day given for the 225 B shovel is much greater than could be maintained for an entire year's operation. On the other hand, the 1500-yd. figure for the 150 B shovel would seem a low average for this type of shovel.

In connection with the drag-line excavators, it will be interesting to know that the Lehigh Coal and Navigation Co. is about to install one of these machines on a clay stripping at their Nesquehoning colliery to excavate an average depth of overburden of 65 ft. (19 m.), which, due to local conditions, will require that part of the excavated material be handled a second time and a lesser quantity a third time. The results of this operation will undoubtedly be watched with great interest.

Merit Rating of Coal Mines Under Workmen's Compensation Insurance*

BY E. C. LEE, † B. S., E. M., PITTSBURGH, PA.

(St. Louis Meeting, October, 1917)

THE safety of mine workers has received more attention from both State and Federal law-making bodies than any other industry, a fact that shows clearly the hazardous nature of the industry. The last, but by no means the least, of the measures adopted for the protection of mine workers, is workmen's compensation legislation.

This is proving to be a powerful instrument for greater safety in the industries, from the fact that special stress is laid not on how the accident happened or who may have been responsible for it, but on how badly the employee is hurt by it. The injured person does not have to employ lawyers to prove his case, because the mere fact of his employment is proof of his right to compensation. The immediate effect of such legislation is the certainty confronting the mine operator that he must pay out large sums for every accident, regardless of its cause, thus adding materially to the cost of the production of his coal.

This fact furnished to the operator the strongest incentive to reduce the causes of accidents; not that the operator has ever lacked reasonable consideration for the welfare of his employees and a proper humane interest in their safety, but his engrossment in earning his daily livelihood has been so great that he has not felt that he had the time to devote to the safety of the operation. This time he must now find because of the actual cash value of the prevention of accidents in his mines.

This agency alone proved inadequate to bring about a material improvement because operators have always been willing enough to improve safety conditions when possible to do so within reasonable limits of their financial ability. The requirement in the Workmen's Compensation Laws of several of the States, that the mine operator shall insure as a guarantee of his ability to meet his compensation obligations, has furnished the needed incentive, moral and financial, necessary to the prompt and effective dissemination of a more general knowledge of the causes and

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* Originally presented at a joint meeting of the Chicago Section and the Illinois Mining Institute, at La Salle, Ill., on May 18, 1917.

† Chief Inspector of the Department of Inspection and Safety, The Associated Companies.

the means of preventing accidents, and of the cash value and the cost of removal of each mining hazard.

This latter condition is the result of the bringing together of 10 of the largest and strongest stock casualty insurance companies for the insurance and protection of coal mines. Mining is the last of the industries to benefit by insurance protection, because it has heretofore been the belief of insurance companies that the risks in mining were so great and their causes so little known that they could not safely underwrite mines. It was to take care of this risk in a safe manner, by providing a wide spread for the catastrophe hazard, and to promote a safety engineering service which should devote itself to the improvement of the safety conditions of the mines, that The Associated Companies was formed.

The result of this undertaking has been the standardization of inspection methods on the basis of accident causation. The activities of the Bureau of Mines in classifying and gathering statistics relative to the causes of accidents, in determining the nature of these causes, and in devising means for the prevention thereof, were a necessary preliminary to the method now taken for coal-mine insurance, since without the statistical and technical data so gathered, and the scientific principles thus evolved, it would be impracticable for insurance to offer graduated premium rates for relative safety.

An analysis was made of the various accident causes with a view to fixing a numerical value on each, and applying to each safety measure a corresponding credit and to each hazard a debit value. In this work, the originators of the schedule rating system for coal mines were materially aided by the engineers of the Bureau of Mines, as well as by several State mine inspectors, mine operators and higher officials of the mine workers' organizations, with the result that there was developed a system of inspection and rating of mines as to safety which is based on the percentage causes of accidents furnished by the statistics of each State and checked by the average for the whole United States.

Thus, for example, in Illinois falls of coal and roof cause 47.5 per cent. of the injuries, haulage accidents account for 12.5 per cent., and so on. Each of these primary causes is made up of separate hazards, and to each of these latter is given a percentage value. Thus, the frequency of accidents from falls of roof and coal is dependent on the character of the roof and rib, the amount and kind of timbering, supply of timber furnished at the working places, inspection and testing of roof, and a dozen other elements; similarly for shaft, haulage, electrical and other accidents.

The inspection of a mine by the engineer or inspector of an insurance company, with the fixing of numerical values for relative safety of each of the elements which may produce accidents, has a marked influence in drawing the attention of mine officials to the hazards of their operations, the existence of which may not have occurred to them. Even

State inspection, with its police powers, is not so influential in pressing home the nature of the risk and the value of preventive measures as is an increased insurance rate or a reduction in insurance premium under workmen's compensation legislation.

The problems of safety in mining thus become at once a matter of earning power for the mine. Instead of the operator's spending large sums of money in the employment of lawyers in combating litigation, these sums may be devoted to improving the safety of his mine, thereby earning a reduction in the insurance premium which under compensation legislation he feels compelled to pay. The effect on efficient and economic operation is further felt in the improvement of appliances whereby the accident causes are reduced. Thus, haulage accidents are largely the result of bad roadbed, light rails, bad track laying and maintenance, faulty illumination in the haulageway, a bad haulage system, and similar elements. Improvement in the character of the roadbed and track, and the cars and the haulage system, will add greatly to the safety from haulage accidents, and will greatly increase the efficiency of operation by permitting the hauling of larger loads and at higher speeds for each trip. Similarly, the adoption of undercutting as a means of preventing blown-out shots and the injuries resulting therefrom, leads under many conditions to an improvement in the size of the coal and in the efficiency of production, as will many other safety measures. And finally, the unknown financial obligation which must be met by placing a higher price per ton on the coal under uninsured liability legislation is replaced by a known and fixed obligation under compensation acts protected by stock company insurance, whereby the mine owner may figure to a nicety his liabilities on accident acco unt.

All of these improvements in mining, in efficiency and safety' have been the result of the employment of a higher grade of officials, largely those trained in mining engineering, and of a better trained corps of inspectors, and also of the technical experts in the service of the Bureau of Mines. The mining industry may now well look upon agencies which have heretofore seemed inimical to its welfare as fraught, on the contrary, with promise of a better future for the industry.

Considering now in detail the various causes of accidents to miners and the means that may be adopted for their prevention: an inspection of the statistics of coal-mine accidents as compiled by the Bureau of Mines from State inspectors' reports, and reduced to a comparable basis as to cause, shows that these may be divided into three main groups, viz., underground, shaft, and surface. These three groups may be each again divided into 10 or more separate causes, the methods for prevention of which are coming to be fairly well known.

There is another group of accident causes differing from those enumerated, in that the foregoing involve usually the safety of only one or of

a very few employees, whereas this other group may involve the safety of several or even all of the men working underground. These include, in the inverse order of their seriousness, mine fires, gas explosions, and coal-dust explosions.

Whatever other means may be adopted for reducing the number of injuries to mine workers, ultimately the strongest influence for good in this regard will be through education of the owners and managers of mines concerning the value to their property of the safeguarding of the lives of the workers, when they are subject to the positive liability for injury involved under workmen's compensation acts; the consequent necessity which the operator is under to secure insurance protection; and, finally, the tremendous influence for safety under such system which will come from the provision of skilled insurance inspection whereby reductions are given in premium rates for safe practices, based on the relative hazard of such practices and the means of preventing injuries, as developed through the investigations of the Bureau of Mines.

The effect of this new element in the mine safety problem has been felt in some measure in all of the coal-mining States in which there are workmen's compensation acts that encourage private insurance enterprise. These are the States of Illinois, Indiana, Iowa, Maryland, Kansas, and Michigan. Under legislation more favorable to compulsory competitive compensation insurance, the effect has been most strongly felt in the States of Pennsylvania and Kentucky.

In Pennsylvania, the law has been in effect for 17 months, since Jan. 1, 1916; and there is now no mine operator, be he insured with the State Fund or with The Associated Companies, who is not thoroughly familiar with the Mine Safety Standards of The Associated Companies, these standards having been adopted by the Pennsylvania Insurance Department for the inspection and schedule rating of all coal mines, no matter with whom they may be insured. The more than 2000 mines in Pennsylvania under the observation and protection of the inspectors of the insurance companies have, in this time, been inspected from two to ten times, depending on the condition of the mines themselves. A splendid coöperation has been built up between the operators, the State mine inspectors, the Federal Bureau of Mines, and the insurance inspectors, to the end that they are all working harmoniously along identical lines for the improvement of safety conditions in the mines concerned. The measure of this is to be found to some extent in an examination of the accident statistics for Pennsylvania for the years 1915 and 1916. In the anthracite mines of the State, there were 527 fatalities underground and 61 on the surface during 1915, while in 1916, the first year in which the compensation law was effective, there were 491 fatalities underground and 65 on the surface, a reduction of 32 for the year. In the bituminous mines for the same periods, there were 407 fatalities underground in

1915, and 35 on the surface, while in 1916 there were 412 fatalities underground and 21 on the surface, a reduction of 9 for the year. Thus, in the combined anthracite and bituminous mines of Pennsylvania, there was a reduction in the number of fatal accidents from 1030 in 1915 to 989 in 1916, a total reduction of 41 for the year. This was in spite of the fact that the output was practically normal, and that the opening of many new mines meant the introduction of considerable unskilled labor into the mines. It is, of course, impossible to determine to whom the credit for this reduction should go, but it seems fair to assert that some credit should go to all the agencies concerned.

The Compensation Law did not become effective in Kentucky until Aug. 1, 1916, so that the State has been operating under the law for only nine months, but a great improvement has been made in the safety condition of the mines, and it seems reasonable to predict that there should be a reduction in the accident record for the State. No statistics are at hand to determine whether or not this prediction is being fulfilled, and it may be rash to predict that this improvement will be reflected in the statistics for the first year under this plan, but the next 2 or 3 years should show a considerable improvement in this respect.

As evidencing the attitude of the mine operators to Workmen's Compensation Insurance thus administered by private stock insurance companies, it is well to record that for the first several months of this inspection service it was a subject of opposition and criticism. Since inspections have been completed and reduced premium rates promulgated in many cases as a result of improved conditions, a much more friendly attitude has been shown. One of the leading Pennsylvania bituminous operators says: "The operating officials at our different properties were first inclined to be a little skeptical and critical, believing that no inspector acting under State or insurance authority could show them how to improve the condition of their mines. Since we have gone through it, however, it gives me great pleasure to say that we have found your inspection of great service to us. Your inspectors have disclosed conditions which have enabled us to remove hazards, so that our properties are in far better condition as regards safety, and the change we have made in our organization and our methods puts us in close touch with our working force, with the result of increased efficiency and economy in operation. We shall be more than repaid for the expense we have been put to in making the changes recommended."

The Chief Mine Inspector of one of the large mining States was somewhat skeptical, as were others, as to the possibility of bringing about other improvements than were practicable through his Department, as a result of Workmen's Compensation application and the cost of insurance thereunder, and he wrote thus: "If all the suggestions contained in your Safety Standards are complied with, a marked reduction in accidents

will no doubt result. I have found that saying and doing are entirely different things, and if you can succeed in forcing compliance with these rules, you will accomplish what the State Mining Department has tried to accomplish for years, without complete success, and there should be a reduction of at least 50 per cent. in the number of accidents."

We have been unable to do as effective work in Illinois as we should have liked to do, largely because of the compensation laws in effect. Under these laws, the employer may insure his mines and maintain a benefit system, or he may reject the provisions of the act and continue under the old liability acts.[1] We believe that our method of work will in time bring about a marked reduction in the number of accidents in and about coal mines, and hope that conditions may change so that we may have an opportunity to work effectively in this State.

A concrete example of the manner in which insurance inspection for fixing merit rates under workmen's compensation has effected improvement in the condition of mines and the consequent safety of the workmen will show the value of this system of rating coal mines, from the standpoint both of the operator and of the insurance company. The following are selected as representative of conditions occasionally encountered among the more dangerous gaseous and explosive districts.

The following example of a typical mine of bituminous Pennsylvania, is printed by authority of John Whalen, General Superintendent of the Pittsburgh & Eastern Coal Co.:

Mine No. 1 of the Pittsburgh & Eastern Coal Co. On first inspection, reported Feb. 1, 1916, the mine was found to be in a very hazardous condition, due to the presence of large quantities of explosive gas and fine coal dust. The mine was otherwise unsafe, because of lack of protection to the mine workers from falls of roof and coal, careless handling of explosives, bad condition of the haulageways, etc. These conditions were especially due to the mistaken belief of the mine superintendent that the management desired economy in operation and cheapest possible tonnage cost of coal, and in consequence he devoted himself to getting out coal cheaply, to the neglect of safety conditions. These facts were immediately brought to the attention of the owners, with the result that one of the higher officials and the general manager immediately went over the situation with the insurance agents and inspectors.

The premium schedule rate computed for the insurance of this mine at the time of the first inspection was $5.81. During the next year several rating inspections were made, each of which disclosed improved conditions, and each of which was reported with recommendations for further improvement. All of these recommended improvements were invariably made by the owners as soon as received, with the result that on the date of the last inspection, Feb. 23, 1917, a year later, the mine

[1] July 1, 1917, compensation became compulsory in Illinois.

was in practically perfect safety condition, and was charged with a schedule rate of \$2.82. The estimated premium to be paid as a result of first inspection computed was \$11,840, and on last inspection \$5640, or a saving on the one mine of \$6200, more than enough to pay for making all the improvements recommended.

A typical anthracite mine, Pennsylvania, is that of the Buck Ridge Mining Co. This example is printed by the authority of Superintendent Raymond Lewis.

The rate resulting from first inspection, made Nov. 9, 1915, was \$4.50, with a total of 27.8 charge points, as against an average for the anthracite region of 30, or slightly better than average. The last inspection, made Jan. 2, 1917, or over a year later, produced a schedule rate of \$3.15 and a total of 5.94 charge points, a very material improvement in conditions. The estimated cost was \$9000 per annum, and on last inspection \$6700, a saving of \$2300.

A typical bituminous mine in the Central States is the Barker mine of the Federal Coal Co., near Pineville, Ky. This example is printed by the authority of General Manager M. S. Barker.

The adjusted schedule rate resulting from first inspection, made Aug. 1, 1916, was \$4.86, with a total of 34.1 charged, as against an average for the State of 25. On last inspection, made 6 months later, Jan. 17, 1917, the schedule rate was \$2.82, or a total of 4.0 charges. This resulted in the reduction of the total premium paid of about 40 per cent.

As a last example, the Oakdale mine of the Oakdale Coal Co., Colorado, which example is cited by the authority of Vice President and General Manager H. F. Nash:

On the occasion of first inspection, Aug. 18, 1915, there was computed an adjusted schedule rate of \$6.62, resulting from 34.6 charges. On the occasion of the last inspection, a year later, July 27, 1916, the adjusted schedule rate was \$4.70, total charges 7.1. The estimated total premium on first inspection was \$15,120 and on second inspection \$11,280, a saving to the mine owner of \$3840 per annum.

In detail, the relative safety condition of the mine on first and on last inspection was as follows:

The rate resulting from first inspection, made Aug. 18, 1915, was \$6.62, with a total of 34.62 charge points, as against an average for Colorado of 40, which is slightly better than the average. The last inspection, made June 27, 1916, produced a schedule rate of \$4.70, with total charge points of 7.12, a material improvement in the condition and the estimated total premium, which on first inspection was \$15,120 and on last inspection \$11,280, a saving of \$3840.

For further illustration of the effects of this system, I have taken at random several reports covering mines in Pennsylvania, Illinois and Colorado, showing the rate produced by each inspection and the average

number of men employed in each mine. The saving in insurance can readily be determined by assuming an average yearly earning per man.

Pennsylvania Bituminous Mines, Base Rate, $3.83

File No.	No. of Men	First Inspection Rate	Last Adjusted Rate	Reduction
70	469	$4.17	$3.270	$0.900
71	197	4.06	3.230	0.830
95	192	3.51	3.025	0.485
111	306	3.16	2.618	0.542
113	133	3.05	2.537	0.513
128	115	3.13	2.625	0.505
131	79	4.06	2.679	1.381
133	110	3.77	2.531	1.239
135	27	3.91	2.658	1.252
136	14	3.46	2.480	0.980
138	49	3.72	2.451	1.269
140	80	4.02	3.058	0.962
145	277	3.60	2.680	0.920
146	76	3.23	2.385	0.845
147	65	3.16	2.415	0.745
151	214	3.37	3.228	0.142
152	21	3.22	2.900	0.320
158	625	3.63	2.400	1.230
166	267	3.99	2.404	1.586
178	184	3.61	2.846	0.764

Illinois

File No.	No. of Men	First Inspection Rate	Last Inspection Rate	Reduction
25,005	190	$5.21	$4.36	$0.85
25,022	506	4.28	3.48	0.80
25,027	22	3.34	2.81	0.53

Colorado, Ex-Medical Base Rate, $6.30

File No.	No. of Men	First Rate	Second Rate	Third Rate
5,004	50	$4.96	$4.93	$4.36
5,007	57	5.17	4.79	4.31
5,010	113	6.31	5.18	4.18
5,019	43	4.94	4.75	4.28
5,038	85	5.03	4.82	4.10

Discussion

H. M. Wilson, Pittsburgh, Pa.—The statement in the paper[1] of T. T. Read, "Increasing Dividends Through Personnel Work," that the offering of substantial money rewards for avoidance of accidents is much more effective than mere propaganda or education respecting

[1] *Bulletin* No. 130 (October, 1917), 1833.

accident prevention, is a point fully substantiated in the experience of insurance inspection under workmen's compensation acts.

The first year under the Workmen's Compensation Law in Pennsylvania, during which over 2000 mines were under the observation and protection of the inspectors of the insurance companies, has resulted in several inspections of each mine, a splendid coöperation between the operators, the State mine inspectors and the insurance inspectors, and an average reduction in premium rate for insurance from $3.83 to $2.77, a reduction of more than $1 per $100 of payroll, on an average payroll of nearly $100,000,000. In other words, the insurance companies have voluntarily reduced their premium earnings by a million dollars, thereby making available to the mine operators that sum for the improvement of safety conditions in their mines.

This reduction in premium rate represents a corresponding improvement in the safety condition, an improvement which may be expressed by the statement that the average condition of the mines at the beginning of the year was 75 per cent. of perfect, and at the close of the year was 92 per cent. of perfect. That this improvement in condition is not merely theoretical would seem to be verified by the fact that the pure premium experience of the year, as submitted to and checked by the Insurance Commissioner of the State of Pennsylvania, has shown that a reduction in the amounts paid out for compensation for accidents during the year, in other words, the reduction in the number of accidents in these mines, was so great as to warrant a reduction from the manual or base rate of $3.83, to one of about $3.25 per $100 of payroll.

A few examples of the opinions of mine superintendents respecting the improvement in the safety conditions in their own mines produced by merit rating of insurance premium rates, may be of interest.

Supt. James McGuire of the Cross Creek mine, Waverly Coal & Coke Co., Pennsylvania, says "We have a better organization, brought about by safety committee meetings. Our foreman and assistant know that unless the roadways are clean, the proper clearance and shelter holes provided, as required by the standards of The Associated Companies, the matter will be brought to the attention of the manager by the inspector, as a reflection on the ability of the officials in the mine. Our accident record has been decreased by this method."

General Manager W. R. Wilburn of the Madeira Hill Coal Mining Co., Pennsylvania, states that "as a result of the insurance inspection under workmen's compensation, two additional superintendents have been added, and a mine safety committee and a central safety committee have been organized at our operations. Each superintendent has been instructed in first aid, and our superintendents now inspect the mines as a part of their regular duty. Guard rails have been placed around all moving machinery."

The Rich Hill Coal Mining Co. has trained in first aid 100 men out of a total of 300. It has improved its method of storing and handling material on the surface; and has introduced the use of permissible explosives, from which it claims to obtain as good results as from the use of less safe powders.

Numerous mines in other parts of the State of Pennsylvania, as well as in Illinois, Kentucky and other States, have made improvements in their haulage, their timbering, their ventilation, their general safety conditions, and in many gaseous mines the use of open lights or of safety lamps has been discontinued, electric having been provided; while in many non-gaseous mines the oil torch has been replaced by the oil lamp. Among such mines are those of the United Coal Co., the Whyel Coal & Coke Co., Ellsworth Collieries Co., Clyde Coal Co., Diamond Coal Co., Oliver & Snyder Steel Co., and many others.

The Coal Industry of Illinois

BY C. M. YOUNG,* B. S., E. M., URBANA, ILL.

(St. Louis Meeting, October, 1917)

THE following paper has been prepared with the object of placing on record in the *Transactions* some facts concerning the present condition and future prospects of the coal industry of Illinois. In presenting it, the writer wishes to say that a considerable amount of the material contained is taken from the publications and records of the Illinois Coal Mining Investigations, the annual Coal Report of the State Mining Board, and the work of the State Geological Survey.

Coal Beds and Mining Methods

According to the latest maps of the State Geological Survey, there are only 19 of the 102 counties of the State some portion of whose territory is not within the borders of the coal measures. If outlying areas in Calhoun and Whiteside Counties are considered, the number is reduced to 17. This does not mean that all of the 85 counties that are partly or wholly underlain by coal measures will become producers, as there are some portions of the State in which the coal does not occur in beds of such thickness as to warrant the hope that it can be mined. The mining of coal is widespread, however, for in 1916 there was production from 51 of the 102 counties of the State. There seem to be no cases in which operations will be limited by depth, as none of the coal measures lie at depths much greater than those at which it has been proved that coal may be profitably mined in this field. In fact, the deepest shaft reaching a bed of bituminous coal in the United States is located in this State at Assumption, where coal only 3½ ft. (1.07 m.) thick is worked at a depth of 1004 ft. (306 m.).

The coal beds occur in the form of a basin which is deepest along a line extending through the central part of the State and bearing a little west of north. The northern, western and southern borders of this basin lie within the boundaries of the State, but on the eastern side the Illinois field is continuous with the Indiana field. The glacial drift covers all except the extreme southern end of the State and the visible outcropping

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of the coal is much less frequent than it would be if it were not concealed by this material.

The older beds are found over a somewhat larger area than the more recent beds; for example, coal No. 1 is worked in Rock Island County at the extreme northwestern part of the coal area and as far south as Monmouth in Warren County. Southeast of this district are places where No. 2 is worked, as at Colchester, Macdonough County, and Augusta, Hancock County. Further to the southeast, workings are found in beds 5, 6, and 7.

The areas in which the different beds are most largely worked are shown in Fig. 1, a map showing the division of the State into districts for the purposes of the Illinois Coal Mining Investigations. These investigations are being carried on in coöperation by the University of Illinois Engineering Experiment Station, the State Geological Survey, and the United States Bureau of Mines. The districts are areas in which certain coals are worked under fairly uniform conditions. The districts and the beds worked in each are as follows:

District	Bed		
1	2		Longwall
2	2		Room-and-pillar
3	1 and 2		Room-and-pillar
4	5		Room-and-pillar
5	5		Room-and-pillar
6	6	East of Duquoin anticline	Room-and-pillar
7	6	West of Duquoin anticline	Room-and-pillar
8	6 and 7		Room-and-pillar and stripping.

Most of the coal of the State is produced from beds 1, 2, 5, 6, and 7. The production from bed No. 1 is small, amounting to only about a half million tons. If the indefinitely correlated coal at Assumption is excepted, the working of the No. 1 coal is confined to the territory in the neighborhood of Rock Island. In this district the average thickness of the coal is 4 ft.

It should be said that the term "No. 1" is somewhat indefinite in its application. Coal No. 2 is quite well marked, and anything occurring below it is called No. 1, though it is not certain that all coals called No. 1 belong to exactly the same horizon.

The production from bed No. 2 during the fiscal year ending June 30, 1916, was 4,305,330 tons. This bed is more widely distributed than any other and is to be found, so far as is now known, over the whole coal area of the State with the exception of a few counties in the east-central part. The coal is now mined in 16 counties, and if the Assumption mine is in No. 2 coal, in 17 counties.


JO DAVIESS
STEPHENSON
WINNEBAGO
BOONE
MC HENRY
LAKE
CARROLL
OGLE
COOK
DE KALB
KANE
DU PAGE
WHITESIDE
LEE
KENDALL
ROCK ISLAND
HENRY
BUREAU
LA SALLE
GRUNDY
WILL
MERCER
PUTNAM
STARK
MARSHALL
KANKAKEE
KNOX
WARREN
HENDERSON
PEORIA
WOODFORD
LIVINGSTON
IROQUOIS
FULTON
TAZEWELL
MC LEAN
FORD
HANCOCK
MC DONOUGH
SCHUYLER
MASON
DE WITT
CHAMPAIGN
VERMILION
ADAMS
BROWN
CASS
MENARD
LOGAN
PIATT
MACON
MORGAN
SCOTT
PIKE
SANGAMON
DOUGLAS
EDGAR
CHRISTIAN
MOULTRIE
COLES
GREENE
MACOUPIN
MONTGOMERY
SHELBY
CUMBERLAND
CLARK
CALHOUN
JERSEY
FAYETTE
EFFINGHAM
JASPER
CRAWFORD
MADISON
BOND
CLAY
RICHLAND
LAWRENCE
CLINTON
MARION
ST CLAIR
WAYNE
EDWARDS
WABASH
WASHINGTON
MONROE
JEFFERSON
HAMILTON
WHITE
RANDOLPH
PERRY
FRANKLIN
JACKSON
WILLIAMSON
SALINE
GALLATIN
UNION
JOHNSON
POPE
HARDIN
ALEXANDER
PULASKI
MASSAC
1
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DISTRICTS
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Overlap 3&4
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8
Scale:
0 10 20 30 40 50 100 Miles

FIG. 1.

The coal is of good quality for coals of this field, but it is not always of workable thickness, and is nowhere very thick. The greatest thickness is probably in Jackson County, where the two benches together have an average thickness of 5 ft. 9 in. (1.75 m.). In the longwall field (District 1) the average thickness is 3 ft. 2 in. (0.96 m.).

The No. 2 coal is most largely worked along the northern edge of the field in Bureau, LaSalle, Grundy, and Will Counties. It is also worked along the western border of the field, in Warren and McDonough Counties at the north, and especially in the neighborhood of Murphysboro, Jackson County, at the south. In this latter district the bed is divided into two benches by a parting varying from ⅛ in. to 36 ft. (3.175 mm. to 10.9 mm.) in thickness. The average thickness of the bottom bench is 3¾ ft. (1.15 m.) and that of the top bench 2 ft. (0.6 m.). Where the parting is less than 4 in. (101.6 mm.) thick the two benches of the seam are worked as one, but where it exceeds this thickness only the lower bench is worked. The area over which the two benches are worked together is approaching exhaustion, though there still remains a large area over which one bed alone can be worked. The coal of this district is better in quality than that from most other parts of the State, and finds a ready market at a higher price than that commanded by most of the other coals.

Coal beds are most numerous in the southern part of the State, and some mining on a small scale is done in some of the lower coals which outcrop to the south of the outcrop of No. 6. Coal thus mined is called No. 3. The same number is also popularly associated with the coal mined at Roanoke in Woodford County, and at Pleasant Plains in Sangamon County. However, the terms "No. 3" and "No. 4" do not belong to any definitely known coals, and no mining is done in any coal known as No. 4.

No. 5 is a well known bed occupying a large territory. It is extensively worked in the north-central part of the State and is again found of good thickness and quality in Saline and Gallatin Counties in the southeast part of the State.

Of the No. 5 coal there was produced in the fiscal year ending June 30, 1916, 12,986,274 tons. As found in the north-central part of the State, the average thickness of this coal is 4 ft. 8 in. (1.42 m). One of the characteristics of this district is the large number of clay veins crossing the coal bed. There are also small falls, slips, and rolls. There are numerous cases where the coal has been eroded and the space filled with drift. These conditions make it difficult to estimate the total contents of the bed in any area before it has been thoroughly developed.

In Saline and Gallatin Counties the No. 5 coal is of variable thickness, averaging 5 ft. 4 in. (1.63 m.) in Saline County and 4 ft. (1.22 m.) in Gallatin County. The No. 6 coal is also worked to some extent in

Gallatin County, and one shaft is to be sunk from No. 6 to No. 5, thus permitting the working of both beds. There are also in Saline County some operations in what is locally known as Coal No. 7, but this coal is really No. 6. In Gallatin County there are also some operations in the No. 6 coal and one in what is known as No. 9 coal.

In southern Gallatin County there is a district in which several beds of coal are found. These have not yet been correlated, and it is impossible at present to assign definite names to all of them, but such investigation as has been made shows that some of these coals at least are of excellent quality, better in fact than the No. 2 coal as known at present. This district has not yet been extensively developed, partly because the strata are somewhat folded and the conditions are unfamiliar to Illinois operators, and partly because there is no railroad to handle the coal. A railroad is now being built, and it is probable that coal from this district will soon reach the market.

All through the south-central part of the State the No. 6 coal is worked, and this is the bed from which the greatest tonnage is now produced. In Fig. 1, the area in which the No. 6 coal is worked is shown divided into two districts, 6 and 7. This is because the physical conditions are different on the two sides of the Duquoin anticline, which extends approximately north and south across eastern Perry County and into Jackson County. Because the No. 6 coal is of greater thickness and is generally found at greater depth on the east side of the anticline, Franklin County and the northern part of Williamson County have been separately studied. In these counties, and to some extent in Saline County east of them, there has been the most rapid growth of the industry in recent years, and in the same district present activities give promise of the greatest expansion in the near future.

According to E. W. Shaw and T. E. Savage, the thickness of No. 6 coal in District 6 ranges from 7 ft. 6 in. to 14 ft. (2.29 to 4.3 m.), and an average thickness of 9 ft. 9 in. (2.97 m.) was found in 130 borings.[1] There are limited areas, however, in which a thickness of 15 ft. (4.6 m.) or more is found.

A persistent band of impurity, known as the "blue band," is found wherever the No. 6 coal is known. Almost everywhere it is from 18 to 30 in. (0.46 to 0.76 m.) above the floor. No satisfactory explanation has been offered for the persistent occurrence of this bed at almost the same elevation and over a wide area.

Near the top of the No. 6 coal in District 6 is a persistent parting, at which the coal of the main bench is easily separated from the top coal. Neither main bench nor top coal is of uniform thickness. In some places

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[1] Murphysboro-Herrin Folio, Illinois. *U. S. Geological Survey, Folio* No. 185, (1912).

the main bench is only about 6 or 7 ft. (1.8 or 2.2 m.) thick, and in others it reaches perhaps 10 ft. (3 m.). The top coal varies in thickness from about 18 in. (0.46 m.) to about 5 ft. (1.5 m.). The greatest thicknesses of the two benches commonly accompany each other.

There is no uniform difference in quality between the main bench and the top coal. While it is true that neither one is uniformly superior to the other, it has been found that the top coal is occasionally superior to any known coal of the main bench. According to Parr,[2] at certain places where the ash content of the lower bench is 6.41, 6.98, and 9.46 per cent., that of the top coal is only 3.60, 3.05, and 3.12 per cent. respectively. The heat values of the two benches at the same places, "as received," are: top coal, 11,912, 12,786, 12,275; lower coal, 12,000, 12,001, 11,395 B.t.u. These analyses show that there are places in which the top coal is quite noticeably superior to that of the main bench. No attempt has been made to mine and ship these coals separately, but it seems possible that this may be done at some time in places where the top coal is of such high quality as to make it desirable, and especially where the bed is thick. At the present time a considerable part of this top coal is left as a permanent roof.

On the west side of the Duquoin anticline the thickness of the No. 6 coal ranges from 2½ to 14 ft. (0.76 to 4.3 m.) and averages 7 ft. (2 m.). In this part of the district, also, there is a top coal which is left up where a black shale roof is found and where the coal is 7 ft. thick or more. According to the Coal Report, the production of No. 6 coal in the fiscal year ending June 30, 1916, was 43,011,832 tons.

Near the southern end of the State at approximately latitude 37° 30′ N., the coal measures are brought to the surface by a general uplift and the coal has been eroded along a line extending a little south and east from the neighborhood of Murphysboro. Some of the coals are found again in Kentucky.

The No. 7 coal is worked to a small extent in the north central part of the State but the principal production is from the neighborhood of Danville, where this coal has an average thickness of 5 ft. (1.5 m.). Special interest attaches to this district because it was here that the first large revolving steam shovel was put in operation and proved so successful that similar and larger shovels were used here and in other districts. The No. 6 coal is worked in the same neighborhood but by ordinary mining methods. The production of No. 7 coal in the fiscal period referred to was 2,649,280 tons.

Outside of the longwall district, most of the operations in the State are conducted with some form of the room-and-pillar system. In most

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[2] S. W. Parr: Chemical Studies of Illinois Coals. Illinois Coal Mining Investigations, *Bulletin* 3 (1915), 51.

cases this is either a cross-entry method or a so-called panel method. In the latter, however, there is not often sufficient care in maintaining the isolation of the groups of rooms turned from room entries to warrant the use of the term panel.

One of the advantages to be derived from the application of the panel method is the limitation of squeezes. The mining practice of Illinois has been such that these are of frequent occurrence. It has been and still is the general custom to obtain most of the coal on the advance, with little regard to the extraction of pillar coal. Under these circumstances, an effort is made to obtain as much coal as possible from the rooms and entries, with the result that pillars are left too small to sustain the weight upon them without movement, and squeezes are produced.

In most parts of the State the coal is overlain by one or more beds of rock sufficiently strong to resist breaking in the ordinary course of mining. When a pillar fails, these strong beds transfer the weight of the overlying material to the surrounding pillars, with the result that these are likely to fail. A squeeze thus produced may travel over a considerable distance until it is stopped by some pillar sufficiently strong to bear the weight, or until the strong rock breaks.

One result of such movements is the production of subsidence upon the surface, and, because of the high value of Illinois lands over a considerable part of the State and of the poor surface drainage due to the level topography, subsidences are more noticeable than in most other bituminous districts. For this reason there has been a tendency on the part of coal producers to plan their operations with the idea of leaving in the ground sufficient coal to maintain the surface in its original position. In some cases this appears to have been done successfully at the expense of an average loss in the ground of about 50 per cent. of the coal. In other cases the attempts to prevent squeezes and subsidence have not been successful. It seems very desirable that plans should be adopted in which the subsidence of the rock overlying the coal will be recognized as a necessary accompaniment of mining and in which the damage from movement will be minimized by such planning of the workings as will confine the movement to districts ready to be abandoned. This can probably be done only by making the workings of such dimensions that there will be no danger of subsidence during the advance, and taking out all or nearly all of the coal on the retreat, so that the subsiding top will have no partial support to retard its fall and promote squeezes.

In most cases it is possible by proper tiling and ditching to restore the surface to good condition for agricultural purposes, and the subsidence of the surface should not be considered an obstacle to high extraction except in cases where the coal producers do not own surface rights and are forced to pay for surface subsidence much in excess of a just compensation for the damage done.

In spite of the long line of the boundary of the coal district, there is little stripping except in the Danville district. In most cases this is due to the fact that the surface is covered with glacial drift so that the coal is not exposed. Some stripping is done in the southern part of the State, and it is probable that more will be done, as there is a considerable stretch of territory where the No. 6 coal approaches close enough to the surface to be profitably extracted by this method.

The mining practices in the different districts are for the most part founded upon the cheap production of coal of which the value in the ground is low. In the symposium on Studies of Illinois Coals,[3] is given a discussion by George S. Rice of the relation between the low price paid for coal in the ground and the mining methods employed in extracting it. Attention is there drawn to the fact that coal rights are sold for very low prices and that for this reason there has been little incentive, so far as the value of coal in the ground is concerned, to extract the highest possible percentage of the coal. On the other hand, the desire of the producers has been to place coal on the market at so low a price that it could compete with other coals.

Since the publication of this symposium there has been some increase in the values of coal rights, but even yet the value of coal in the ground has not reached such a point as to make high extraction a necessary consideration in the choice of mining methods. Therefore, it is found that over a considerable part of the State wasteful methods are employed and it is safe to say that an average of only about 50 per cent. of the coal in the ground is taken out, and that the remainder is left in such form that it cannot later be extracted at a profit.

Shortly before the beginning of the present abnormal conditions in the industry due to the war, there was evidence of a desire on the part of several coal producers to increase the percentage of coal won, and it is probable that various plans for the increase of the percentage of extraction which have been temporarily abandoned will be taken up again when more nearly normal conditions exist.

An exception to the statement made concerning low extraction is found in the longwall district, whose location is shown in Fig. 1. Formerly beds 2 and 5, known locally respectively as the third and second veins, were worked by room-and-pillar methods, but after a time the longwall system was introduced and since that time the No. 2 bed has been worked by this system while nearly all the operations in the No. 5 coal have been abandoned. It is to be supposed, however, that the No. 5 coal will be worked later.

The extensive development of southern Illinois coals and the establishment of freight rates that permit these more cheaply mined coals to enter

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[3] *Trans.* (1909), **40**, 3.

the markets formerly supplied by coal from the northern part of the State, greatly interfered with the prosperity of the longwall district, and the termination of some of the operations was considered a possibility. A recent application of new methods has greatly improved the prospects of the coal industry in this district.

The only remedy for the depressing conditions seemed to be in a decrease of the cost of production, which apparently could be brought about by the introduction of machine mining and electric haulage. According to C. C. Swift,[4] the first attempt at machine mining was made some 20 years ago or more. Not unnaturally this early attempt was not successful. The first experimental work in recent years was commenced in September, 1914, and the machine tried was adopted. Other experiments resulted in improvements in the machines used, and in June, 1917, 11 machines were in operation and some 40 more had been ordered.

Whether the introduction of machines would have been so rapid if the unprecedented demand of the last few months had not occurred is open to question, but the movement had already been started and the use of machines had been decided upon in some cases before there were any indications of the present large demand upon the resources of the district. In fact, electric haulage and machine cutting had been adopted not as a means for meeting an increased demand, but as a measure of economy which would permit the continued operation of the mines under adverse conditions.

Growth and Condition of the Industry

An inspection of the present condition of the coal-mining industry of Illinois and of its past history shows that this industry has not yet entirely emerged from the formative period. It is not unified; forces are constantly pulling in different directions, so that there is lack of harmony and no possibility of wide planning for the most economical conduct of operations along lines leading to full utilization of the coal deposits. Among the operators the desirability of harmonious relations is beginning to be felt, but there is as yet little practical coöperation. The competition between individual operators working in the same parts of the district has been partially eliminated in some cases, though not in all; but there is by no means such a harmonious relation of different districts over the State as would make possible the economical distribution of output and the elimination of a part of the waste due to over-capacity and destructive competition.

It is probable that coöperation among producers will become thoroughly established only after some wide control of the industry has been

[4] C. C. Swift, General Manager, LaSalle County Carbon Coal Co., personal communication.

assumed by the State or by the Nation. Such desire for coöperation as is felt is hampered in transformation into effective results by laws intended to prevent improper combinations of interests controlling industries. While these laws were undoubtedly necessary in view of the conduct of some combinations of capital in the past, there can be no question that they operate to prevent such coöperative control of industry in some cases as would work to the ultimate benefit of all parties concerned. At present the public does not trust any combination that seeks to do away with competition, and probably will not until there is such machinery for the just and adequate control of combinations as will certainly and permanently remove all danger of abuse.

It is true that the companies owning the larger resources are in position to plan for long periods of time, but this fact has not yet led to such methods of mining as seem desirable. The endeavor still is to produce coal at what is thought to be the lowest cost, with little if any regard to the waste of immense quantities of coal in the ground. When we realize that the production in the fiscal year ending June 30, 1916, was 63,673,530 tons, and that practically the same amount of coal was left in the ground in an unrecoverable form, we see that the industry is not conservatively conducted.

This is not wholly a reason for criticism of coal operators, for the industry has developed along the line which made its development possible under existing commercial conditions; that is, by the production of coal at so low a cost that it can be sold in competition with other coals.

When there is unrestricted competition in an industry which is occasionally if not always profitable, there is likely to be over-development, as there is in Illinois at present. With over-development there cannot be the most economical working, but an effort is made to produce coal as cheaply as possible without regard to waste of coal in the ground. Some operators are endeavoring to remedy the present conditions and there is no doubt that improved methods of mining will be adopted in the future.

In view of the present demand for American coal and the probable future demand, it is interesting to observe the changes that are taking place and to look into the possibility of maintaining the present rate of output or of increasing it. In the fiscal year ending June 30, 1916, the production of Illinois coal was greater than that of any preceding year but the record can probably be easily exceeded if there is demand for the coal and if the output of the mines can be transported.

A change in the nature of the industry has been going on for some years with regard to the output of individual mines, and this is naturally accompanied by changes in the number of mines. To go back to 1906, at which time the number was greatest, we find that there were 1018 coal mines in the State. By 1916 this number had decreased to 803,

yet the number of men employed in the industry had increased from 62,283 to 75,919, and the amount of coal produced had shown an even greater increase, rising from 38,317,581 tons in 1906 to 63,673,530 tons in the fiscal year ending in 1916.

One of the changes in the industry shown by these figures is in the rate of production per employee. In 1906 this was 615 tons per man, and in 1916 it was 838 tons per man. It may be noted at the same time that the wages paid per ton have increased in the same period. This change in rate of production per man is probably accounted for principally by two things: first, the increased use of explosives; and second, the increased use of mining machines.

There is apparent a change in the character of the operations with regard to the output from individual mines similar to that which has taken place in some of the older districts; that is, the gradual increase in the average output is accompanied by a decrease in the number of small producers. In other words, there has been an increase of large producers. To make comparisons between the years that have previously been used, the average production in the fiscal year ending June 30, 1906, was 37,640 tons, while in 1916 it was 79,294 tons.

This tendency toward larger production from individual mines is further indicated by the decrease in the number of shipping mines from 419 in the fiscal year ending in 1906 to 284 in 1916. Meanwhile the production from such mines had increased from 37,133,811 tons in 1906 to 62,283,236 tons in 1916, the average output of shipping mines having increased from 88,624 tons to 219,307 tons.

There has been a slight decrease in the number of local mines (that is, mines which do not ship coal by rail), the number being 599 in 1906 and 519 in 1916. The production from mines of this class has increased slightly, being 1,194,770 tons in 1906 and 1,390,294 tons in 1916. The average output from these mines was 1994 tons in 1906 and 2678 tons in 1916. It is apparent that the proportion of coal produced at local mines is constantly decreasing.

The increase in the importance of large producers is further shown by the fact that the number of mines producing 200,000 tons or more has increased in the period under discussion from 47 to 111, or from 4.62 per cent. of the total number to 13.82 per cent.

The distribution of the increased production from the shipping mines is interesting as an indication of the tendency toward large outputs. In the fiscal year ending in 1906, the mines producing 200,000 tons or more per year had a combined output of 12,565,030 tons, while in 1916 the output of this class had reached 48,375,758. No other class of mines showed an increase for this period as a whole, but all classes producing less than 200,000 tons showed a decrease. In other words, the increased production in the State has come entirely from the larger mines. As the

the total increase in this period was 25,355,949, and the increase in production from the larger mines was 35,810,728 tons, we see that these mines have not only produced all of the increased tonnage, but that they have made up for the decrease in production of 10,454,779 tons from mines of the other classes. This does not mean that individual mines of the smaller classes may not have shown increases in production, but that the mines producing 200,000 tons or more are the only ones showing increased production as a class.

This seems to indicate a struggle for existence in which the largest mines are proving victorious. Probably this is in part due to cheaper production from these large mines, but it is probably also due to the financial condition of the companies operating them. The past few years have been a period of very low profits for producers of Illinois coal in general, and many were unable to survive this period without being placed in the hands of receivers, while a considerable number succumbed entirely. This may account for the decreased production from smaller mines because of the decreased number of such mines. The increased production from the larger mines may be due in part to greater ability to dispose of the coal, for it seems, from an inspection of the tables of the Coal Report, that the operation of these smaller mines continued for a smaller number of days than that of the larger mines.

The concentration of production at larger mines is shown by the number of mines whose output in the fiscal year ending June 30, 1916, was 100,000 tons or more. No mine in the State reached an output of 1,000,000 tons, though Superior No. 3, came within less than 4 days' average output of it.

Production Over Tons	Number of Mines
100,000	171
200,000	111
300,000	74
400,000	53
500,000	37
600,000	22
700,000	14
800,000	7
900,000	5

It may be interesting as illustrating the capacities of some of the Illinois mines to mention some of the daily averages and the largest day's outputs. Considering the fiscal year ending June 30, 1916, the largest average daily output was made at the No. 8 mine of the Old Ben Coal Corporation at West Frankfort, Franklin County. This average was 4865 tons. The mine was operated 193 days. The depth of the shaft is 460 ft. (140 m.).

The total output of this mine was 938,868 tons, and the other of the

two mines then operated by this company produced 743,651 tons. The company has recently acquired four other mines, whose combined production in the same period was 2,474,542 tons.

In some ways, the most impressive record of production is that made at the Superior Coal Co.'s No. 3 mine at Gillespie. This mine produced, in the fiscal year ending June 30, 1916, the largest quantity of coal taken from any one mine in the State, 985,482 tons. The average daily production was 4380 tons. A somewhat better record was made in the calendar year 1916, as the rate of production reached 4403 tons and the total output was 1,023,753 tons in 232½ working days.

Besides this large average daily production and large output, the same mine holds the world's record, so far as can be learned, for production from a shaft mine in a single day. The highest record yet made is 5502 tons in 8 hr., an average of 687.7 tons per hour. This required an average of about 3½ hoists per minute throughout the 8 hr. The depth of the shaft is 340 ft. (103.6 m.). A steam hoist is used.

At the present time this company is opening its No. 4 mine in the same field. It is announced that a daily output of 6000 tons is expected, and there is every reason to expect 7000 tons or more when the operation is fully developed.

The closest rival of Superior No. 3 in point of rapid production is found at the Livingston mine of the New Staunton Coal Co., which is in the same district and is operated under very similar conditions. In the fiscal year ending June 30, 1916, 961,726 tons of coal were produced from this mine, and the average daily output was 4646 tons.

The highest day's output was made on Sept. 30, 1916, when 5287 tons of coal were hoisted in 8 hr. This is an average of 660.8 tons per hour, more than most mines could produce in a day at a time easily remembered by the older persons associated with the industry. There were 1796 hoists, an average of 3.7 hoists per minute. The depth of the shaft is 287 ft. (87.5 m.).

That the coal industry of the State is not in as good condition as is desirable is shown by the fact that the average number of days worked in the fiscal year ending June 30, 1916, according to the Coal Report, was 163. This is hardly a fair statement, for it is the average days worked at all of the mines, and is not a weighted average. If we consider only mines producing more than 100,000 tons in the fiscal year, the number of days of operation was 203.

It is true that the coal industry is a seasonal one to a considerable extent, but not to so great an extent as to warrant an idleness of one-third of the time. If with such a length of idle time the mines produced enough to supply the demand, it is very apparent that there were too many mines and that a smaller number operating more steadily could have supplied the market. The steadier operation of a smaller num-

ber of mines would require the stocking of coal and this subject is likely to receive much more attention in the future than has been given it in the past.

This is one reason for the low profits realized by producers. The cost of operation under such conditions is higher per ton produced than it would be with more steady work, for all the elements of cost, except wages and part of the power and the wear of machinery, continue through idle times. Moreover, the selling price of the coal is made low by the active competition, as the operator naturally desires to keep his mine in operation during as much of the time as possible, and to do this he is inclined to cut the selling price of his coal to the lowest point at which he can make a profit. In many cases this point has been passed and the price has been cut to the cost of production and even to less than this in order that the operation of the mine might be continuous. In some cases this may be good business because it makes possible the maintenance of an efficient working force which might be scattered if the work became too irregular. It would be much better, however, if the operations could be made continuous without sacrifice. These facts have been realized by operators, but they have seen no way to amend the condition without placing themselves in a position to be threatened with prosecution for the violation of Federal laws.

One remedy for this condition of over-production is increase of consumption, and the coal producers of this district have been paying a considerable amount of attention to the increase of markets. Naturally the Illinois coal moves principally to the north and northwest, because the State is environed on the east, south, and west by other coal-producing States. To the east lies Indiana with coals similar to those of Illinois, while farther to the east are Ohio, Pennsylvania, and West Virginia. To the south and east lies Kentucky, whose western coals are similar in type to those of Illinois, while the eastern coals more closely resemble those of the Appalachian region. All of these States ship considerable amounts of coal into Illinois, and even farther to the west and to the northwest.

Illinois is subject to practically no local competition from the west. The Illinois coals, however, have to seek outlet to the west, and in doing so come into competition with the coals of Iowa, Missouri, and Kansas.

To the northwest there is a large field which is a natural outlet for the coals of Illinois. This field, however, was already occupied when Illinois became a large producer, and coals from farther east can reach it at a comparatively low cost because of the advantages of transportation by water over a large part of the distance. This fact explains the interest of coal producers of Ohio, Pennsylvania, West Virginia, and Kentucky, as well as of Illinois, in the cost of transportation into the northwest.

The cost of transportation also has a large part in controlling the movement of Illinois coals toward the west, a reduction in freight rates having made it possible to place some Illinois coals in the market in Kansas City in competition with the Kansas and Missouri coals, which find a natural market in that district.

It is a highly desirable thing for the ultimate welfare of the industry that there should be some control of the markets and the cost of transportation. While the Interstate Commerce Commission properly directs its energies toward the just regulation of freight rates with regard to the cost of the service rendered, it has no right to so control these rates as to lead to the most advantageous development of each coal-producing district. It would be better for the fuel industry if there were some such control in this country as exists in some European countries, through which there should be a proper regulation of output and of markets, so that each district might have its share of the business and at the same time be assured of a fair profit, instead of being forced to surrender almost all, if not all, of that profit in an effort to maintain its position in the industry. I have no doubt that the operators of Illinois, as well as of most other States, would welcome such control, if they could only feel sure that it would be a constructive one.

Capacity of Illinois Coal Mines

An interesting subject at the present time is that of the capacity of the different coal-producing districts. The fact that the mines of Illinois are idle, on an average, during about one-third of the time, shows that the production might be made much larger than it is. The simplest way to get at the possible annual output of the mines is to multiply the average production per working day by 300. An objection to this method is the fact that the average number of days worked is not a weighted average, but is independent of the capacity of the mines. However, if we assume that the average production for the State is about 60,000,000 tons per year and that the mines are operated for 200 days per year, we may say that the capacity of the mines is about 90,000,000 tons per year. This assumption, however, does not allow for the necessity of occasional extensive development work and repairs. It is the custom to assume that there will be a period of low activity during the warm months and to plan for doing some of the development work, construction and repairs during this time. Undoubtedly some of this work could be distributed over the year, but not all of it, and it would be necessary to make some allowance for the time needed for it. This necessity, however, may be balanced against the facts that most mines are not operated at their highest capacity during all of the working period, and that the days counted as working days are probably not all full days. There are no

definite figures available, but it seems safe to say that the mines of Illinois are easily capable of a steady production of not much if any less than 90,000,000 tons per year. The opening of new mines and the better equipment of old ones during the past year have undoubtedly increased the capacity beyond the point indicated by the figures available, but it is not possible at present to say how far.

Discussion

Carl Scholz, Chicago, Ill.—Mr. Young was kind enough to allude to the development of a mine in which I am now engaged. Several months ago the Burlington Railroad decided to open a mine in the field which they own in Franklin County containing some 15,000 acres (6070 ha.) of coal, ranging in thickness from 8 to 12 ft. (2 to 3.6 m.). This proposition in itself is attractive to any mining engineer, but with the backing of a large company I saw an opportunity to do some novel work in the construction line, and for several months I have been investigating and working on the problem of developing what I hope will be the largest and best equipped mine in the State of Illinois.

The shaft will be sunk at a place where the coal lies at a depth of 600 ft. (182 m.) and conditions both of the surface and of the coal are ideal in every way; the seam is almost level, and we had only 7 ft. (2 m.) of surface clay to go through. We are going to hoist by skips in order to reduce the rope speed. I believe that is a long step in the direction of economy, because we not only double the output of the mine by hoisting the contents of two cars at one time, but we reduce the proportion of dead weight. In our case there will be 20 tons of coal and 18,000 lb. of skip, or 55 per cent. net and 45 per cent. tare. That, of course, necessitates less than half the rope speed for the same tonnage, and we all know that speed means power and power means cost. Skip hoisting permits the use of solid-end gate cars, which is an advantage. The cost of these cars will be in the neighborhood of $250 apiece. They are well built, with roller-bearing wheels of the most up-to-date type, spring draw-bars, and refinements of that character which are necessary to move the loads most advantageously.

The most important departure at this mine is the adoption of our air shaft for other purposes in addition to ventilation. The air shaft is the largest shaft at the mine, having clear dimensions of 13 by 29 ft. (3.9 by 8.8 m.), and will contain four compartments. The air compartment will be 10 by 13½, or 135 sq. ft. of area, because the haulage ways will be very long, an ultimate distance of 2 miles in all directions from the shaft. The second compartment contains a stairway and the other two are for cages. One cage will be utilized temporarily for hoisting coal, and is equipped with a platform cage 7 by 12½ ft. The other cage has only half the floor area, but has two decks and will carry as many men as the main cage. The main cage is long enough to

accommodate mining machines, locomotives, and other large appliances, without having to take them apart. The two-deck cage weighs the same as the hoisting cage, the dead loads thus counterbalancing. When coal is being hoisted, we shall put on the double-deck cage a 9000-lb. counterweight, a steel car filled with sand, in order to counterbalance the loads absolutely. When hoisting coal, the weight on the cage will be 14,000 lb.—10,000 lb. of coal and 4000 lb. of car; the net load on the hoisting motor will therefore be 5000 lb. When the empty car goes down, the hoisting load will be 9000 lb. minus 4000 lb. In this way, peak loads are almost eliminated.

The air-shaft tipple will be built of steel and will be used temporarily for the hoisting of coal until the main tipple is built. It will, however, remain as a permanent feature and will be used for the hoisting of domestic coal and for the disposition of rock; it will be able to produce 1200 tons a day economically. The main hoist will be electrically driven and capable of making 555 trips in 6½ hr., with 8-sec. loading delays. That will give us a capacity of 6000 tons in 6½ hr., if all goes well; but I have allowed 20 per cent. time losses, which I hope wi l not be necessary. We should be able to obtain an output of 6000 tons at the main tipple and 1000 tons at the air-shaft tipple, if desired.

The mine will be equipped electrically throughout. The cost of the electrical equipment is very high at present. The latest estimates indicate that the main electric hoist will cost about $100,000 but when compared with the cost of steam, it means a saving of nearly $30,000 a year in operating cost, a figure which was quite a revelation to me.

The mining system that we expect to adopt is one which prevails in Franklin County, pillar and room workings, but I hope to improve upon present methods, because the waste in Franklin County is great. While this mine will supply coal for company use only, the interests of the railroad company lie in the conservation of the field for generations to come, because the coal in the ground is the only asset that the company has to offset its expense for building several hundred miles of railroad.

We have decided to use combination storage-battery and trolley locomotives after a very thorough investigation, because transportation, to my mind, is the most essential thing in coal-mining operation. These locomotives are equipped with 100 A6 cells, which we hope will give us 8-hr. gathering force on level track. The motors are wound so as to work on a trolley as well as from a battery, in anticipation of grades that we may encounter, and also that we may be able to run on trolleys when in the main entries, where a regular 250-volt trolley system will be installed. In case a larger battery is needed for 8-hr. gathering work, there is nothing to prevent us from installing it; and if the storage battery is not a success, we have lost only the value of it and can simply put some pig iron on the locomotive and use it as a trolley machine

exclusively. That seemed the safest method to adopt for a mine in which the grades were not definitely known.

John Stevenson, Jr., Sharon, Pa.—I am surprised to hear Mr. Scholz say that he proposes to mine over 5000 tons per day, and yet will buy his electric power instead of making it at the mine. Has Mr. Scholz also considered the market conditions in his district, where competition is already active?

Carl Scholz.—First as to the power. We can buy electric power from the Central Illinois Co. for a good deal less than it would cost us to make it, because we need power for only 8 hr. a day, and at a time when the power company has no other demand for it; consequently the power company is able to make us a price which is less than our cost would be. Furthermore, the high price of electrical equipment at this time makes it inadvisable to purchase generating plant, the great cost of which would necessitate higher depreciation and interest charges than in normal times.

As for market conditions, our mine has a steady customer for all the coal we can produce, for so long as there is any coal left in the ground, at a very satisfactory price.

F. F. Jorgensen, Gillespie, Ill.—Our mine No. 3 produced 5500 tons in one day, for a record, and the daily average for the last fiscal year was a little under 4500 tons; it is a shaft mine, but we do not use skips. At our new mine we expect to get 6000 to 6500 tons a day. We considered skips for our new mine No. 4, which we are now sinking, but did not put them in—we may regret it—but we shall equip this shaft in such manner that we can put in skips afterward, if we so desire. We shall watch with interest the operation of the skips at Mr. Scholz's new mine. Just now we are hoisting the coal through the air shaft, which is somewhat smaller than Mr. Scholz's air shaft. It is 11 by 17 ft. (3.35 by 5.18 m.) and the main shaft is 11 by 21 ft. (3.35 by 6.4 m.) inside the concrete. We shall have a concrete stairway, but shall not use cages in the air shaft. I believe that is a very good plan, but our company thought one shaft "half the size of the township," as one visitor expressed it, was enough.

In regard to our hoisting equipment, I agree with Mr. Scholz that we can hoist coal cheaper with electricity than we can with steam power. However, the question of reliability of electric hoisting has not been settled. There has been some trouble with the large electric hoists at some mines in the State, which I believe has not yet been overcome, and so we decided on a steam hoist. This will have 26 by 42-in. (66 by 106.7-cm.) cylinders with an 8-ft. (2.4-m.) boiler-steel drum. Our steam plant will consist of three 500-hp., water-tube boilers of the Erie City type, built for 200-lb. pressure and 150° superheat. We shall use turbo-generator sets for our electric power, a small one of 375 k.-v.-a.

for night load and a large one of 938 k.-v.-a. for the day load. If we wish to make this our central power station, we have made provision for extending the boiler and the power plant. The boiler feed pumps will be of turbine type.

Our underground locomotives will probably be the same as we have used in the other mines: 15-ton locomotives for the main-line hauling, while the smaller ones, for gathering, will be either the trolley or the storage-battery type. We are trying storage batteries now, and although we are not entirely satisfied yet, I believe they will ultimately be successful.

H. M. Wilson, Pittsburgh, Pa.—I should like to ask expert operators like Mr. Jorgensen and Mr. Scholz about the relative safety of the storage-battery locomotive for gathering service near the face in a gaseous mine. Trolley locomotives you can restrict, but accidents in mines are usually caused by disobedience of orders. The apparent tendency is toward trolley locomotives when you are unprotected.

F. F. Jorgensen.—We have but little gas in our field, but in a gassy mine I believe the storage-battery locomotive is safer than the trolley locomotive. In the storage-battery locomotive there will be less sparking at the commutator on account of the low voltage, while a trolley motor uses a 250-volt current and has a much greater chance of sparking, not only at the commutator and the trolley wire, but also at the contact of the wheels with the rails.

Lead Mining and Smelting at Galetta, Ont.

BY WILLIAM E. NEWNAM,* B. S., COLLINSVILLE, ILL.

(St. Louis Meeting, October, 1917)

LEAD mining has been carried on in several localities of the Province of Ontario in a desultory fashion for the past 60 years, but up to 1916 the results have not been of much commercial importance. The most ambitious attempt was made by an English company which built a lead smelter at Kingston, Ont., in 1880 to treat the concentrates from the Frontenac mines. After 2 years' operation, the mines and smelter were abandoned. Up to the present there has not been a sufficient tonnage of concentrates to support an efficient blast-furnace plant and as most of the small properties could not stand the heavy freight and duty charges into the United States, development work has not been carried on in a systematic manner. As a result the industry has languished.

At Galetta, in southeastern Ontario, considerable prospecting and development work has been carried on in the last few years and very promising deposits of galena have been found in the Chats Island group by the James Robertson Co., Ltd., of Montreal. The chief deposit is that of the Galetta or Kingdon lead mine, located on Chats Island in the Ottawa River about 5 miles east of the town of Arnprior.

The rocks[1] in the vicinity of the mines consist of an interbedded series of crystalline limestone and biotite gneisses of pre-Cambrian age. This part of the Ottawa River basin has been severely faulted and the overlying Paleozoic limestones may be seen in normal fault contact with the pre-Cambrian. The vein filling is of a very well-marked fault fissure type and although the amount of displacement is not determined at this point, similar faults to the east show a displacement of from 1,500 to 1,800 ft. which would indicate mineralized fault fissures of considerable depth.

The ore occurs in a highly crystalline calcite with some barite and fluorite, and the galena occurs more or less richly disseminated in clusters and crystal aggregates. In some cases these masses weigh several hundred pounds. Small amounts of sphalerite are occasionally encountered.

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* Superintendent, St. Louis Smelting & Refining Co.

[1] Report of the Ontario Bureau of Mines.

The workings at the east and west ends of the vein are about 1000 ft. apart and the vein width frequently reaches 10 ft. Although workings have not been carried to any great depth, sufficient development work has been done to disclose a large orebody.

Early in 1916 a small concentrating plant was placed in operation. Owing to the coarseness of the galena grains and the highly crystalline character of the calcite, a high-grade product is obtained with a low percentage of lead in the tailings.

The following is an average analysis of the concentrate, which consists of about 50 per cent. jig product and 50 per cent. table concentrates:

Ag Ounces	Pb	Insoluble	Fe	CaO	Zn	S
	Per Cent.					
1.14	79.0	0.60	1.20	0.80	2.00	14.40

On account of the absence of lead smelting facilities in Eastern Canada, it was decided, early in 1916, to build a smelter on the property as quickly

Fig. 1.

as possible as the concentrates were piling up and the demand for lead was great. The writer was called upon to furnish plans for the work and decided that a mechanical hearth plant would be the only solution, from the standpoint of both first cost of installation and operating expense. The concentrates being high-grade and rather coarse, they are an ideal product for hearth treatment.

Active construction was begun in July and although considerably hampered by lack of building material, the first hearth was placed in operation on Oct. 9, 1916. Since that date the hearth has produced about 15 tons of lead daily. From 2 to 3 per cent. of fine crushed limestone is mixed with the galena as fed to the hearth and about 3 per cent. of coke breeze has been found sufficient to maintain the correct smelting temperature.

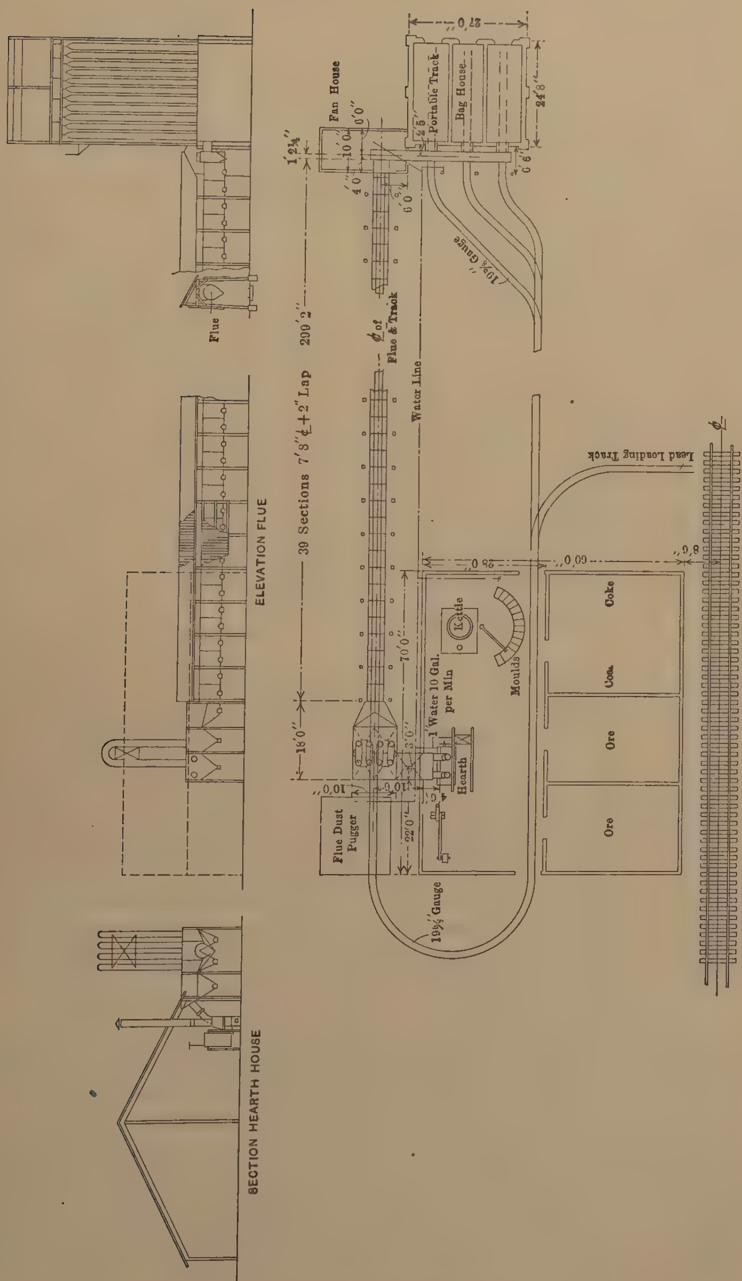
SECTION HEARTH HOUSE
ELEVATION FLUE
Flue
Fan House
Portable Track
Bag House
39 Sections 7'8" ℄ +2" Lap
299'2"
Flue & Track
Water Line
19¾" Gauge
Lead Loading Track
Kettle
Moulds
1" Water 10 Gal. per Min
Hearth
Flue Dust Pugger
Coke
Coal
Ore
Ore
19¾" Gauge

FIG. 2.

Fig. 1 and 2 show the general arrangement of this simple but effective plant. The main hearth building contains the latest type of 8-ft. hearth, drossing and molding kettle and ore-, coke- and coal-storage bins. In order to secure a quick drop in flue temperatures and separation of the coarse particles of dust, four goose-necks in parallel are placed immediately back of the hearth. This sudden lowering of the temperature makes possible a short flue, 300 ft. long, balloon-shaped with clean-out hoppers every 4 ft. This flue is housed to equalize the temperature in summer and winter and in connection with a No. 10 Sirocco exhaust fan delivers about 8,000 cu. ft. of dust-laden gas per minute to the bag house. Below the thimble floor the bag house is divided into three compartments each connected to the damper flue by gas-tight dampers, thus any compartment can be cleaned without interrupting the operations. Above the thimble floor are 99 cotton bags, 18 in. in diameter and 30 ft. long. These bags are shaken from the outside by the efficient Murray bumping system. The temperature of the gases in the bag house does not exceed 180° F.

An industrial track runs under the goose-neck hoppers and flue system and by means of a closed dust buggy the dust may be conveyed to the pugging room without mechanical loss or endangering the health of the workmen. This dust is pugged with a small quantity of burnt lime and together with the burnt bag-house product is bedded with the concentrates.

A total of about 15 hp. is used in the plant and as cheap electric current is obtainable in this district a power plant was not required.

The hearth is worked by two men each on the three 8-hr. shifts, each shift producing about 100 pigs of 100 lb. each. The work is paid for according to the weight of pig lead made, 6 c. per 100 lb. being paid the charger and 5 c. to the helper. This is a direct labor cost of $2.20 per ton of lead produced. A total of 14 men per 24 hr., including the man in charge, is required to handle all the smelter operations and when the low labor, fuel and power items are considered it can readily be seen that the total cost per ton of lead produced is low and could not be approached by any other process.

As the smelting is paid for according to the metallic lead produced, the only cost in connection with the fume and dust is that of transporting it back to the hearth bins, which is a small item.

The lead pigs from the hearth are melted down in a kettle, skimmed clean of dross and molded into trade bars. The dross is returned to the hearth.

The following figures show the average metallurgical outcome per 24 hr.:

	Pounds
Ore charged (dry weight)	51,100
Lead contents	40,369
Pig lead made	29,873
Gray slag made	11,815
Dust and fume (73 per cent. lead)	7,742
	Per Cent.
Coke breeze used	3

The percentage of the total lead in products is approximately as follows:

	Per Cent.
Pig lead	74
Gray slag	12
Dust and fume	14

Analysis of Gray Slag

Pb	Insoluble	FeO	CaO	S
Per Cent.				
41.0	11.2	13.0	11.6	1.7

Analysis of Lead

Ag	Pb	Cu
Ounces	Per Cent.	
5.06	99.95	0.03

As the amount of gray slag produced is small, it is saved up and periodically shipped to a distant blast-furnace plant, the low sulphur contents and its self-fluxing qualities making it a desirable blast-furnace material.

With the installation of larger concentrating facilities, in the near future, additional hearths will be erected, and at that time provision will be made to smelt the gray slag at the plant.

The lead loss in this plant will not exceed 1½ per cent. and will be principally confined to the blast-furnace loss in smelting the gray slag. The sanitary conditions of the furnace room are perfect and no plumbism is likely to occur in that department.

Several other mines in this district are developing well, and no doubt in time southeastern Ontario will occupy a creditable position among lead producers.

As Canada produces only about 60 per cent. of her lead requirements, the government is encouraging the production of refined metal from Canadian ores by placing a bounty thereon and this will no doubt stimulate the exploration and development of this field as will also the close proximity of a lead smelter which will eliminate the heavy freight charges of the past.

The Tredinnick-Pattinson Process

BY WILLIAM E. NEWNAM,* B. S., COLLINSVILLE, ILL.

(St. Louis Meeting, October, 1917)

WHEN Hugh Lee Pattinson discovered, in 1829, that the crystals formed during the slow cooling of molten lead were poorer, and the remaining liquid richer in silver, than the original lead, an important step was made in the metallurgy of this metal. Being the first process applicable to the desilverization of low-grade lead bullion, it soon found its way into all European countries having such a product to treat, thereby effecting the saving of many ounces of silver annually which previously had been thrown away. The old hand process was expensive, as viewed from the standpoint of more modern methods; the labor was great and the tonnage was small, but on the other hand the results were fairly satisfactory.

This process flourished for some time after the introduction of the cheaper Parkes process in 1852 and it is still in use in a few European smelteries.

Only one notable improvement was made on the original process, that of Luce and Rozan at Marseilles, France. It consisted in the application of steam for stirring and the tapping of one-third of the liquid lead, through perforated plates covering the tap holes flush with the bottom of the kettle, thus leaving two-thirds in the form of crystals in the original kettle. This modification considerably lowered the operating costs as it permitted of much larger kettles, an increased tonnage and eliminated much of the slow, hard labor of the hand process. The Luce and Rozan process has persisted in its original form to the present day.

Stephen Tredinnick, English born, spent most of his years around the Luce and Rozan plants at Marseilles and elsewhere in Europe, coming later to Eureka, Nev., in 1878, to operate the Luce and Rozan plant at that point.

Mr. Tredinnick was a competent Luce and Rozan operator and being devoted to the process he firmly believed that with modifications the costs could be so lowered as eventually to supplant the Parkes process.

Out of many schemes he decided that this could be accomplished by placing the Luce and Rozan kettles upon hydraulic rams so that each

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kettle could be raised or lowered at will. He accordingly took out patents to cover this idea.

As Mr. Tredinnick's age and infirmities incapacitated him for active service, the writer was commissioned by a large lead refinery to design, erect and place in operation a plant of the Tredinnick-Pattinson type.

Working on refined desilverized lead from the Parkes process, carrying variable percentages of bismuth, two objects inspired the installation: First, to produce a large tonnage of refined lead low enough in bismuth (0.05 of 1 per cent.) for corroding purposes; and, second, a small tonnage in which the major portion of the bismuth would be concentrated so that it might be recovered by further treatment in the Betts electrolytic process, all of which constituted a step in metallurgy hitherto untried.

After considerable experimental work, it became evident that in order to produce a large tonnage of corroding lead at one end of the plant and at the other end a small tonnage of anode lead carrying 1.0 per cent. of bismuth from a refined lead supply carrying 0.33 per cent. bismuth, an 11-kettle plant would be required with the charge-lead going into the eighth kettle from the corroding end.

As the operation of such a plant would resemble a train of gear wheels, inasmuch as the breaking down of one unit would stop the entire plant, it was necessary to make decided changes in the Luce and Rozan equipment so that all parts from the kettle down could be quickly replaced.

In the Luce and Rozan operations the crystallizing of the kettle required about 1 hr., and the melting of the 30 tons of crystals resulting from the operation required about 2 hr., thus 3 hr. were required to an operation.

It was evident that such speed would be fatal to our requirements and that it would be necessary to crystallize and tap in about 30 min. and melt 42 tons of crystals in about 45 min. This was eventually accomplished in the first instance by increasing the steam pressure from 45 to 110 lb., thus permitting the more copious use of water for cooling down, and in the second instance by an efficient application of fuel-oil heating. The increase in steam pressure necessitated a different and much stronger type of steam valve and a secure anchorage for the baffle plate at the bottom of the kettle.

As each kettle would be raised and lowered 7 ft. at frequent intervals, the air, water and oil connections would, of necessity, be flexible rubber and metallic hose and the steam, fume, and smoke connections of a telescoping type.

Thus it will be seen that this plant necessitated a considerable departure from the Luce and Rozan as regards equipment and methods of operation, as the liquid lead would be tapped directly to the next kettle to be operated, thus avoiding the use of auxiliary melting pans.

In this paper, I will omit mention of the mechanical difficulties en-

countered, although they were numerous, describing only the strong and simple final construction.

Equipment

The completed 11-kettle plant contained the following:

Eleven stands of Pattinsonizing kettles having a working capacity of 63 tons each and placed in a line at 12 ft. centers;

One 180-ton storage furnace receiving refined lead direct from the Parkes process refining furnace;

One 150-ton molding furnace for the corroding lead coming from No. 1 kettle in frequent taps of 21 tons each;

One 42-ton anode molding kettle for the high-bismuth anode lead coming from No. 11 kettle in occasional taps of 21 tons each;

One 150-ton dross-reducing furnace;

One oblong spout kettle, filled with hot lead, for heating the tapping spouts, the spouts being kept therein until needed;

One 18-ton ladle for charging molten lead from the holding furnace into the charge kettle, which is usually No. 8.

All of the above equipment was covered by a 25-ton traveling crane which was used primarily for charging molten lead to the process, and secondarily for the replacement of broken kettles and other heavy equipment. (A defective kettle can be removed and a new one installed in 20 min.)

Each stand consists of a 63-ton kettle and combustion chamber inclosed in a brick-lined steel casing having an I-beam base, all superimposed upon a hydraulic ram 26 in. in diameter with a 7-ft. stroke. These rams are connected with a pump and accumulator and operate under a pressure of 500 lb. per square inch. Thus any kettle can be raised and lowered at will, by means of a Critchlow valve, and its contents tapped to either adjacent kettle (Fig. 1).

Each kettle is supplied with a special steam valve which enters the side close to the bottom and terminates under the center of the heavy baffle plate which is 45 in. in diameter, being perforated so as to secure as equal distribution of the steam as possible throughout the lead mass (Fig. 2). This steam valve is so constructed that it can be replaced in a few minutes. The steam is carried to the valve through a telescope pipe connection at a pressure of 110 lb.

Each kettle is provided with a truncated cone cover having four working doors. Four inches above the top of the cover is placed a circular $\frac{3}{4}$-in. water pipe, 24 in. in diameter, having eight equally spaced $\frac{3}{32}$-in. holes on the under side for introducing water into the kettle by means of funnel cups which pass through the cover. The water connection to the stand is a $\frac{3}{4}$-in. hose 9 ft. long. The water and steam are both controlled by the operator on the second, or kettle floor.

Each kettle cover has a 13-in. round opening and collar in the center connected by a telescope pipe to a sheet-metal flue. This flue is provided with an 8-ft. exhaust fan running at 300 r.p.m. whereby a strong draft is created to remove the waste steam and lead oxide dust from the top of the kettle.

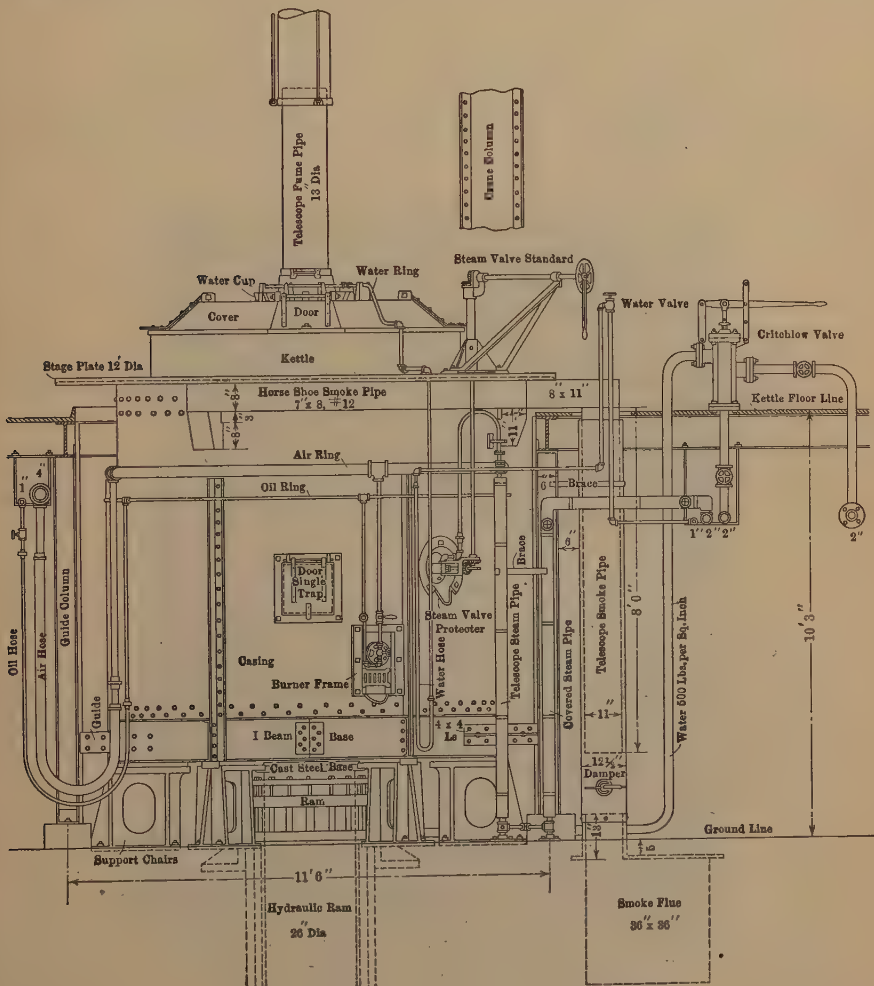


FIG. 1.—KETTLE STAND AND CONNECTIONS.

Heat is applied by three fuel-oil atomizers equally spaced around the casing and operated on an oil pressure of 40 lb. per square inch and a blast pressure of 24 oz. The oil is connected to the stand by a ½-in. metallic hose 9 ft. long and the air by a flexible 2-in. suction hose.

The waste gases from the oil combustion are drawn off at three equally spaced openings by a horseshoe pipe around the top of the casing and

provided with a telescope pipe connection to an underground smoke flue. This smoke flue is provided with a 48-in. fan running at 440 r.p.m. and serves to draw the flame well up around the sides of the kettle, thus providing equal heat to its entire surface.

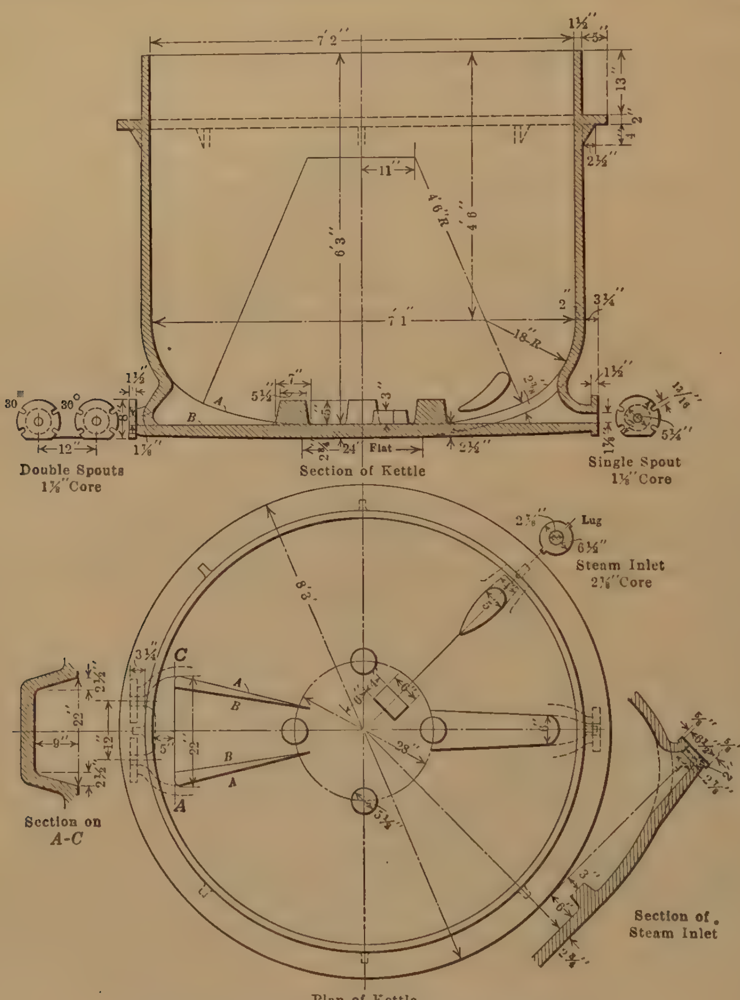


FIG. 2.—KETTLE FOR PATTINSON PLANT.

Method of Operating

In the hand Pattinsonizing, it was possible, with very low-grade bullion, to dip out as much as seven-eighths of the contents of the kettle as crystals, leaving one-eighth remaining as liquid, although the more common practice was to remove two-thirds as crystals, leaving one-third liquid behind. In the Luce and Rozan process, as the crystals remain

in the kettle and the liquid is tapped out through screens which hold the crystals back, it is not practicable to thicken to more than two-thirds crystals and effect a good separation. Therefore, there was no choice but to follow the custom of thirds and make 42 tons of crystals and 21 tons of liquid at each operation.

Order of Operations

Assuming that the supply of lead is fairly constant as to its bismuth contents, the periodical charge of lead to the plant will always be made

FIG. 3.—CHARGING NO. 8 KETTLE.

in the same kettle. In the present instance this would be the eighth kettle from the corroding end of the plant (Fig. 3).

In order to place the plant in operation, each kettle must contain a certain tonnage of lead with a fixed percentage of bismuth depending upon the position of the kettle in the string. That the process may be continuous, the kettles must be crystallized according to some definite system, as otherwise the plant would soon be in a serious muddle which would require considerable time to straighten out. Two such systems

are possible, each permitting of numerous variations, and have been designated the Tredinnick system and the Newnam system (see Fig. 4 and 5).

If the plant is correctly charged to begin operations, according to either system, it is evident that the first operation will change the distribution of the lead in the plant, as will each additional operation, and a

Condition of kettles at beginning of cycle.

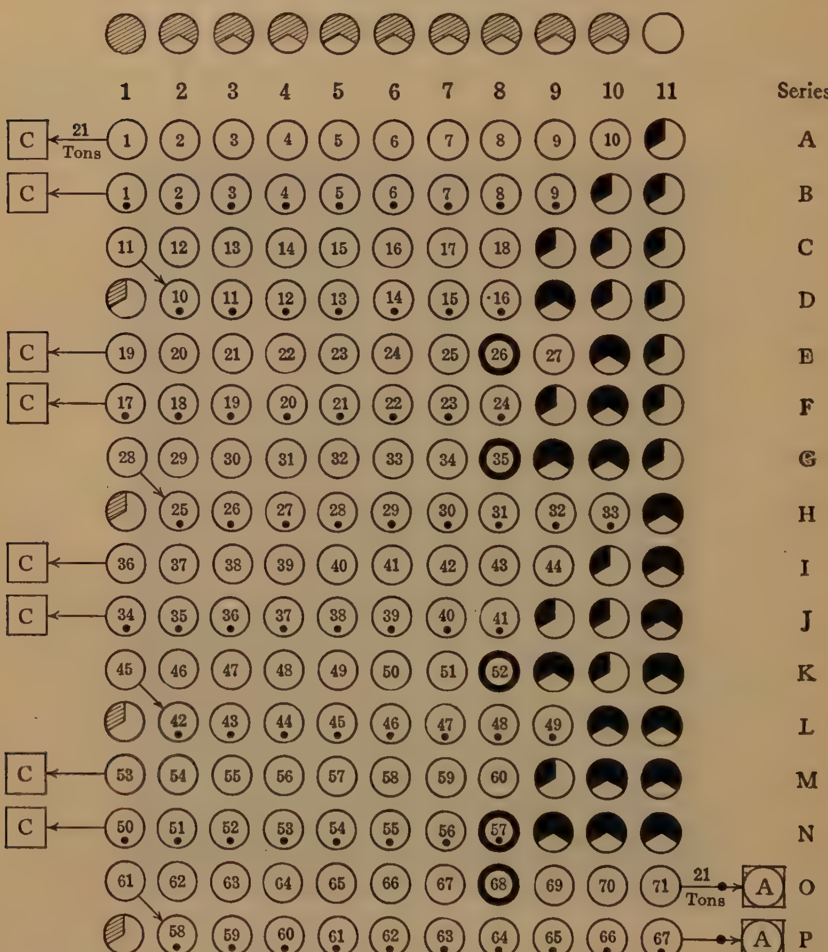


FIG. 4.—THE NEWNAM SYSTEM.

fixed number of operations must be performed before the plant will work back to its original state.

This period of operations is termed a cycle, and, charging in No. 8 kettle, two types of cycles are possible. The first, or single cycle, is that in which one tap of anode lead is made during the period, and the second, or double cycle, that in which two taps of anode lead are made before the plant returns to its original condition.

When the plant is in such a condition that the 9th, 10th and 11th kettles are not operated, the work on Nos. 1 to 8 is called the short string, whereas when these upper kettles come into play the long string is operated.

With the short string, 40 to 50 operations are possible in 24 hr., while, with the long string 70 to 80 operations can be made in that time.

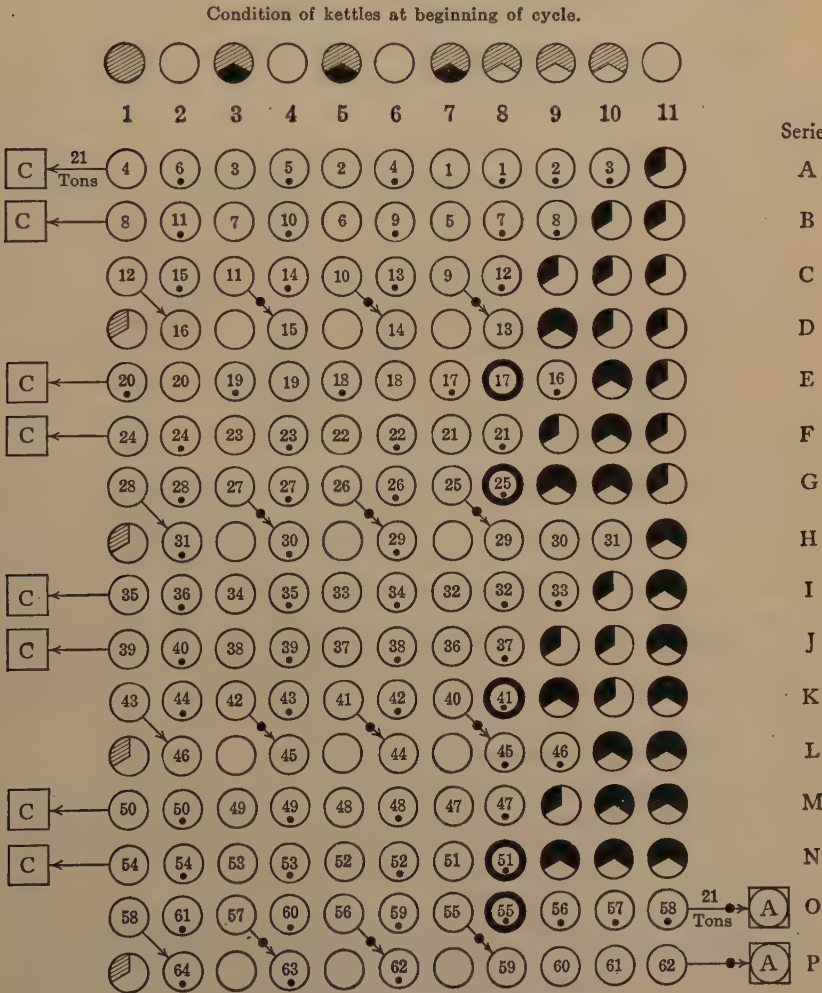


FIG. 5.—THE TREDINNICK SYSTEM.

As very few persons on this continent are acquainted with Pattinsonizing operations, a detailed description thereof will be necessary to make the accompanying cycle diagrams intelligible. I will first describe the Newnam system (Fig. 4) as that is the easiest to follow, using the following conventional figures to illustrate the process.

Empty kettle.

Kettle containing 21 tons melted crystals.

Kettle containing 42 tons melted crystals.

Kettle containing 21 tons enriched liquid.

Kettle containing 42 tons enriched liquid.

Full kettle made up of 42 tons of melted crystals and 21 tons of enriched liquid.

Kettle containing charge of 42 tons.

Showing direction of the movement of melted crystals.

Showing direction of the movement of the enriched liquid.

C Corroding lead-molding furnace.

A Anode or enriched lead-molding furnace.

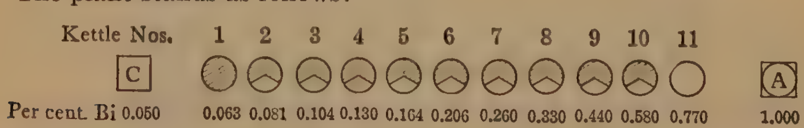
1 First crew.

1 Second crew.

The following diagram will represent an 11-kettle plant as it stands at the beginning of a double cycle, operating on the Newnam system.

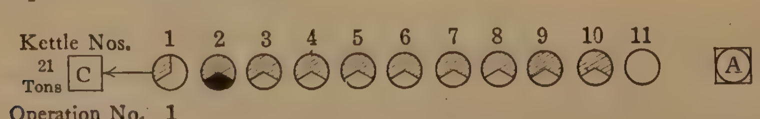
No. 1 kettle is full (63 tons) of melted crystals ready to crystallize. Kettles Nos. 2 to 10 inclusive are each two-thirds full (42 tons) of melted crystals, and No. 11 kettle is empty. The percentage of bismuth in each kettle is indicated below the kettles and represents the approximate grade at which they should be maintained. All charges to be placed in No. 8 kettle.

The plant stands as follows:



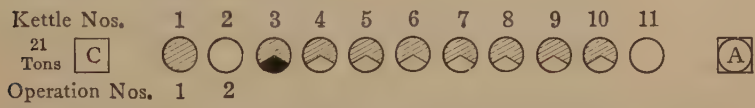
No. 1 kettle is now crystallized, the kettle elevated and 21 tons of liquid are tapped into No. 2 kettle, making it a full kettle. The 42 tons of crystals in No. 1 kettle are melted and 21 tons are tapped to the corroding lead-molding furnace, thus completing operation No. 1.

The plant stands as follows:



No. 2 kettle being full, it is now crystallized and 21 tons of liquid tapped into No. 3 kettle, filling it. As soon as the 42 tons of crystals in No. 2 are melted the kettle is again elevated and contents tapped to No. 1 kettle, filling the latter and completing operation No. 2.

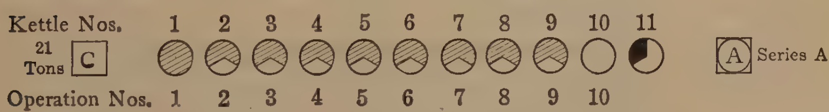
The plant stands as follows:



Kettles Nos. 1 and 3 now being full, it is evident that these two kettles may be operated at the same time, requiring two crews. As the first crew progresses up the string, the second crew would follow behind and as close up as melting of the crystals would permit. In actual practice, usually two or three kettles intervene between the two crews. In order to avoid confusion in the cycle diagram, the work of the two crews is shown in alternating lines; thus, the first crew works out the string marked Series A after which the second crew works out the Series B although, as stated, in actual practice they follow as close behind one another as plant conditions will permit.

Returning to the last diagram, the first crew will successively crystallize the kettles 3 to 10 inclusive and the operation on No. 10 will place 21 tons of liquid into the empty kettle No. 11. In each case the melted crystals are tapped toward the corroding end.

The plant will stand as follows:

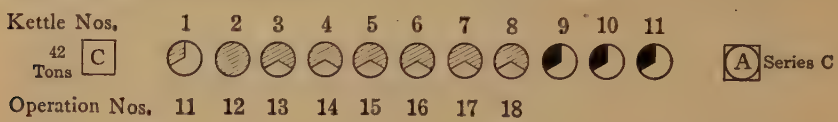


The first crew has thus made 10 operations on the long string, Series A. The second crew now begins on No. 1 kettle and, in like manner, successively operates the kettles Nos. 1 to 9 inclusive and the operation on No. 9 will place 21 tons of liquid into the empty kettle No. 10.

The plant will stand as follows:

The second crew has thus made nine operations on the long string Series B, and the first crew now returns to No. 1 kettle to work out its second string, or Series C. Owing to the necessity of correcting the grade of the kettles, as explained later on, a variation of the procedure now takes place in Series C. During this series none of the melted crystals

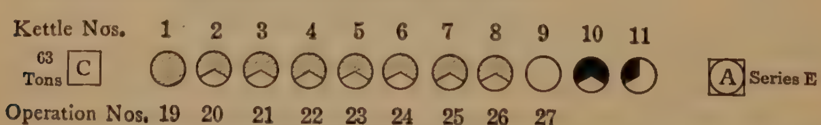
are tapped toward the corroding end but are left in their original kettle, with the exception of the crystals in No. 1, 21 tons of which instead of being tapped to the corroding furnace are tapped into No. 2 kettle, and the plant stands as follows:



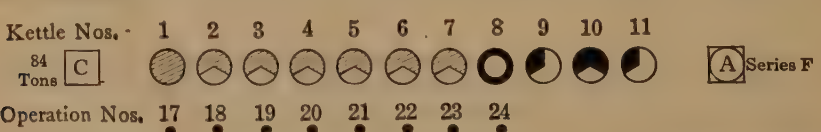
The second crew returning to the corroding end to work out its short string, Series D, is obliged to pass over kettle No. 1 and begin operations on the full kettle No. 2, thus proceeding straight up the line with the operation on No. 8 kettle placing a second tap of 21 tons of liquid in No. 9 kettle. As all crystals on this series are tapped back as usual, and as this will leave the charge kettle No. 8 empty, it is in order to give it a 42-ton charge.

The plant stands as follows:

No. 8 having been charged, the Series E will now be run out by the first crew, and it will be observed that the operation of No. 8 will fill kettle No. 9 which must then be worked, placing an additional 21 tons of liquid in No. 10, or 42 tons in all. Therefore, at the close of Series E by the first crew the plant will stand as follows:



The second crew now begins its 17th consecutive operation with kettle No. 1 of the Series F, at the conclusion of which the plant will stand as follows:

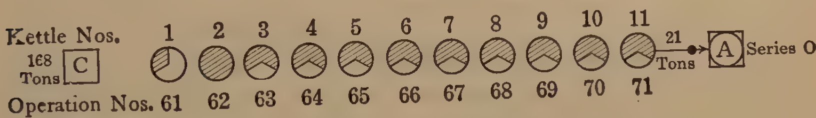


As No. 9 was not operated in Series F, there were no return crystals to No. 8; therefore, the second charge of 42 tons was introduced into that kettle.

On Series G the correction for grade is again made by omitting the corroding tap and running 21 tons of No. 1 crystals into No. 2 kettle.

The plant is steadily worked according to the above plan until the end of Series N, when the plant stands as follows:

As the 9th, 10th and 11th kettles now contain 42 tons each, it is evident that the operations on Series O will include all kettles from 1 to 11 inclusive and that 21 tons of anode lead will be tapped from No. 11 kettle into the anode molding kettle. As the correction of grade occurs at this time, none of the crystals produced on Series O will be tapped back, and at its termination the plant stands as follows:



The first crew will now run out the last string of the cycle, Series P, operating all the kettles from No. 2 to 11 inclusive and placing a second tap of 21 tons of anode lead into the anode kettle. All the crystals having been tapped down the line, at the termination of this series the plant returns to its original condition, as first shown, and a new cycle begins.

During this cycle the following operations were made:

First crew	71
Second crew	67
Total	138

As four corrections for grade were made during this period, $\frac{138}{4}$ or $34\frac{1}{2}$ represents the number of operations made for each correction in grade. On the cycle shown, corroding lead was made in sets of two consecutive taps between corrections; now should these taps be made in groups of one, three or four, an entirely different cycle would result, necessitating a separate diagram for each instance.

During the period, the ingoing and outgoing lead was as follows:

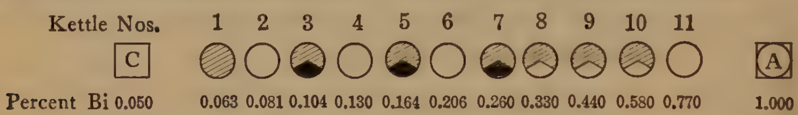
	Tons
Corroding lead, eight taps of 21 tons each	168
Anode lead, two taps of 21 tons each	42
Charge lead, five charges of 42 tons each	210

Dividing these tonnages by the total number of operations, 138, the tons of lead charged and produced may be expressed as follows:

Tons per Operation

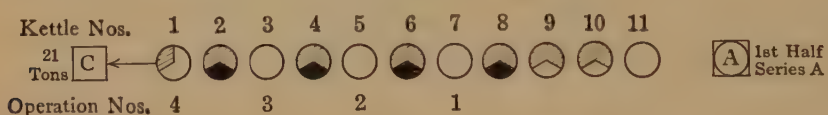
Charge	Corroding	Anode
1.522	1.218	0.304

In order to handle the same tonnage and obtain the same results by the Tredinnick system, an entirely different procedure is necessary. Whereas in the Newnam system each series is worked out by one crew, in the Tredinnick system each series is split up between the two crews and in order to understand its workings the series must be shown in two parts. The condition of the kettles at the beginning of a double cycle is entirely different, and the plant, ready to operate, stands as follows:

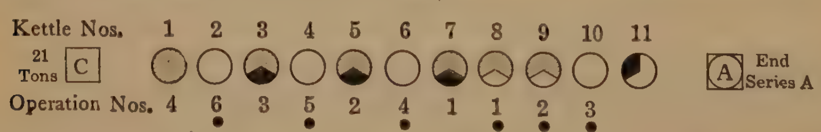


Observe that there are four full kettles, Nos. 1, 3, 5 and 7, and that Nos. 8, 9 and 10 contain 42 tons each while Nos. 2, 4, 6 and 11 are empty.

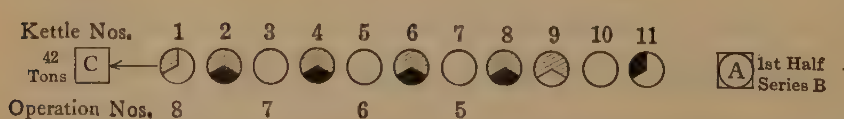
The first crew operates in the following order: No. 7, No. 5, No. 3, No. 1. All resulting crystals are tapped down the line, 21 tons of No. 1 crystals being tapped to the corroding furnace. At the end of the first half of Series A, the plant stands as follows:



The second crew will now work successively Nos. 8, 9 and 10, placing 21 tons of liquid in No. 11. They will then operate in order Nos. 6, 4 and 2 and the plant stands as follows:



The first crew now returns to No. 7 and works successively Nos. 7, 5, 3 and 1, 21 tons of crystals being tapped to the corroding furnace from No. 1 kettle. At the end of the first half of Series B the plant stands as follows:



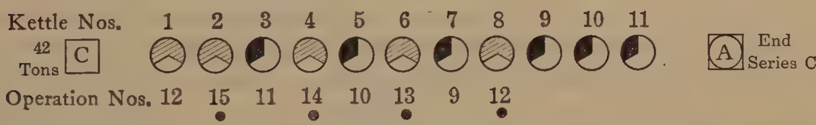
The second crew will complete Series B by operating kettles 8 and 9,

placing 21 tons of liquid in the empty kettle No. 10 and then operating in order Nos. 6, 4 and 2.

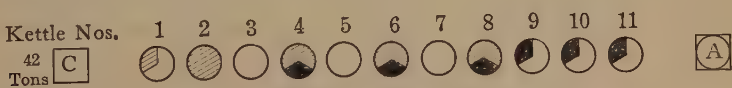
The plant stands as follows:

Series C is now run out by the first crew taking the kettles 7, 5, 3 and 1 and then the second crew the kettles 8, 6, 4 and 2. As a result of this series, 21 tons of liquid are placed in the empty No. 9. Since preparation for correction of grade begins at this point, the tap to the corroding furnace is omitted, and the crystals in 8, 6, 4 and 2 are not tapped back.

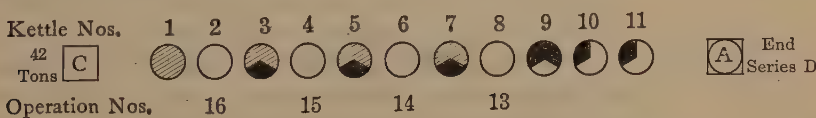
The plant stands as follows:



Now the 21 tons of liquid in Nos. 7, 5 and 3 are tapped respectively to Nos. 8, 6 and 4 and 21 tons of crystals in No. 1 are tapped back into No. 2, and the plant stands as follows:



The first crew now operates Nos. 8, 6, 4 and 2 and the resulting crystals being tapped back as usual, Series D is terminated and the plant stands as follows:



As No. 8 kettle is now empty it must receive a charge of 42 tons.

With slight variations the foregoing procedure is repeated until the plant returns to its original condition, which will take place at the 64th operation of the second crew.

The above double cycle produces the following data:

Operations

First crew	62
Second crew	64
Total	126

Tons per Operation

Charge	Corroding	Anode
1.666	1.333	0.333

A comparison of the two systems may be summed up as follows from the two cycles shown:

	Tredinnick	Newnam
Maximum lead in process	399	483
Minimum lead in process	336	399
Number of operations per cycle	126	138
Tons charged per operation	1.666	1.522
Tons corroding lead produced per operation	1.333	1.218
Tons anode lead produced per operation	0.333	0.304
Total tons charged	210	210
Total tons produced	210	210

From an inspection of the above figures, it would seem that the Tredinnick system has decided advantages in every respect. But with the material on which we were experimenting we found it necessary periodically to make a correction for grade. We also found that the necessity of skimming kettles one-third full of lead was a laborious task, and if not properly done the bottom of the kettles soon took on a heavy coating of litharge most prejudicial to the heating and to the life of the kettles.

In overcoming these disadvantages, we worked out the Newnam system, and found that with our particular class of work its relative simplicity more than offset the advantage of the Tredinnick system. Other plants might have a different experience and might find the Tredinnick system superior.

Grade of Kettles

Each kettle in the string has its own particular grade, as determined by experience, and it must be maintained at that grade to insure the production of corroding lead from No. 1 kettle crystals.

The grade of a kettle is governed by the bismuth contents of the mixture of one-third liquid and two-thirds crystals going to fill that particular kettle. It is manifest that this should be the same at all times, but this is far from being the case, and thereby serious complications arise.

As the liquids all travel toward the anode end of the plant and all the crystals travel toward the corroding end, it is evident that if one kettle becomes overgrade it will communicate that effect to its neighbors and a correction of some kind will be necessary.

Owing to the natural percentage of enrichment of bismuth in the liquid lead by crystallizing, there is a decided tendency for the grades of all kettles between the corroding end and the charge kettle to increase steadily. This insidious tendency has been termed the "crawl."

The following semi-mathematical exposition of this "crawl" will clearly show its cause and effect.

The average percentage of bismuth in the liquid from any operation is about 1.4 times the percentage of bismuth in the kettle charge before crystallizing. Although not strictly true, it has been the custom to call this 40 per cent. enrichment.

The average percentage of bismuth in the crystals is about 0.8 times the percentage of bismuth in the kettle charge before crystallizing. In like manner, this is spoken of as 20 per cent. impoverishment.

For the sake of simplicity, consider the effect of this 40 per cent. enrichment on one kettle only and it will be evident that this applies to any kettle between No. 1 and No. 11, except that the kettle considered, No. 8, has a partial correction made to it every time it receives a charge of new lead, hence the crawl is not so constantly accumulative in No. 8 as it is in any other kettle.

Assume that No. 8 is charged with lead containing 0.300 per cent. Bi.

Crystallize the kettle of 63 tons which will furnish:

No. 7 with 42 tons of crystals carrying 0.240 per cent. Bi.

No. 9 with 21 tons of liquid carrying 0.420 per cent. Bi.

It is legitimate to assume, at this point, in the next series of operations up the line that No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.240 per cent. Bi.

The kettles stand thus:

		No. 7	No. 8	No. 9
I.	Tons..........................	63	Empty	21
	Per cent. Bi....................	0.240		0.420

Now crystallize No. 7, which will furnish:

No. 8 with 21 tons liquid carrying 0.336 per cent. Bi.

No. 6 with 42 tons crystals carrying 0.192 per cent. Bi.

Charge No. 8 with 42 tons of lead at 0.300 per cent. Bi; the average contents of No. 8 will then be 63 tons carrying 0.312 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.437 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.250 per cent. Bi.

Observe that the liquid from No. 7 to No. 8 is 0.336 per cent. Bi, or 12 per cent. higher than it should be. This before it has been subjected to the influence of any of the higher kettles. It is therefore correct to say that the liquid from No. 6 to No. 7 will increase 12 per cent. every time the operations proceed up the line. Hence, as No. 7 is approached the next time, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.268 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
II.	Tons..........................	63	Empty	42
	Per cent. Bi....................	0.256		0.428

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid carrying 0.358 per cent. Bi.

Charging No. 8 kettle with 42 tons lead at 0.300 per cent. Bi; it will contain 63 tons of lead carrying 0.319 per cent. Bi.

Crystallize No. 8, which will furnish:

No. 9 with 21 tons liquid carrying 0.446 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.225 per cent. Bi.

No. 9 now contains 63 tons of lead at 0.434 per cent. Bi, which crystallized will furnish:

No. 8 with 42 tons crystals carrying 0.347 per cent. Bi.

As No. 7 is approached the next time, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.300 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
III.	Tons........................	63	42	Empty
	Per cent. Bi..................	0.270	0.347	

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid at 0.378 per cent. Bi. It will now contain 63 tons carrying 0.357 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.500 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.286 per cent. Bi.

As No. 7 is again approached, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.336 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
IV.	Tons........................	63	Empty	21
	Per cent. Bi..................	0.302		0.500

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid carrying 0.422 per cent. Bi.

Charge No. 8 with 42 tons of lead at 0.300 per cent. Bi, and it will contain 63 tons carrying 0.341 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.477 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.273 per cent. Bi.

As No. 7 is again approached, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.376 per cent. Bi.

The kettles will stand:

		No. 7	No. 8	No. 9
V.	Tons........................	63	Empty	42
	Per cent. Bi..................	0.341		0.488

To Recapitulate

	No. 7	No. 8	No. 9
I.	0.240	0.300	0.420
II.	0.256	0.312	0.437
III.	0.270	0.319	0.446
IV.	0.302	0.357	0.500
V.	0.341	0.341	0.477

It can readily be seen that this is an accumulative "crawl" and while it is aggravated each time No. 9 is brought into play and in like manner more so when No. 10 and No. 11 are drawn upon, yet it would exist were there no kettles above No. 8. Furthermore, the percentage rate of "crawl" of No. 2 or any other kettle will be the same as that of No. 7.

It is evident that the percentage of enrichment does not fit the method of crystallizing in "thirds" as it does in the case of the enrichment of silver. Compare the kettle grades as they are maintained in the case of bismuth and silver.

Kettle No.		1	2	3	4	5	6	7	8	9	10	11	
Bi per cent.	0.050	0.063	0.081	0.104	0.130	0.164	0.206	0.260	0.330	0.440	0.580	0.770	1.000
Ag oz.	¼	½	1	2	4	7	12	20	35	60	100	180	300

It has been shown that with "40 per cent." enrichment and consequent "20 per cent." impoverishment, if the impoverished lead from any other kettle be crystallized it will yield a liquid 12 per cent. higher than the grade of the original kettle from which it came. This does not represent the rate of "crawl" proportional to it on account of the charges introduced into No. 8 of proper grade. For the sake of comparison, however, the following table will show what the relative per cent. of "crawl" is for the different percentages of enrichment.

Enrichment, Per Cent.	Faulty Increase, Per Cent.
10	4
20	8
30	10
40	12
50	12.3
60	12
70	10
80	8
90	4
100	0

Therefore, the only percentage of enrichment with which there would be no "crawl" would be 100 per cent., which is nearly the case in that of silver, thus in the old Pattinson operations for silver the "crawl" was insignificant.

Correction of Grade

In order to offset the "crawl," it is necessary to resort to some mechanical method of correction (at least up to our present knowledge), such as tapping a kettle overgrade to the kettle of next higher grade or to increase the number of crystallizations upon a given tonnage of lead.

This is accomplished by a so-called "jump"[1] whereby a combination of these two schemes is effected but which results in greatly lowering the tonnage of the plant with a corresponding increase in the cost per ton charged.

The "jump" is made by periodically tapping 21 tons of melted crystals from No. 1 kettle onto the 42 tons of crystals in No. 2 kettle and then going straight up the line, without any of the crystals in advance being tapped back. Thus the crystals receive an extra operation and one-third liquid is pushed by successive stages up the line.

The method of "jumping" is shown by the cycle diagrams under "Order of Operations."

Table 1 gives the relative tonnages according to the frequency of the "jump" and the kettle receiving the charge.

TABLE 1

Charge No.	Jump	Cycle Operations	Tons per Operation Charged	Tons per Operation Corroding	Tons per Operation Anode	Anode Fraction of Tons Charged
6	1–28–⅚	256D*	1.634	1.471	0.163	1/10
7	0	244S†	2.754	2.668	0.086	1/32
7	1–40	120S	1.750	1.575	0.175	1/10
7	1–31–¾	190D	1.547	1.326	0.221	1/7
8	0	131S	2.565	2.405	0.160	1/16
8	1–44	176D	1.670	1.432	0.238	1/7
8	1–34–½	138D	1.520	1.210	0.310	1/5
9	1–36	76D	1.658	1.105	0.553	1/3

* D = double cycle, two taps of anode metal.
† S = single cycle, one tap of anode metal.

Experience has shown the necessity of jumping:

	No.
Once in 28⅚ operations if charging in........................	6
Once in 31¾ operations if charging in........................	7
Once in 34½ operations if charging in........................	8
Once in 36 operations if charging in........................	9

In each of the above four cases, two consecutive taps of corroding lead are made between corrections for grade, the operations in each case being similar to those shown in the cycle diagrams.

The following analytical results shown in Table 2 were made on a

[1] So called on account of passing over No. 1 kettle during the period of correction.

10-kettle plant, each determination being made on a sample taken from a full kettle just before crystallizing.

TABLE 2

No. 1.—Showing the Steady Increase in Grade Without "Jumping"

Corroding "C"	1	2	3	4	5	6	7	8	9	10	Anode "A"
0.061	0.075	0.110	0.170	0.208	0.251	0.315	0.400	0.468	0.675		
0.061	0.073	0.131	0.161	0.211	0.285	0.348	0.401	0.529			
0.067	0.079	0.129	0.169	0.229	0.295	0.358	0.412	0.462			
0.071	0.093	0.138	0.148	0.233	0.276	0.346	0.454	0.498	0.793	0.862	0.878
0.081	0.093	0.138	0.181	0.237	0.287	0.358	0.407	0.562	0.641		
0.076	0.093	0.154	0.197	0.254	0.299	0.331	0.399	0.529			
0.076	0.095	0.148	0.197	0.249	0.299	0.362	0.445	0.482			
0.078	0.102	0.142	0.199	0.257	0.328	0.388	0.486	0.476	0.691		
0.074	0.092	0.146	0.204	0.261	0.345	0.366	0.413	0.548			
0.086	0.104	0.158	0.202	0.262	0.316	0.370	0.463	0.495			
0.088	0.117	0.164	0.219	0.274	0.342	0.405	0.445	0.478			
0.095	0.116	0.168	0.225	0.280	0.353	0.408	0.423	0.571	0.796		

No. 2.—Showing Maintenance of Grade by "Jumping"

Corroding "C"	1	2	3	4	5	6	7	8	9	10	Anode "A"
								C			
0.050	0.063	0.079	0.103	0.136	0.187	0.236	0.316	0.390			
0.048	0.060	0.086	0.112	0.157	0.202	0.265	0.378	0.442	0.521		
0.048	0.055	0.096	0.127	0.180	0.219	0.296	0.367	0.425			
0.045	0.058	0.099	0.136	0.177	0.227	0.285	0.357	0.428			
0.056	0.069	0.108	0.141	0.192	0.248	0.307	0.373	0.422	0.581	0.775	1.045
	0.073	0.110	0.153	0.203	0.254	0.314	0.371	0.475	0.636		
	J	0.081	0.119	0.161	0.212	0.282	0.325	0.409			
0.048	0.058	0.097	0.132	0.174	0.236	0.291	0.359	0.436			
0.049	0.064	0.102	0.137	0.179	0.237	0.294	0.364	0.415			
0.058	0.074	0.109	0.147	0.197	0.245	0.296	0.369	0.417	0.588	0.829	1.103
	0.083	0.111	0.150	0.190	0.253	0.297	0.358	0.436	0.631		
	J	0.080	0.114	0.158	0.218	0.274	0.318	0.373	0.500		
0.053	0.070	0.103	0.136	0.178	0.234	0.282	0.345	0.423			
0.054	0.081	0.105	0.134	0.183	0.229	0.270	0.330	0.353			
	0.066	0.099	0.148	0.197	0.250	0.308	0.329	0.338	0.560	0.830	1.177
	J	0.079	0.124	0.163	0.204	0.275	0.294	0.325	0.430	0.595	0.853
0.046	0.062	0.094									
0.055	0.071										
0.056	0.076										

Description of an Operation

A full kettle (63 tons) skimmed clean and ready to crystallize, should have a temperature not much above the melting point of lead, but the brickwork surrounding the kettle should be hot to prevent a crust freezing

on the inside of the kettle. Correct temperature is an important factor, as lead too hot greatly prolongs the time of crystallizing and produces an excessive amount of dross. In practice, the lead is tested by thrusting a broom handle into the molten metal; if the lead freezes to the handle the temperature is low enough to proceed with the operation. If the lead does not freeze to the handle the kettle must be cooled down by freezing crusts with water and pushing these crusts under the surface with a pole until the proper temperature is secured. As this also consumes time, it is necessary to keep a sharp watch of the temperature.

The kettle being in the proper condition, the operator opens the steam valve slowly until the surface of the lead is violently agitated. (In order to prevent slop the kettles are filled to within 15 in. of the top only.) Water is now cautiously introduced through the eight water cups on the cover, by a valve close to the steam gear wheel. The operator regulates the admission of steam and water so that a maximum amount of water is introduced without causing explosions, the formation of chunks or the slopping of lead through the cover doors.

The water cups occasionally become clogged with lead and have to be freed by a special punching rod in the hands of the barman.

Lead soon freezes in a crust on the upper ring of the kettle and to the cover. Periodically the water is turned off, the cover doors thrown back and the crusts barred down with a 6-ft. steel bar, 1 in. in diameter, having a chisel point. Considerable judgment must be exercised not to allow the crusts to become too thick, as in this case they are difficult to break up with the steam and tend to form chunks. Also, too frequent barring down consumes time, as the water is turned off during that period.

To facilitate barring down, the inside of the cover and the upper ring of the kettle should be as smooth as it is possible to make them.

Soon crystals of lead, from 1/16 to 1/8 in. in diameter, begin to appear in the bath, and from this point on they multiply with ever increasing rapidity, the violent agitation by the steam keeping the crystals from adhering to one another.

When the consistency of two-thirds crystals is reached, the surface of the lead appears as an exceedingly thick mass of boiling crystals. At this stage the water is shut off, the kettle barred down for the last time and the crusts broken up by steam. Steam is now turned off, the kettle elevated and two hot spouts, just out of the spout kettle, are placed on the double lead cocks. These cocks (which are kept hot with charcoal) are opened slowly and the one-third liquid tapped to the adjoining kettle through a screen in the bottom of the kettle covering the double taps. This screen has 96 holes 3/16 in. diameter, 2 in. centers.

As soon as the last crust is barred down, the burners are fired, and by the time the liquid has run out the temperature is rapidly rising.

It is surprising how accurately a good crystallizer can judge the pro-

portion of liquid and crystals in the finished kettle. A good man will seldom be in error over 2 tons and this inequality may be eliminated on a subsequent operation by slightly over- or under-crystallizing as the case may require.

Under the proper conditions, a kettle can be crystallized in 15 min. An average operation, crystallizing and tapping, requires about 30 min.

It will be noted from the cycle diagrams that any delay on one kettle affects the whole plant. Quick crystallizing and quick melting are therefore necessary to speed and the more rapidly these are performed the fewer kettles need intervene between the kettles being operated, thus permitting of more kettles in operation at the same time.

On the short string, two crews are operating simultaneously, whereas on the long string as many as four crews may be operating.

As soon as the crystals are melted the lead is skimmed and tapped to the opposite adjacent kettle through the single lead cock without a screen. The average melting period is about 45 min. This completes an "operation."

The kettles near the corroding end are much more difficult to operate than those near the anode end, also the finished kettle at the anode end appears much thicker. In both instances this is due to the crystals near the anode end being larger than those at the corroding end of the plant.

Dross

Dross has been one of the most objectionable features of a Pattinson plant, both hand and steam. In the early stages of the Tredinnick plant, 21 per cent. of the lead charged was skimmed out as dross. This was not only expensive to handle, reduce and recharge, but it also left the lower kettles short of lead and thereby reduced the tonnage materially.

It was found that by throwing small quantities of fuel oil into the kettle on top of the crystals during the melting period that the melting was greatly hastened and a large percentage of the dross formed during the crystallizing was reduced in the kettle.

Handled in this manner, the reducing furnace operates only 4 or 5 days a month, and the lead removed as dross is from 2 to 3 per cent. of the lead charged.

During the crystallizing, about 50 lb. of litharge, fine as flour, is formed, which is drawn off by the telescope fume pipe and caught in the flue. A very strong draft is necessary to keep this fume from blowing out through the cover doors and it is absolutely essential to the health of the men that none of it be allowed to get into the working atmosphere.

Fuel

The fuel requirements of the plant depend upon the number of operations. An average of 50 gal. of fuel oil is required per operation. Of

this, 40 gal. pass through the burners and 10 gal. are thrown in the top of the kettle over the melting crystals.

Labor

The accompanying labor table is based on a crew sufficient to keep the operations up to the melting and capable of making from 50 to 70 operations each 24 hr., depending on the number of kettles in the operating line or "string." Thus, whereas an 8-kettle plant would mean 50 operations per day, an 11-kettle plant would make 70 operations, the number of kettles in the string depending upon the condition of the kettles above the charge kettle. An inspection of the flow sheet will make this point clear.

As this plant was placed in operation at a time when the 10- and 12-hr.

TABLE 3.—*Labor Table*

Day Shift, No. Men	Rate	Duties	Per Month
1	$2.65	General foreman	$80.00
1	2.40	Burnerman	72.00
1	2.20	Burnerman assistant	66.00
2	2.25	Crystallizers	135.00
2	1.85	Barmen	111.00
1	1.75	Barman in training	52.50
1	2.00	Tapping and cleaning kettles	60.00
1	2.25	Craneman	67.50
2	1.75	Clean-up and general utility	105.00
12			
Lap Shift			
1	$2.40	In charge at lap and noon time	$72.00
1	2.00	Crystallizer in training	60.00
1	1.85	Barman	55.50
2	1.75	Barmen in training	105.00
5			$1,041.50
Night shift		Same as above	$1,041.50
Reducing Furnace			
1	$2.00	Furnaceman, 5 days per month	$10.00
1	1.65	Furnaceman helper, 5 days per month	8.25
3	1.65	Drossmen, 5 days per month	24.75
1		Day superintendent	125.00
1		Night superintendent	87.00
		Total per month	$2,338.00

day was in vogue, and as it was necessary to keep the plant working 24 hr. per day and not disturb the labor conditions in other parts of the works, a lap shift was introduced, four men coming on at 9 o'clock and working until 7 on each shift, thus keeping the plant moving between shifts and during the luncheon hour.

This arrangement was necessary, as temperature control is of the highest importance to secure tonnage. When the plant is moving at its maximum rate the temperatures take care of themselves to a great extent, as kettles ready to operate are not allowed to stand, but if for any reason the plant is held up for an hour or more it is very difficult to keep both the brickwork and the lead at the proper point.

Experiments

The addition of a third substance or metal was tried in many experiments, in a vain endeavor to find a cheap addition agent that would increase the natural enrichment of bismuth in the liquid.

Out of the many substances tried, tellurium was the only one that had a beneficial influence, but on account of its cost and the difficulty of adding it to the lead it was not available.

Arsenic, tin, cadmium and zinc actually decreased the enrichment, the actions of arsenic being especially harmful.

The small quantity of gold and silver in the charge lead is practically all recovered in the anode metal.

The following shows the distribution of copper in the kettle samples taken after the plant had been in operation for a considerable period on refined lead.

Kettle	Cu, Per Cent.
1	0.000075
2	0.000050
3	0.000050
4	0.000250
5	0.000250
6	0.000250
7	0.000250
8	0.000400
9	0.001500
10	0.002800
11	0.003500
Anode	0.009800

The Tredinnick-Pattinson Process as a Desilverizer

Under the heading of "Grades of Kettles," it was shown that the enrichment in the case of silver is nearly 100 per cent. and that a correction of grade would only occasionally be necessary. It was this ideal natural enrichment that made the Pattinson process a success in its day, and although the Tredinnick-Pattinson process has never been used as a

desilverizer, I believe that I am justified, with the figures at hand, in indulging in some speculation as to its possibilities.

As a basis of calculation, I assume that one Tredinnick-Pattinson operation costs $7.50, this including all overhead expense such as insurance, taxes, amortization, royalty, general expense, etc. The average Parkes cost per ton of bullion charged in a moderate-sized plant treating lead of 100 oz. silver, with zinc at a normal figure of 5 c. per pound, is about $4 per ton.

The Tredinnick-Pattinson plant would turn out a high percentage of the lead treated as common lead containing ¼ oz. of silver. This common lead would be suited to the ordinary purposes to which common lead is put but it could not be used as corroding lead in the manufacture of white lead as its copper contents would be prohibitive, as the analytical results given in Table 4 will show.

Table 4

Kettle	Copper, Per Cent.
Bullion charged	0.084
Enriched lead	0.044
12	0.052
11	0.080
10	0.084
9	0.056
8	0.056
7	0.064
6	0.056
5	0.052
4	0.048
3	0.044
2	0.048
1	0.040
Impoverished lead	0.038

The small percentage of rich lead varying from 100 to 500 oz. of silver would be best treated by the Parkes process, therefore a combination of the two processes would be in order.

Disregarding the periodical correction of grade and avoiding fractions, Table 5 will show the approximate distribution of tonnage for varying amounts of silver, the kettle into which the charge is introduced and the number of kettles in the plant.

From Table 5 and the costs given above, it will be seen that 4-oz. bullion charged in No. 4 kettle in a 9-kettle plant will require 4.35 tons of charge for each operation in that plant, and the cost per ton would be $\frac{\$7.50}{4.35} = \$1.72$. Again, 0.07 ton of rich lead would go to the Parkes process for treatment at a cost of $0.28 (0.07 × $4), which would bring the total cost of the combination treatment to $2 per ton of bullion charged.

TABLE 5.—*The Tredinnick-Pattinson Process*

Silver, Ounces per Ton	Charge in Kettle, No.	Kettles in String	Tons per Operation: Charged	Tons per Operation: Desilverized	Tons per Operation: Rich Lead	Ounces Silver Rich Lead	Cycle, No. of Operations
4	4	9	4.35	4.28	0.07	100	309
4	4	10	4.28	4.25	0.03	180	628
7	5	9	3.71	3.60	0.11	100	181
7	5	10	3.61	3.56	0.05	180	372
12	6	9	3.33	3.12	0.21	100	101
12	6	10	3.17	3.07	0.10	180	212
20	7	10	2.90	2.72	0.17	180	116
20	7	11	2.77	2.68	0.09	300	243
35	8	11	2.56	2.40	0.16	300	131
35	8	12	2.45	2.37	0.08	500	274
60	9	12	2.30	2.16	0.14	500	146
100	10	12	2.27	1.99	0.28	500	74

Table 6 shows the probable outcome by applying the foregoing cost figures.

TABLE 6.—*Comparative Costs of Processes*

Silver, Ounces per Ton Charged	Charge in Kettle, No.	Kettles in String	Tredinnick-Pattinson Cost per Ton Charged	Parkes Cost on Rich Lead per Ton Charged	Total Cost Combination Process per Ton Charged	By Parkes Process Alone	In Favor of the Combination Process	Value of Silver Recovered, 60 c. per Ounce
4	4	9	$1.72	$0.28	$2.00	$4.00	$2.00	$2.25*
4	4	10	1.75	0.12	1.87	4.00	2.13	2.25*
7	5	9	2.02	0.44	2.46	4.00	1.54	4.05
7	5	10	2.08	0.20	2.28	4.00	1.72	4.05
12	6	9	2.25	0.84	3.09	4.00	0.91	7.05
12	6	10	2.37	0.40	2.77	4.00	1.23	7.05
20	7	10	2.58	0.72	3.30	4.00	0.70	11.85
20	7	11	2.71	0.36	3.07	4.00	0.93	11.85
35	8	11	2.93	0.64	3.57	4.00	0.43	20.85
35	8	12	3.06	0.32	3.38	4.00	0.62	20.85
60	9	12	3.26	0.56	3.82	4.00	0.18	35.85
100	10	12	3.30	1.12	4.42	4.00	0.42	59.85†

* Loss by Parkes.
† Loss by combination.

The figures in Table 6 indicate the possibilities for the combination process with bullions assaying between 4 and 60 oz. of silver per ton, but the price paid for fuel oil or gas and the labor rate would have a very vital bearing on the results.

A plant such as described should treat from 150 tons to 200 tons of bullion per 24 hr., and the construction cost of a completely equipped 12-kettle plant, with material at its normal figure, would be approximately $85,000.

The Metallurgy of Lead Ores in the Lower Mississippi Valley

BY HERMAN GARLICHS, E. M., ST. LOUIS, MO.

(St. Louis Meeting, October, 1917)

THE development of the extensive Southeast Missouri deposits greatly preceded that of the Iowa and Wisconsin deposits. It began about 1720 at Mine La Motte and other localities, and has continued uninterruptedly to the present time. It is estimated that this district produced about 184,000 tons of lead in the year 1916, having a value of approximately $25,000,000. This is a new high record.

The Southwest Missouri orebodies were hardly known before 1845 and were not extensively developed until 1870. Lead production in Southwest Missouri for 1916 was 38,788 tons. Oklahoma produced 14,399 tons and Kansas 2,345 tons, making a total for Southwest Missouri, and the adjoining districts in Oklahoma and Kansas, of 55,532 tons of lead.

The first metallurgists in Missouri were the Indian and the hunter. The early settler learned to procure his bullets either by melting ore in his camp fire or by throwing pieces of galena on an old stump and depending on the usual roasting and reaction method to obtain the metal. A little more refined method was to arrange two flat stones in the form of a V. Wood and galena were placed in the furnace and ignited. As heat developed, some of the metal was extracted and molded into bullets.

The next step in utilizing the lead ores was the log hearth. All lead in Missouri was smelted in this crude affair before 1820. It was built on sloping ground, and consisted of a hearth of stone, surrounded on the front and two sides by a stone wall. The wall was 7 ft. high in front. The top and rear end were left open and in front an arch or opening was made, forming the eye of the furnace. In front of this a pit was dug in the ground, to receive the molten metal. Large logs were rolled in at the back and made to rest upon ledges formed inside, to raise them from the hearth and to give a draft. These logs filled the entire width of the furnace. Small split logs were then set up around the two sides and the front, and the ore was piled on until the furnace was full. Finally, the mass was covered with logs and fuel until the ore was completely surrounded. A gentle heat was started, which was raised very gradually. After 12 hr. the heat was increased and continued for 12 hr. more, 24 hr. being required for each smelting. The furnace treated about 5,000 lb. per 24 hr. and the ore yielded about 50 per cent. of its metallic lead. A considerable quantity of the ore was not desulphurized, and fell between

the logs into the ashes, forming a kind of slag, which was called "lead ashes." This was rich in lead and was frequently treated in a furnace of a peculiar construction, called an "ash furnace." This ash furnace was introduced from Virginia and was a crude reverberatory, built of limestone, which lasted 15 to 20 days on continuous work. As the ashes or residues were more or less oxidized, the charge was immediately reduced at a moderately high temperature. Silica as chert or sand was mixed with the charge. In about 2 hr. the furnace was ready to tap. The slag was tapped first, then the lead on the opposite side of the furnace. The total extraction from the two operations was about 75 per cent. of the lead contained in the ore.

The next step in the progress of metallurgical operations in Missouri was the reverberatory furnace patterned after practices in Carinthia, Silesia and England. At present this type of furnace is no longer in operation. The Desloge Consolidated Lead Co. operated the Flintshire furnace on a small scale until recently.

Ingalls[1] gives the daily capacity as 5 tons of dry concentrates, the cost at $7 per ton of concentrate and the extraction at 91 per cent. To this cost should be added the overhead expense which, in a moderate-sized plant, might increase the cost to $8 per ton.

The reverberatory requires pure ores and on this class of ores gave fair results, considering that no attempt was made to recover the fumes.

The Backwoods hearth was another type utilized for the reduction of Missouri ores and was extensively used in both the Southeast and Southwest districts. The early ones were built entirely of stone and the blast was supplied by bellows.

An improvement was the water-backed hearth, which permitted of continuous work. The Jumbo hearth of Joplin was an enlarged furnace of this character. Before the recovery of the lead fume, these furnaces extracted from pure ores about 70 per cent. of lead contained, not considering the values in the gray slag and fume.

It was not until E. O. Bartlett, in the 70's, installed the bag house for recovering the fume that the hearth could be considered an economic furnace.

On account of the purity of the Southwest galena, the Lone Elm Smelting Co. (now the Eagle-Picher Lead Co.) volatilized all lead possible in short shaft furnaces (slag-eyes), producing a pigment which they called sublimed white lead, being a so-called basic lead sulphate consisting of $PbSO_4$ with varying percentages of PbO and ZnO, the zinc being derived from the Joplin ores, which carry from 2 to 3 per cent. zinc in the high-grade concentrates.

The Cupola process was the next one in line, and differed from the methods just described in that it divided the operations into two distinct steps, each performed in separate furnaces. The concentrates were

[1] *Engineering & Mining Journal* (Dec. 16, 1905), **80**, 1111.

roasted in the usual reverberatory 55 to 60 ft. long at gradually increasing temperatures. By later additions of sand, the roasted concentrates were sintered and partly slagged. The usual amount of sulphur left in the sinter was 5 to 6 per cent. To reduce the sulphur further meant increased temperature to decompose the lead sulphate with resulting heavy losses in volatilized lead. The round (Pilz) blast furnace for reducing the sintered product was later replaced by the rectangular (Raschette) furnace. This method allowed a large production of lead, but the total extraction was not over 90 per cent., due to losses in sintering, smelting and retreatment of mattes.

The reverberatory and hearth methods require pure ores. On this class of ores, these processes give fair results when care is taken in smelting. The yield ranges from 80 to 90 per cent. of the lead in the ores, allowing no recovery in fume. The blast furnace preceded by roasting in reverberatory is adapted to ores that are less rich in lead, and contain impurities which would cause severe losses if treated by any other process. Also, when there is a large amount of ore to be treated with scarcity of labor, the blast-furnace process is the more economical.

I have passed lightly over these old practices, all of which, at least in part, are now abandoned. I obtained much of my information from *Missouri Geological Survey*, Volume VI; *Illinois Geological Survey, Bulletin* 21; and Pulsifer: *Notes fora History of Lead.*

Analyses of Ores as Mined in Southeast Missouri, and of Various Grades of Resulting Concentrates

	Ag, Oz.	Cu, Per Cent.	Pb, Per Cent.	SiO_2, Per Cent.	Fe, Per Cent.	Al_2O_3, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	Zn, Per Cent.	S, Per Cent.	Ni and Co, Per Cent.
Ore	0.12	0.06	5.7	5.0	4.1	4.9	25.5	14.2	0.8	2.0	
High-grade concentrates	0.7	0.13	73.2	1.0	3.5	...	2.6	0.8	0.4	15.0	0.05
Medium concentrates	1.3	0.12	68.6	1.4	4.6	...	3.1	1.4	0.8	15.5	0.06
Low-grade concentrates	1.0	0.30	65.8	0.5	3.1	0.5	4.3	2.8	1.7	13.7	
Flotation slimes	....	0.50	45.0	9.6	4.4	3.1	7.5	4.2	4.0	12.3	
Flotation slimes, high-grade	3.7	0.05	57.8	6.0	2.7	...	2.2	1.4	9.4	15.5	
Joplin concentrates	....	....	80.2	1.1	1.0	...	0.4	...	1.7	13.3	

Analyses of the ores as mined in Southeast Missouri—which are about the average of the district—also analyses of various grades of resulting concentrates, show that the metallurgy is simple, the removal of the sulphur and the slagging of the dolomite in the blast furnace being of prime importance.

There is a little nickel and cobalt in the Southeast Missouri ores, more especially in those found at Mine La Motte and the North American mine near Fredericktown. The nickel and cobalt concentrates with the copper in the mattes, but these metals are eventually lost in the refining of the copper.

James W. Neill[2] made a number of successful blast-furnace runs in the 80's at Mine La Motte with nickel and cobalt containing copper and lead, producing a bullion, a matte low in nickel and cobalt, and a speiss containing a large percentage of nickel and cobalt (23 to 24 per cent.), which was shipped elsewhere to be refined. He obtained the necessary arsenic from Western argentiferous speisses, recovering the silver and gold therein. Later the North American Lead Co. built a plant to separate nickel, cobalt and copper by the Hybinette process. This process produced mattes containing the three metals, which were then separated by a wet method. The plant has been closed for many years, but is now controlled by the Missouri Cobalt Co. and is being remodelled for another campaign.

The big improvements which have revolutionized the metallurgy of the Mississippi Valley ores in the last 10 or 12 years are:

1. The introduction of Huntington-Heberlein pot roasting preceded by a preliminary roast in a mechanical furnace (Godfrey), subsequent smelting in the blast furnace and filtering all fumes through cotton or woolen bags.

2. The introduction of the Dwight-Lloyd sintering process which eliminated preliminary roasting for the concentrates, but not for the mattes, with subsequent smelting in blast furnace with bag-house attachment.

The Godfrey preroasting furnaces were abandoned in the Huntington-Heberlein process, and the Wedge roaster was adopted at Herculaneum for the preroasting of mattes. Later it became apparent that elimination of sulphur was still unsatisfactory, and that there was an undesirable, but necessary, dilution of the concentrates from 70 and 65 per cent. to 40 and 45 per cent. in lead to render the Dwight-Lloyd machine applicable with a single roast.

To avoid these objections, the double roast is being installed by both the Herculaneum plant of the St. Joseph Lead Co. in Missouri and the Federal plant of the Federal Lead Co. in Illinois. A rapid preliminary roast is given on a separate line of Dwight-Lloyd machines, and this preroasted material is then turned over to another set of Dwight-Lloyd machines (Herculaneum), or to Huntington-Heberlein pots (Federal) to reduce the sulphur of the finished product to about 2 to 2½ per cent. and with its lead contents increased to 50 or 55 per cent.

It is expected that the double roast will effect the following improvements over present methods:

1. To avoid the large production of mattes which lock up lead and require expensive re-treatment for its recovery.

2. To avoid diluting a 65 to 70 per cent. lead concentrate to 40 to 45 per cent., which means large additions of barren fluxes.

3. To utilize to the fullest extent the fluxing values in the ores. This

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[2] *Trans.* (1884–85), **13**, 634.

means that no limestone should be added to blast-furnace charge, while some silica and iron will be required. The resulting blast-furnace slags will contain from 6 to 7 per cent. MgO with a corresponding increase in blast-furnace capacity for original concentrates.

4. To recover a part of the copper in the concentrates, which metal is at present mostly oxidized and lost in the large amount of slags made.

The other important improvement in the metallurgy of Mississippi Valley lead ores was the mechanically rabbled hearth, known as the "St. Louis hearth." This machine was developed at the Collinsville Plant of the St. Louis Smelting & Refining Co., and has been fully described by W. E. Newnam.[3]

The St. Louis hearth is applicable only to concentrates assaying 68 per cent. or better in lead, and therein differs from the Huntington-Heberlein and Dwight-Lloyd methods which treat concentrates of varying grades.

When concentrates assay over 70 per cent., it is considered that the hearth shows better recovery in lead, and is the most economical method, although the double roast, when perfected, promises to be equally as cheap as to cost, with the advantage of being able to treat all grades of concentrates.

The peculiar advantages of the mechanical hearth for the higher grade concentrates are:

1. It treats the concentrates undiluted with any flux.

2. It is a very efficient desulphurizer, expelling 95 per cent. of the sulphur.

3. It re-treats all the dust and fume produced, leaving the gray slag as the only product to be sent to the blast furnace, with a large reduction in slag and matte produced.

4. The hearth makes a total extraction of 82 to 84 per cent. of the lead contents of 70 per cent. concentrates. With concentrates assaying 75 to 80 per cent. in lead, 90 to 95 per cent. represents the total extraction, leaving from 5 to 10 per cent. of lead contents in the gray slag. This practically eliminates any re-treatment of mattes.

The mechanical rabbling machine is limited to 68 per cent. lead concentrates, and, for this reason, where lower grades of concentrates are received at Collinsville, the Dwight-Lloyd machine must be used as an auxiliary. All mattes produced by smelting the gray slag of the hearths and the Dwight-Lloyd sinter at Collinsville can be easily handled *raw* on the Dwight-Lloyd, so that a Wedge roaster is not needed for preroasting.

The St. Joseph Lead Co. is erecting at Herculaneum 12 St. Louis hearths to be used in connection with the double-roast process for the treatment of excess of high-grade concentrates.

In the Southeast district, about 75 per cent. of the output of the mills

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[3] The Newnam Hearth. *Trans.* (1916), **54**, 485; Lead Mining and Smelting at Galetta, Ont., *Trans.*, this volume, 579.

is a high-grade product, applicable to hearth work, the remainder being lower-grade concentrates and flotation slimes. In order to sweeten the mixture for the double roast (*i.e.*, to raise its grade) it is necessary to use a proportion of the high-grade concentrates for this purpose. The balance of the concentrates should be worked by the mechanical hearth to obtain the most economical results both as to cost and recovery.

The lead produced from Missouri ores contains very few impurities, and when subjected to a single liquation process can be used for all purposes except corroding. A typical analysis of Missouri undesilverized lead is as follows: Ag, 0.0080 (2.4 oz. per ton); As, trace; Sb, 0.0030; Bi, trace; Cu, 0.0800; Fe, 0.0015; Zn, trace; Ni and Co, 0.0080; lead by difference, 99.8995.

The Herculaneum and Collinsville plants desilverize part of their lead, recovering about 1 to 2 oz. of silver per ton, and producing an unusually pure lead for the manufacture of white lead and other purposes requiring a very pure lead.

The following analysis of Southeast Missouri refined, indicates the unusual character of Missouri desilverized lead: Ag, 0.0005; As, trace; Sb, 0.0020; Bi, trace; Cu, 0.0002; Fe, 0.0004; Zinc, 0.0004; Lead by difference, 99.9965.

The difference from Western refined is almost entirely due to the bismuth contents (about 0.05 in Western refined) from which the Missouri ores are singularly free.

The refining of the Missouri crude lead for corroding purposes follows the usual practice, except that no softening furnaces are necessary.

At Collinsville the lead from the blast furnace is poured molten into the drossing kettle. After carefully drossing to eliminate copper as much as possible, it is pumped into a desilverizing kettle. The hearth lead is charged as pig directly into the desilverizing kettle, drossed and desilverized. Zinc is added, stirred in and the zinc-silver-lead alloy (crust) is removed. This first crust is reworked in succeeding kettle charges until sufficiently high in silver to be set aside. The resulting retort bullion (lead riches) assays from 500 to 600 oz. Ag per ton, and represents a concentration of the silver contents of 250 to 300 tons of lead into 1 ton of retort bullion. The zinc consumption per ton of refined lead produced is about the same in amount as is consumed in refining argentiferous bullion. Special attention must be given to removing copper as much as possible from the lead to be desilverized.

The quantity of zinc-silver-lead alloy made is insufficient in quantity to warrant the operations of retort and cupels, and heretofore the alloy has been sent to some outside refinery to be separately distilled in retorts, and returns are based on the sampling of the various products obtained, such as retort bullion and dross, spelter and blue powder. The assay results check quite closely. Retorts and cupels are being installed at Collinsville, so as to be able to treat the alloy whenever occasion demands.

The Media Mill, Webb City, Mo.

BY H. B. PULSIFER,* CH. E., CHICAGO, ILL.

(St. Louis Meeting, October, 1917)

THE unprecedented high price of zinc ore prevailing through the early months of 1915 caused great activity in the Joplin district of Missouri. The Media mill is conspicuous as one of the first of the new "big mills" of the district. The mill was built over a mine previously worked but abandoned in 1912 because of the leanness of the ore and the large quantity of water encountered. The ore prices prevailing—$100 a ton in June, 1915, as against $50 in 1912—favored the new undertaking. The mill was designed for good recovery so that it might operate in times of depression. In design and equipment, the mill is exclusively local in conception and construction.

Former Exploitation

The tract of land is a part of the large Guinn holdings; 70 acres are comprised in this particular lot, which is situated just a mile north of the center of Webb City on the Alba line of the Southwest Missouri Electric Railway. The ore lies 242 ft. below the surface at the top of the knoll which is surrounded by the well known properties: Black Cat, Ground Floor, Florence, Hurry-Up, Electrical, and Mercantile. Since the abandonment, in 1913, of the Yellow Dog a bit farther to the north, the Black Cat, Ground Floor, and Florence workings had been flooded. However, as there was no cut through from this side into the area it was proposed to exploit, it was thought that moderate pumping facilities would take care of the water.

Three shafts had already been sunk to the ore horizon during the previous operations; the ground was connected by wide drifts and possibly 10 acres mined out, principally about the two southerly shafts which at surface are 8 and 13 ft. lower, respectively, than the collar of the shaft at which the new mill was built. The former mill on the lease was known as the Bohemian Girl. This mill, built in 1910, was removed after some 3 years' work. A head frame and hopper still existed at the middle shaft, sometimes known as the lower Hold Out shaft, but at the most northerly

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of the three shafts there was nothing but a boulder pile when the building of the Media commenced on June 5, 1915.

The previous work at mining this particular ground indicated that the ore from a 10-ft. face would mill to recover slightly over 2 per cent. of concentrate. The concentrates were expected to mill to the usual 60

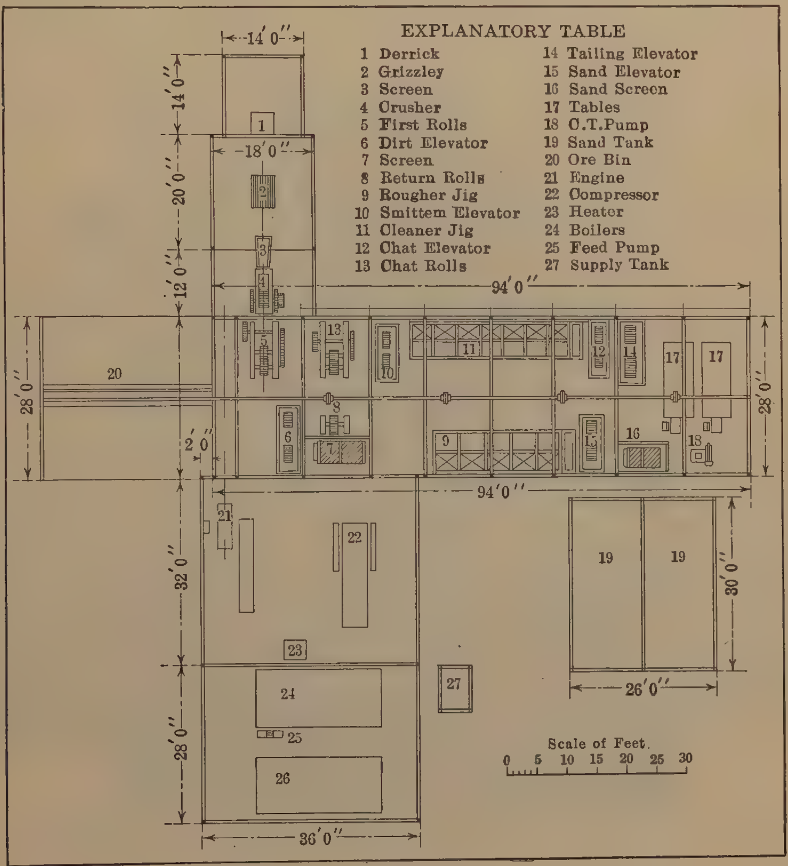

Fig. 1.—Floor Plan of Bohemian Girl Mill.

per cent. zinc content, to contain not over 2 per cent. iron, and to be accompanied by some lead.

Fig. 1 and 6 are of interest because they show jig rooms of almost identical size (28 by 100 ft.), but, with widely different capacities, that of the Bohemian Girl mill per 10-hr. shift being only 250 tons as compared with 750 tons for the Media mill. The Bohemian Girl mill is a typical example of Joplin district practice. It has comfortable space only for the usual ore fall from the elevated hopper through the crusher-elevator screen-jig and tailing elevator system with elevation of rougher heads to the cleaner jig. The two tables occupy about an eighth

of the floor space. The boiler plant with steam-engine drive has always been the main source of power in the district and for the Media mill no system was presented which could compete in low cost of operation and reliability. A year's operation confirms the latter statement. These prime features, which are classical in the district, were merely elaborated in the Media. The Media has the hopper raised enough to get a direct ore flow through the crushers and screens onto the

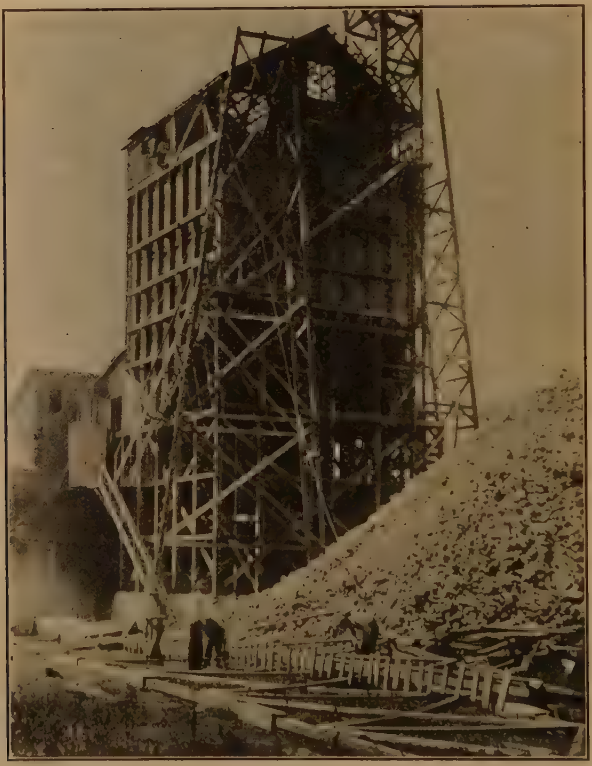

FIG. 2.—HOPPER OF MEDIA MILL ON AUGUST 31.

rougher jigs; this subordinates the first elevators to lifting recrushed material only. It also avoids free "jack" and fines passing the rolls. Other variations from the Bohemian Girl detail consist in treating ground "chats" and coarse sand on a sand jig after elevation and screening. This, however, is again a practice not foreign to the district; dewatering the rougher tailings has long been partially used, although the screen method has come into vogue only during the last decade.

Building the Media Mill

On Saturday, June 5, 1915, decision was reached to start building the mill and prosecute mining on the lease.

William Kenefick and Paul Easby, as proprietors of the Electrical mine, built the Media out of its profits and acted as overseers of the new operations. Paul Easby, in particular, was often on the ground and procured much of the machinery. James J. McLellan acted as manager, did most of the buying and had personal charge of the pumping and shaft enlargement. The author had nominal supervision of the plans and construction of the mill, although, like the others, he devoted fully half the time to other properties. Claude Raymond was appointed engineer of machinery and pumps, also on part time, and William Turner was employed as carpenter foreman.

Fig. 3.—Close View of Media Mill From Tailing Pile.

We had some discussions about general plans. The author proposed a double wing, duplicate section mill with four 42-in. rolls and maximum choke in the 6-cell, 48 by 42-in. roughers. Mr. Easby held for the most compact setting possible, the least possible number of units, and ore elevators opposite the crushers, as in the Bohemian Girl plan. Mr. McLellan strongly favored ore elevators beside the rolls, these to be used only as elevators for recrushed material, by getting one continuous fall from hopper to tailing elevator on the main ore course. A combination of the Easby and McLellan ideas prevailed, thus fully establishing the Missouri type. It was decided to get the mill equipment into a room 28 by 100 ft. and have the return elevators at the side of the three 42 by 16-in. rolls which should be below the crusher and allow of an ore fall from hopper to crusher, to screen, to rougher with a drop of only the over-

size to the rolls to be elevated back into the stream between the crusher and trommel (see Fig. 5).

Construction was practically always ahead of what drawing was done for the mill, proper. By June 27 the mill was covered in. During July no spectacular building took place, although the work advanced rapidly on the interior, the office building, the sand plant, the change house, the engine and boilers, the sand tanks and the pond. The 10 by 10-in. hopper timbers having at last arrived on Aug. 2, the hopper was quickly put in shape (see Fig. 2).

The entire surface plant was about 98 per cent. finished when the author's work was completed and he left for Chicago. The plant could

Fig. 4.—Tailing Pile in May, 1916.

possibly have been in shape for actual milling on that day if the underground water had not made unexpected trouble, which held up operations for 7 weeks before the drifts could be entered. The pace on the mill had been slackened, although it had already been turned over to the operating crew which was adjusting jigs, lining spouts, and putting up pulleys and belts. The mill was first run on Nov. 7, 1915.

The Mill Equipment

The mill equipment included:

Two 12 by 18-in. Webb City crushers; 400 r.p.m.

Two 8 by 4-ft. Webb City trommels, extra heavy fittings; 20 r.p.m.; jackets with 0.5-in. holes.

Three 42 by 16-in. Webb City rolls.

Two return elevators, 22-in. cups, 18 in. apart on belts over 30-in. pulleys at 40 r.p.m.

Two 6-cell rougher jigs, Cooley type; ⅛-in. grates on 48 by 42-in.

cells; 95 r.p.m., 3-in. drop between cells; plunger stroke graded from 2.5 to 1.0 in.

Two 48 by 48-in. Ford-type dewatering screens, belt suspension; 8 r.p.m.

Two tailing elevators, 20-in. cups on 20-in. belt over 30-in. pulleys.

One smittem elevator, 20-in. cups on 20-in. belt.

One chat elevator, 20-in. cups on 20-in. belt.

One sand elevator, 20-in. cups on 20-in. belt.

One chat roll, 30 by 14 in.; Webb City make.

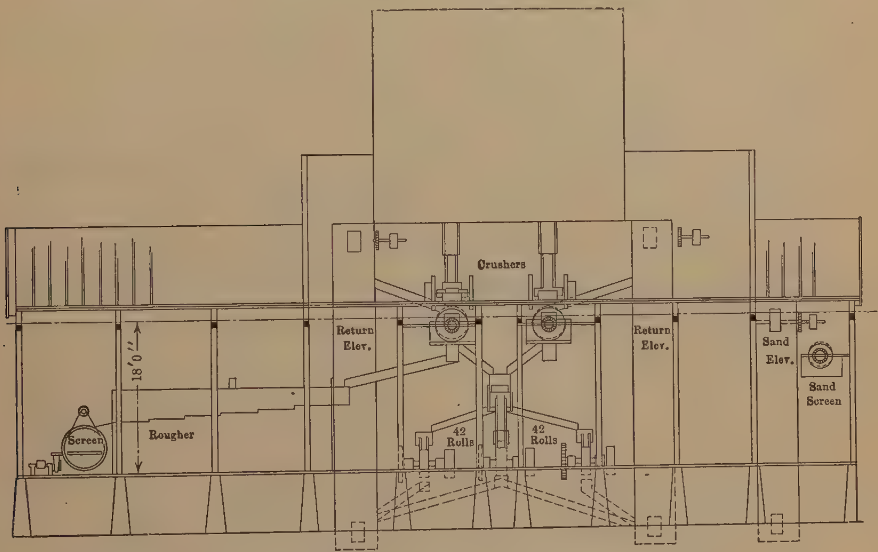

FIG. 5.—LONGITUDINAL ELEVATION OF MEDIA MILL.

One 7-cell cleaner jig, Cooley type; ⅛-in. and 1/12-in. grate slots on 36 by 48-in. cells; plunger stroke, 140 per minute; 1.5-in. drop between cells; plunger stroke from 1.25 to ⅜ in.

One 36 by 72-in. sand trommel, 2-mm. holes.

One 3-cell sand jig, Cooley type; 1/12-in. grates on 36 by 36-in. cells; 175 strokes per min.; 1.5-in. fall between cells, stroke from 1.25 to ⅜ in. in length.

One 6-in. Swaby centrifugal pump, 600 r.p.m.

One 2.5-in. American Well Works centrifugal pump to supply water to crushers.

Hoist

Cross compound, 15 by 30-in. cylinders and 48-in. drum steam hoist; 2-ton skips, in balance, capable of round trip each minute; sheaves 110 ft. above ground. Grizzly bars of 60-lb. rail spaced 5 in. apart on incline, 5.5 in. for flat rails.

Sand Plant

Three 20 by 30-ft. concrete tanks, diagonally inclined bottom to outlet 10 ft. deep.

Sand plant building 60 ft. square with flat trussed roof, elevators extending up through roof.

Originally equipped with feed, return and tailing elevators. The return elevator was removed after trial showed the man in charge that the 11 tables did not give enough middlings to bed even one table.

Tables as follows:

Three wooden decks, Lycans and Stilwell.

Two linoleum decks, medium, Lycans and Stilwell.

Three linoleum decks, fine, Lycans and Stilwell.

One linoleum deck, slime, Lycans and Stilwell.

Two Ford slimers, inclined vibration.

Elevators are 4 ft. width with 20 by 30-in. pulleys, direct geared, and belted to make 45 r.p.m.; 16-in. cups on 16-in. belts.

Trommel after feed elevator, 36 by 72 in.; 1.5-mm. holes.

Hindered settling classifiers 1 to 3 ft. deep along overhead trough for first eight tables, wooden construction.

Three 6-ft. thickeners on floor at head of each slime table.

4-in. centrifugal pump.

Tables driven by 4-in. belts over 14-in. pulleys to approximate 245 strokes on coarser tables and 270 strokes per minute on slimers.

Costs

Some of the more general items may be summarized:

Labor, mill construction	$8,100
Lumber	8,000
Equipment, general, mostly in mill	12,000
Supervision and office	1,500
Cement	1,200
Pond and ditch to mill	1,860
Generator for lighting	600
Compressor	7,500
Labor, power and water installations	1,350
Boilers with fittings (first 4)	5,800
Brick and clay	950
Office supplies	70
Tools	500
Insurance during construction	1,500
Concentrating tables	2,400
Painting, material and labor	350
Mill engine (first engine)	650
Hoist engine	925
Condenser and pump	960
Belting	2,500
Approximate cost of plant	$58,715

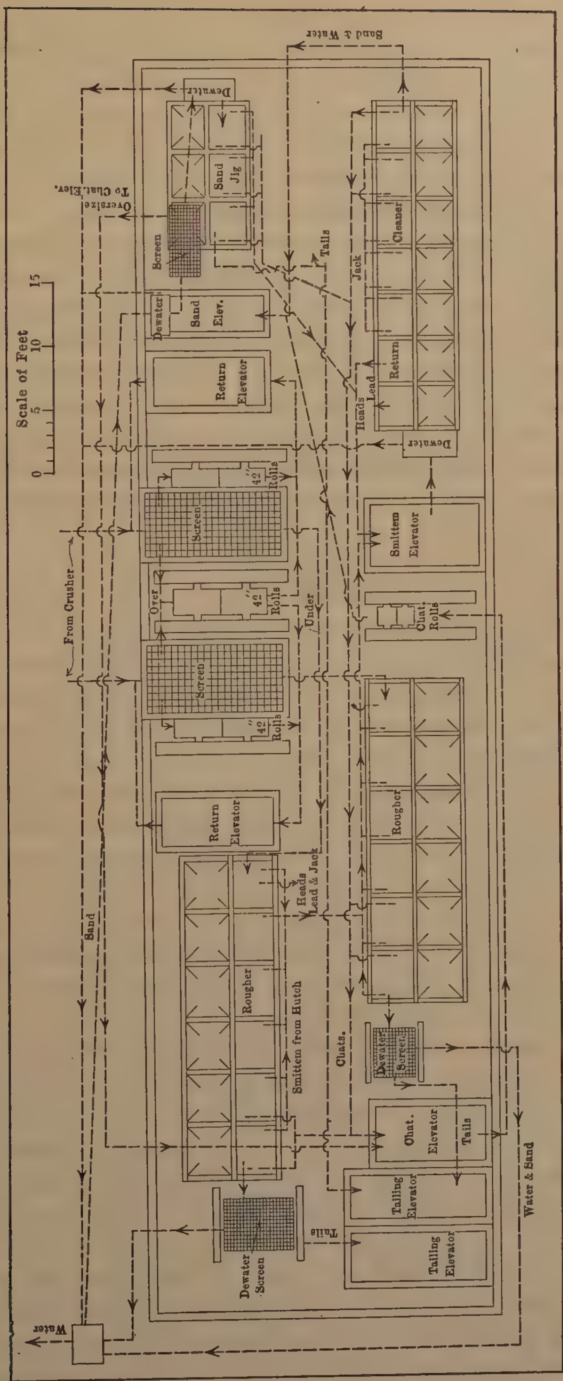

FIG. 6.—FLOW SHEET OF MEDIA MILL.

The above costs comprise the surface equipment. Nearly as much again was expended in draining the mine, enlarging the shaft and excavating the underground hopper for the skips, and for the mine equipment.

Figuring a daily capacity of 1,500 tons and 300 working days a year, we find the plant cost amounts to about $40 per ton of daily capacity, or about 14 c. per ton of annual capacity.

It is interesting to note that after the mine began to produce a respectable tonnage, the first month in 1916, only about 4 months' operation sufficed to yield a net profit equal to the entire outlay for plant and mine.

On Jan. 26, 1916, the Hurry Up mine operations opened a water course which wholly drained the Media workings. To get water for the mill it was then necessary for the Media to put in pumps and build a flume so as to lift water at the Hurry Up and run it across to the Media pond. This was a startling transition from a huge pumping cost to *remove* water, one day, to an almost equally great cost, the *very next day*, to *get* water!

Summary

The Media mill was rushed to completion during the summer of 1915 on war-time prices of zinc ore, and although the mine operation was delayed over 2 months after the practical completion of the surface plant the whole enterprise proved highly successful from every point of view.

The most conspicuous large mill previously existing in the southwest Missouri section was the Yellow Dog which had run its course only 1/8 mile from the Media shaft; the Yellow Dog, however, was really two independent mills with a combined capacity of 1,000 tons, per 10-hr. shift.

The Media has continuously crushed 1,200 tons in 20 hr. in its single jig room. On occasion, 1,800 tons have been treated without difficulty in 24 hr. The capacity has, up to this writing, been limited more by conditions inside the mine than in the mill.

The Media hopper is by far the largest ever erected in this district. The hopper contains, without leveling, 1,200 tons; it rests on thirty-six 24-ft. posts which in turn rest on a massive foundation joined to the concrete lining of the upper shaft. In my opinion, the derrick is far too light for the load and acceleration commonly used, for the vibration at the sheaves during hoisting is considerable. The upper tier of 10-in. posts was never cross-braced and those on the east side of the hopper, which are the longer of the upper tier, gradually yielded to throw the floor of the hopper toward the hoist which is on the north side. The uprights on the sides of the bin remained vertical, but this edge of the floor had shifted approximately 6 in. before it was noticed during the spring of 1916; the movement then apparently ceased, for the measures

taken could not possibly have restrained further progress in this twisting of the floor.

A large pond was constructed at considerable expense; the enormous output of tailings and sand was allowed to encroach until within a few months the effective pond capacity had been reduced to meager proportions. Such management is characteristic of the district.

The three thickeners in the sand plant were overflowing on one side only, at the time of the author's visit in May, 1916. The settling tanks were designed with the intake at the shallow end so as to permit a continuous flow from the deep corner to the tables—a sort of combination of intermittent tank filling and continuous thickening; the local operator, however, quickly changed to the customary way, filling the deep end with sand and making it a wholly intermittent operation. It has already been mentioned that the return elevator was discarded, the middlings being put in the main stream again. These three instances indicate the strong retrograde impulse which is so highly characteristic of the district.

The mine rock has run fairly uniformly a little over 2 per cent. in mineral recovery. The mineral content has averaged about 4 tons of blende to 1 ton of galena. As the tonnage can probably be kept high and the mill has several strong features for unusual recovery, the venture promises well even during periods of low ore prices.

The best features of the ore dressing may be enumerated:

1. Main tonnage fall direct from hopper to tailing elevator; only oversize is re-elevated. Bad turns in main stream nearly absent.

2. Splendid balance of unit capacity; no one unit crowded or choked while others are running light. If any capacity weakness develops it will probably be in the cleaner, chat-regrinding or sand units rather than in main tonnage units.

3. The cleaner is of unusual size, only a very few of this size ever having been built in the district.

4. Ample facilities for dewatering sands and a sand jig in the main mill to treat cleaner sand and ground chats (settled rougher sand can also come back to this jig) relieves the roughers.

5. The sand plant is entirely separate from the jig room, as is now fairly customary in the district. It has an equipment carefully selected and capable of excellent work.

6. Mill water is elevated no higher than necessary; a second small pump supplies the high line to the crushers.

7. The main tonnage flow is through twin units; this arrangement is happily combined with large single units on the enriched segregate. Both streams being in the closest possible proximity, elevator fronts being open and nearly every unit being in full view, minimum attendance is required and instant response to mishap is possible.

The greatest disadvantage which has come to the attention of the author is the undue crowding in the jig room. The elaborate tangle of shafting, belts, spouts and pipes is very unsatisfactory; it is difficult to get units apart and new parts into place. The main aisle is not wide enough, there is no place for spare parts and tools, or even lockers for clothes.

Full use is made of the principle of equipping only with units carried in stock by local supply houses, repairs are thus most fully assured for instant delivery.

The author fully believes this principle of local inheritance was decidedly overdone. A glaring instance is in the case of the tailing disposal—within a very few weeks the great tonnage had built the pile right up to the spout and the customary "dummy" elevator became necessary. The condition existing in May, 1916, is plainly seen in Fig. 4. Tailing disposal as abundantly demonstrated in various other localities could much better have been installed.

The inevitable inquiry as to the application of flotation is answered by saying that no adequate demonstration of its successful application has yet been made in the Joplin district. At another plant the author made experiments on the recovery of blende from the fine waste sludge; preliminary tests indicated that it would not be difficult to obtain a froth product containing over 40 per cent. zinc. However, the present practice is so firmly established that it will be difficult to convince operators of the desirability of adopting flotation on a large scale.

I wish to acknowledge my indebtedness to the management of the Media company for permission to publish this account; to Mr. Fred Hild for assistance in preparing some of the drawings; and to Mr. J. J. McLellan for especial service in confirming the details of this manuscript.

Discussion

J. J. McLellan, Webb City, Mo.—The Media mill, at the time it was built, was the largest mill that had been designed in the Joplin district. It was put up in a hurry, to take advantage of the high prices for zinc ore in 1915, by the owners of an adjacent plant, which had been very profitable. They owned a compound Corliss engine, and to take advantage of a single-unit drive it was necessary to build the mill in the form chosen, which involved making the hopper feed directly through the breakers and rolls, the undersize from the screen passing at once to the jigs, the oversize going through the rolls and returning to the original trommels.

The mill was built in fairly good proportion; crushing was done by jaw breakers, Webb City pattern, and three sets of 42-in. (106.7-cm.) Cornish rolls for reducing the oversize. The jigs were larger than the

usual practice. The roughers were two six-cell jigs, with 42 by 48-in. cells. The cleaner jig was the largest ever constructed in the district up to that time, having seven 36 by 48-in. cells, for treating the rougher jigs. There was also a chat rougher. "Smittem," as used in our district, is the partly concentrated product from the rougher, which is finished on the cleaner jigs. "Chats" are the middlings proper, that is, they are the grains of chert with included particles of sulphide; the very low-grade smittem is sometimes included among the chats. When drawing chats from a jig, we also draw about ten times greater quantity of barren chert in order to insure complete recovery of mineral. This mixture of middlings and tailings is taken to a set of 30-in. rolls, the crushed product then going to a chat rougher, a comparatively small jig, with 26 by 36-in. cells.

The principal departure of the Media mill from the usual practice was to build a hopper sufficiently high to give gravity feed to the jigs, without using elevators. The sludge plant, for table treatment of sand and slime, also contained several innovations in the manner of distributing the feed. The tables were arranged in two rows and the feed ran around the outside in a launder having two or three cone classifiers at the end to settle the slime. The middlings, both sand and lead-zinc middlings, were then returned by an elevator and distributed to two tables, which was new practice in the district at that time.

Tailings are dewatered on a screen, known as the Ford screen, just a large trommel hanging on belts. The tailings are poured over the outside and the water and fine sand, going through the screen, fall into a trough and go to a settling tank. The efficiency of such a screen is about 80 per cent. if it is made large enough. The usual oblong tanks of the district are merely storage tanks, the water entering the deep end and flowing over the shallow end. The bottom of the tank is sloped diagonally so that the mud can be washed out with a minimum of water.

In building the mill, excellent speed was attained. As soon as the decision to build the mill was reached, about 10 days were expended in making a floor plan. No detail drawings were made, as is a common occurrence in our district. We simply tell our foreman and mill builders that we want a mill of a certain capacity and a certain size, and they build it. They are familiar with all the details of construction without plans or drawings of any kind. The lumber is ordered from a lumber yard as they need it; sometimes they make a list beforehand, including a certain amount of 2 by 4's, 6's and 12's, most of which sizes are carried in stock in our district.

In the operation of dewatering the mine, a peculiar experience occurred. During the previous operations, in 1912 to 1913, it had been necessary to buy city water for ore-dressing purposes, the water bill amounting to as much as $200 in some dry months. The map of the underground

workings showed about 10 acres (4 ha.) of excavation, and it appeared to be just a question of putting in enough pumps to drain it. The water stood about 80 ft. (24 m.) deep and about 160 ft. below surface. They started with a "Texas" turbine pump driven from the top, and rented additional pumps and motors. The first pump, delivering about 1400 gal. (5299 l.) per minute lowered the water 40 ft. in 2 weeks. A second pump was lowered alongside the first, but as both pumps were rather roughly constructed it was necessary to repair them both at the end of 3 weeks. They finally got the water down to about the top of the drifts, and there it hung, until they installed a third pump. The mine had been flowing less than 100 gal. per minute in 1912, but when they got it under control, they found it exceeded 2000 gal. per minute, and continued for about 3 months. Then other drainage projects in the immediate neighborhood reduced the water level to such an extent that the Media mill had to install a station at another pumping plant to get the water back again for washing. It cost about $40,000 to unwater the mine within 5 months, and another $5000 to get the water back again. If it had been possible to store a reserve it would have been quite profitable.

The Chairman (Arthur Thacher, St. Louis, Mo.).—Mr. McLellan's plan of raising the feed bin so high that a gravity flow can be obtained to the rougher jigs is very important. I believe this antedates our introduction of the same idea in Wisconsin, although I did not know at that time that the Media mill had done this. At our mill, the great bulk of the material is delivered by the rougher as tailings, and if it can have a gravity flow from the time it leaves the mine bucket until it reaches the tailings elevator, that is a great advantage. The only drawback is that it requires a high headframe; at the Media mill, I observe, they put their sheaves 110 ft. (33 m.) high.

The bins also are exceptionally large, 1200 tons, but that depends on the capacity of the mines. At most of our mines, we could not hoist faster than the mill could treat the ore. Our mills take 1 ton a minute, and we cannot readily, from a single shaft, hoist much more than that amount. At the Media mill they had skips and needed large storage capacity. These Joplin mills are rarely intended to run 24 hr. continuously; a mill runs only while the mine is operating and keeps pace with the mine. Hoisting the ore from one shaft, as is usual, they cannot provide the tonnage to supply a mill for 24 hr., and it is therefore run only one or two shifts.

J. J. McLellan.—The first mill, with which I am familiar, that had direct feed was the Continental mill, built in 1906; the Mammoth Zinc and Lead plant, built in 1907, had the same feature. These two plants are characterized by increased capacity and diminished wear of the

"dirt" elevator belts. Some plants have gone to the other extreme and have put their breakers very low; this causes a large tonnage of 3 and 4-in. material to be handled by the "dirt" elevators with greatly increased wear on the cups and belts of the elevators. Another advantage of direct feeding is that all the fines pass at once to the jigs, without having to go through the rolls, which avoids a great deal of slime.

The Media mill started about the middle of November, 1915, showed a small profit in that month and a very good profit in December. From then until June, 1916, the mill was quite profitable, because of a gradual increase in tonnage, which started at about 800 tons per day and exceeded 1800 tons a day for several months. The mill hoppers in our district are built, if possible, of sufficient capacity to permit enough ore to be hoisted on the day shift to operate the mill through the 24 hr. It has not been possible, recently, to do that, on account of labor shortage, especially of shovelers. We do not operate plants on Sunday; Saturday nights and Sundays are used for repair work. The operating cost at the Media mill, under the best conditions, and maximum production, was as low as 12 c. a ton, including repairs but no fixed charges.

The labor cost climbs rapidly if we do not operate at full capacity, and will reach 20 c. when the mill works in single shifts or two part shifts instead of full time. Our present practice is to work three 8-hr. rather than two 12-hr. shifts. Formerly we used to run one 10-hr. day shift, and a night shift as long as 14 hr. if the ore held out. When there is enough ore, we try to run three 8-hr. shifts, which gives the third shift time to make repairs. Our mill is roughly constructed, and the wear of the chert on rolls, screens, and chutes, is considerable. Barrel shells run less than 2 weeks, and it is about a 2½-hr. job to take out and replace the shells even with an extra set of cores and everything necessary for quick work.

Chairman Thacher.—Cost is closely related to tonnage, and that depends mainly on the mine, in that district. One feature, which prevents these mills from running steadily, is that the mill usually contains only one jig, and as this has to be cleaned periodically, they do not keep in actual operation for 24 hours.

O. M. Bilharz, Miami, Okla.—I consider that a cost of 12 c. per ton is exceedingly low, including labor and cost for repairs. At present, even the largest mill in the Oklahoma-Kansas district is operating at a considerably higher cost, ranging probably between 25 and 30 c. Of course the tonnage is an important factor, and perhaps the reason the Media costs were low is that they were compiled during the early operation of the mill, when repairs were naturally less.

Chairman Thacher.—When comparing Joplin with Miami, Mr. Bilharz must remember that he has a richer ore. In Wisconsin we have

a softer rock; one breaker will handle it, and we need only one pair of 36-in. rolls to crush almost as big a tonnage as the Joplin mills. Our repairs are naturally much lighter, and we can reduce our cost as low as 11 c. for labor, supplies, and power, not including overhead charges.

J. J. McLellan.—Mr. Bilharz is quite right as to the period during which these costs were compiled. Since that time the Media mill has not run at maximum capacity for nearly a year, because of labor shortage, the average tonnage for the last 6 months having been nearer 1200 than 1800 tons. I was speaking only of a short period of maximum output, about the middle of 1916. Since then the cost of supplies has increased 75 per cent. and labor probably 10 or 15 per cent., so that a cost of 25 c. for the Media mill, under maximum operation, would be about right at present. What Mr. Thacher says about the richness of the ore is certainly true. If we were to try to handle 1600 or 1800 tons a day of 20-per cent. ore, such as they have in some of the Oklahoma mines, we could not put enough men in the mill to handle the concentrates, there would not be room for them to work.

S. J. Jennings, New York, N. Y.—Being a newcomer in the Joplin and Miami districts, I am viewing things with a fresh pair of eyes, and some of the questions brought out by this paper seem to me to deserve more critical attention of visiting engineers than we might give without having our attention drawn to them.

The main question that seems to me to deserve attention in the Joplin district is whether the mills are properly proportioned to their mines. We have heard that your mills run only 10 hr. a day because you cannot get more ore out of your mines. The usual arrangement is to have two shafts to a mine, the mill being placed near one shaft, and the haulage tail-ropes being used to hoist the ore, haul it from the other shaft, and elevate it into the mill. It seems to me perfectly feasible to extend that tail-rope haulage to other shafts, at a distance of ½ or even ¾ mile, so that four or five shafts could be contributing to the ore supply of one mill, without adding materially to the running cost and very little to the capital expense.

As to the operation of a mine by leasehold only, extending for 10 years and paying a royalty, that is a favorable circumstance from one point of view and a very unfavorable one from another. It is manifestly unfair to require an operator to work out a mine in 10 years when it ought to be made to last for 20. While the consideration that a dollar today is worth two dollars 10 years from today would impel one to operate his mine quickly, the fact that the market will buy only a certain amount of your material requires you to work it at a reasonable rate. With a certain amount of persuasion from the engineers operating in that district, it is possible that the owners might be induced to alter their

ideas as to the length of leases that they will grant; that would be profitable work on the part of engineers.

As to cost of operation: I was a member of the Zinc Committee of the Advisory Council of the National Council of Defense and a number of operators from the Webb City-Joplin district came before us with elaborate figures to prove that unless they could sell their jack at $70 a ton they could not operate their mines. Jack at $70 a ton, with recovery of 2 per cent. zinc, means $1.40 a ton of mill dirt. I am quite sure that those operators were accurate in their statement of expense. When an engineer states a milling or a mining cost, he is apt to include only a portion of the total costs that have to be paid by an operator. That distinction should always be very plainly brought out, so that the public may realize the difference between what I call total cost and the working cost on which the present discussion is based.

Chairman Thacher.—One point which we must remember very distinctly, as applying to the Southwest, is that, except in rare cases, we have never been able to block out well defined mines; as soon as we can see enough ore to pay for the mill, we put the mill up. We regard ourselves as very lucky if a mine runs two or three years, and we never have enough ore ahead to run five or six years. I think we are all operating mines—I know we are in Wisconsin—in which we could not convince you that we had six months' ore supply. We only have the hope that a mine may last 10 years, but we never have definite information upon which to base large capital expenditure.

H. B. Pulsifer (written discussion*).—Advancing the art of ore dressing in the Joplin district is an undertaking dangerous for the novice, experienced operator or technical expert. Failure is likely to attend any undertaking that introduces innovations, whether for increased tonnage, better recovery or general efficiency.

The Mammoth Zinc and Lead mill, which Mr. McLellan mentions as one in which he was able to introduce several advanced ideas, operated for a few years on unusually low-mineral ground with highly satisfactory mill results. It finally had to close down when the total concentrate recovery dropped below 1.75 per cent. A shaking screen and a thickener which the author installed in that mill worked in spite of all opposition of the other members of the mill crew, simply because no adequate defect showed up to serve as excuse for throwing them out. At that very time, another mill a short distance away was built; it was designed by one of our greatest experts and equipped with the finest machinery from one of the great manufacturing companies; it was possibly the most elaborate and promising mill the district had seen. It made a prompt

* Received Nov 26, 1917.

and depressing failure; a few years afterward an old-style mill was grinding away on the rather lean dirt.

The Media was built with full knowledge of the conflicting conditions; it was attempted to raise the standard at every point possible yet not to invite failure by one or more too radical departures. The sponsors well knew that it would be useless to install any equipment that could not be maintained by local crews.

The main tonnage flow was made as direct and straight as possible, no extra lifts, no bad angles for excessive abrasion, no fall lost to consume precious headroom. The chief obstacles to an even more nearly ideal flow were power distribution and disposal of tailing; a single power source without quarter turns in the distribution practically forced the given arrangement of units. Inability to justify any of the more recent means of tailing disposal held the design to the standard tandem elevators.

Large capacity in all units was conscientiously attended to, every passage being built to avoid an ugly choke. In the Joplin district, it is fairly common for the mills to have some one or more choke units, or turns, or spouts, which hold down the entire mill capacity even though most of the units could care for more; a crusher, a trommel, or an elevator will not infrequently be found performing this retarding function. Considering that with an expectation of handling 70 tons an hour, the mill was able to treat up to 1800 tons a day, we may assert that the flow of dirt and the capacity of the units was correctly planned and executed.

Improved recovery was a feature. Each unit should carry its load easily, while for the rather free-milling rock the sand jig was installed as quite additional. Its load might properly have been sent to the sand plant. The tables were more abundant than in similar plants and they were of a make carefully investigated. Four sorts of decks were used for the different sizes of material treated; the products from classifiers and settlers, and the middlings went to separate tables, and it was possible to handle the sand as fast as the mill delivered it, which gave us an excellent opportunity for developing the sand treatment. Of course nobody was surprised that the practice soon relapsed to former style. In the Joplin district, the jig man and the table man are both likely to be little despots, each in his own room; unfortunately, they are rarely favored with technical information.

An operator who had at one time been making particular effort to better his table practice asked me to check up and see if all was well. A few nights later I observed, and the table man admitted, that the latter was simply shoveling all table middlings into the tailing trough. No wonder he could raise the grade of the concentrates if urged. At another time I asked the librarian in the splendid Carnegie library at Webb City for certain books on ore dressing. She had Dana's mineralogy

and a book on geology, but said that the people of that section were not interested in mining, ore dressing, or metallurgy.

To advance the art of design and maintain the step has been a prominent feature in all Mr. McLellan's mills. They have stood as monuments to his force and courage in the face of the general attitude of labor, the land owners and the interests controlling capital. No matter what increased returns may be promised, the vested interests are strongly against elaborateness, complication, or expense of plant.

Salt in the Metallurgy of Lead*

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THIS paper reports the results of the use of salt in some research work carried on during the past 3 years at the Salt Lake City Station of the Bureau of Mines, which is quartered in the University of Utah, whose Metallurgical Research Department coöperates with the Bureau. Only the work involving the use of salt is described.

The authors wish equal credit given to J. F. Cullen, C. L. Larson, C. Y. Pfoutz, C. E. Sims, H. C. Neeld, R. W. Johnson, F. G. Moses, N. C. Christensen, and G. H. Wigton, who have all in some way contributed toward the success of this work, while connected with the station in one way or another.

Lead Metallurgy Popularly Supposed to be Without Problems

In lead metallurgy it is difficult at first to find metallurgical problems, as the prevailing conception has been that lead is so easily concentrated in its various ores and so easily reduced to its metallic form that the only progress possible is that involved in cheapening methods of mining and of reduction. Most of the problems usually considered are of minor nature and do not involve radical metallurgical changes.

During the three years of work of the Salt Lake Station, the following classification of problems in the metallurgy of lead has been made:

Oxidized Ores.

1. Ores containing only lead.
2. Ores containing lead and silver or gold.
3. Ores containing lead and zinc, with or without silver.

Sulphide Ores.

1. Ores of pyrite carrying some lead.
2. Ores of complex zinc-lead-iron sulphides.
3. Complex sulphides containing precious metals.

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* By permission of the Director, U. S. Bureau of Mines.

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The problems involved in the treatment of oxidized ores of lead are due partly to the nature of the ores themselves and partly to their physiographic environment. It is well known that many lead-carbonate ores cannot be concentrated successfully because of the large losses of lead carbonate in the slime. A circular was issued by the Bureau of Mines in May, 1916, on The Flotation of Oxidized Ores, by O. C. Ralston and G. L. Allen, which described work in the flotation of such slimed-lead carbonate, and a number of mills utilizing the method are now in operation. This method involves the use of the principle of sulphidizing the lead-carbonate particles by the addition of a soluble sulphide, such as sodium sulphide, sodium polysulphide, calcium sulphide, etc. Many oxidized ores contain compounds of manganese or basic sulphates of iron, or peroxides of lead which react with the soluble sulphide used and do not permit the sulphidizing of the lead carbonate particles; as a result, this method cannot be employed for such ores, and the slime problem still remains. In other cases, low-grade oxidized ores of lead are found in localities distant from water so that wet concentration of these ores is impossible. Dry gravity concentration is subject to dust losses in the same way that wet concentration is subject to slime losses, and has the further disadvantage of requiring close screening.

There are portions of Nevada, Arizona, California, Utah, and other Western States far removed from sources of fuel so that only a self-fluxing ore can be smelted at a profit in case the ore is not capable of concentration. In these states it is not uncommon to find mines situated on railroads and not far distant from smelters, whose siliceous lead-carbonate ores are not mined unless the metallic content amounts in gross to from $8 to $14 per ton. The Tintic district, 100 miles south of Salt Lake City, is an example of this. The tonnage of this type of ore which has been developed in many of the older Western mining districts is often large.

The same remarks apply to the other types of lead ores above mentioned. Throughout the semi-arid western mining states are mines in which the water level is quite deep and in which large amounts of these various "dry" lead or lead-silver ores exist.

The presence of small amounts of galena intimately crystallized with pyrite is also a common low-grade type of ore which is used to some extent for the fluxing value of the iron but which is penalized with a heavy roasting charge on account of the high sulphur content. Where the lead cannot be liberated by grinding, this ore has never been milled and is often left untouched in the mine. The southeastern Missouri mills also produce a middling of this description which is largely stored or lost, or milled for the recoverable lead and discarded with some loss.

The problem of the treatment of complex sulphides is so well known that it hardly needs description. The presence of zinc in these mixtures has usually been the stumbling block. The present methods of making

electrolytic zinc or of making leaded zinc-white, have tended toward solving these problems, although the results are still unsatisfactory either on account of the high installation and operation costs or on account of low extractions. The presence of precious metals adds a complicating factor.

Leaching with Salt Suggested by Salt Lake

The use of salt as a metallurgical reagent was immediately suggested by the presence of the Great Salt Lake and by the past work that has been done in "chloridizing" ores with salt. There is not much in the literature about the effects of salt on the lead content of the ores treated by the old chloridizing methods, as lead was not worth much in the days when those methods were applied. In the hyposulphite lixiviation processes, it was known that some lead dissolved in the "hypo" solutions, but the effect of salt during roasting was largely neglected. Nevertheless, as long ago as 1854, M. Becquerel, of Paris, reported work which he had done in roasting lead-sulphide ores with salt, followed by a brine leach in which silver chloride and lead sulphate dissolved, and from which he proposed to recover metallic lead by electrolysis.[1] Due to the fact that there was no commercial electric generator available at that time, his process was doomed to failure.

Chloridizing mills were in operation at Park City and at Silver City, Utah, by Holt & Dern and Knight & Christensen, respectively, at the time that our work was begun. In each of these mills a silver-gold-copper ore was given a chloridizing blast-roast followed by a brine leach, and in each case a small amount of the lead in the ore was found to go into solution, although it was incompletely precipitated by the scrap iron which was used to remove silver, gold, and copper from the solutions. The brines in these two mills were also too dilute to allow of efficient work in the extraction of lead.

The Bunker Hill & Sullivan mine, at Kellogg, Idaho, was also the scene of some interesting work involving a hydrometallurgy of lead, as the Malm dry-chloridizing process was being tested out at the time this work was started. The process involved the treatment of the ore with cold chlorine gas to form chlorides of zinc, lead, iron, copper, silver, and gold, followed by a roast to break up the iron chlorides, and a brine leach to remove the soluble chlorides. The method depended for its success upon the use of zinc compounds to precipitate all of the other valuable metals from solutions with subsequent electrolytic recovery and re-use of part of the zinc. One reason that the process failed on the Bunker Hill & Sullivan ore was the low zinc content of the ore.

Standing on the shoulders of accumulated knowledge we utilized old facts for new purposes. The processes here described were gradually

[1] *Comptes rendus* (1854), **38**, 1095.

worked out, not in the exact order of their description, but in stages, following out various ramifications at various times.

Treatment of Oxidized Ores

The ores containing only oxidized forms of lead and no other valuable metals have been treated successfully by several different methods in our laboratory. One of the simplest methods involves leaching the ore with a strong sodium chloride brine acidified with sulphuric acid. This method depends upon the fact that lead sulphate and lead chloride are both soluble to a considerable extent in a saturated salt solution. "In the cold" it is possible to prepare solutions containing as much as 1.5 per cent. Pb, although it is safer to use enough brine to make solutions containing 1 per cent. Pb or less. If the solutions are heated to the boiling point, it is possible to get concentrations of lead as high as 8 per cent. from lead sulphate and 12 per cent. from lead chloride. Most of our work has been done with cold solutions. After the lead is leached from the ore, it is precipitated from the solutions by the addition of lime, or by electrolysis, depending upon which method is to be preferred.

Several series of small-scale cyclic leaches with both types of precipitation have been made in our laboratory, in order to gain an idea of the impurities which tend to build up in the solutions and their effect on the efficiency of the process. Most of the lead carbonate ores are easily leached and have given extractions of from 80 to 98 per cent. of the total lead, depending upon what portion of the lead was present as unaltered sulphide. The much higher extractions possible with a hydrometallurgical process, as compared with a gravity concentration process, make leaching desirable.

In precipitation with lime, some calcium hydroxide is thrown out of the solution together with the basic hydroxide of lead, and if ferric salts or other metal salts are present they are also liable to contaminate the precipitate. It was found that pre-treatment of the solution with powdered lime-rock precipitated the iron and alumina, formed some calcium chloride which precipitated all of the sulphate present, and did not precipitate the lead. This purified the solution to such an extent that the grade of the lime precipitate could be raised from 30 per cent. Pb to 70 per cent. Pb. In some of the cyclic work, careful washing of all of the salt from the precipitate allowed the preparation of a basic lead hydroxide precipitate containing over 77 per cent. Pb. In all cases this precipitate is somewhat discolored by a small amount of manganese leached from the ore and precipitated with the lead. This precipitation may be prevented by oxidizing the manganese to the manganate or permanganate stage, since these compounds of lime are extremely soluble. As a rule, there is no reason for doing this because the precipitated lead hydrate has never

shown any value as a pigment and our efforts to date have been to produce a high-grade lead product capable of easy reduction to the metallic state.

The lime precipitation is so simple, and settling and filtering of the precipitate so easy, that there is little more to be desired. In case most of the salt is not washed out, a high percentage volatilization of the lead takes place during reduction of the precipitate, the volatile compound being lead chloride. Only in ores that contain some ferrous compounds like siderite, and which hence produce reduced solutions of iron, is there any difficulty in obtaining sufficient purification of solutions with lime-rock preceding precipitation with milk of lime. Under these conditions, long aëration with compressed air in the presence of manganese compounds is necessary during the purification in order to oxidize ferrous hydroxide to the less soluble ferric hydroxide.

Electrolytic precipitation of the lead from the brines by the use of insoluble anodes was not successful, due to re-solution of the deposited lead by chlorine dissolved in the brine. Only iron anodes were found to be permissible. This resulted in a high energy efficiency of deposition, as much as 17 lb. of lead being deposited by the use of 1 kw.-hr. and the solution of one chemical equivalent of iron at the anode. Roughly, ¼ lb. of iron goes into solution for every pound of lead deposited at the cathode. The voltage necessary for this electrolytic precipitation is from 0.3 to 0.8 volts and current densities as high as 60 amp. per square foot are permissible.[2]

The lead is deposited in the form of a sponge which sinks to the bottoms of the electrolytic tanks, from which it can be gathered, washed, dried or pressed, and melted to bullion. The solutions do not build up in iron very much but occasionally excess iron will probably have to be precipitated with lime, a step which has been tested on a relatively large scale at Kellogg.

The necessity of using iron anodes, while it makes a very small energy consumption, is the main drawback to this process of precipitation. Although the electrolytic equipment would be small, it would be more expensive than the simple tanks and filters necessary for lime precipitation and the only place where electrolytic precipitation would be of any advantage would be in the case of ores that give solutions foul with manganese or in which notable amounts of oxidized zinc, which passes into solution with the lead, are present. In this case, the lead can be precipitated by electrolytic methods and the solutions later purified and the zinc precipitated. This phase of the subject has not yet been tested by us beyond the determination that a saturated salt solution will not dissolve much

[2] See paper by C. E. Sims and O. C. Ralston: The Electrolytic Recovery of Lead From Brine Leaches, *Transactions, American Electrochemical Society*, Sept. 28 meeting, 1916, New York City; also *Metallurgical & Chemical Engineering* (1916), **15**, 410.

zinc chloride. This might be expected from an inspection of the data on the solubility of zinc chloride in solutions of sodium chloride.

Summarizing, the process developed consists in leaching a lead-carbonate ore with a saturated sodium-chloride solution, acidified with sulphuric acid, and used in such a ratio to the ore that the solutions shall contain not more than 1 per cent. Pb. After about ½ hr. of agitation, the solution of the lead is complete and lime-rock is added for about 10 min. agitation in order to purify the solution of iron, aluminum and sulphates, and the solution is then filtered from the gangue. Precipitation of the lead with lime leaves a brine which can be used over again. Electrolytic precipitation, using soluble iron anodes, is an alternative.

Commenting on the above process, it must be said that cheap salt, sulphuric acid and lime are necessary. Heretofore there has not been a cheap sulphuric-acid supply in the intermountain regions, but this condition is fast giving way to a new condition in which a number of the smelting companies are anxious to find a market for large amounts of acid. Salt must be cheap, because some is lost in the tailing. Not much washing of the tailing is allowable, as the leaching solutions must be kept saturated in order to dissolve lead. On that account, tailing containing as much as 50 per cent. solution or 12.5 per cent. salt will occasionally be formed, and the only wash water that can be used is the equivalent to that which is held by the damp tailing as compared with the original ore. Not more than two-thirds of the salt can be washed out with this small amount of wash water and the final tailing could hence contain as much as 4 per cent. NaCl, although it is usually less. Since salt can be marketed in Salt Lake City for $2 per ton, and the cost of production in Kansas, New York, California and Utah is known to average less than $1.50 per ton, it is almost certain that there are many localities in which cheap salt will be available. Finally, lime-rock is available in practically every mining locality and can be burned to calcium oxide for about $1.50 per ton and marketed in many localities for not more than $4 per ton.

The second method developed for application to oxidized lead ores was an igneous one in which the temperatures involved did not approach the melting point of the gangue. By mixing salt with the ore and heating to 850° C., it was found that all of the lead would volatilize as lead chloride, without regard to the composition of the gangue. In case powdered coal was mixed with the lead-carbonate ore and the mass moistened slightly so that it left a fluffy mass on drying out, it could be blast-roasted with as low as 2 per cent. of fuel, provided the ore was finely ground. The blast of gas through the charge assisted in the evaporation of the lead chloride and a 3-in. bed would be finished in about 15 min. after ignition. The white fumes of lead chloride contained a portion of the lead in a form which dropped in the flues a short distance from the downdraft blast roaster. The remainder was caught in a single-pipe Cottrell precipitator

which consisted of a 4-in. pipe 5 ft. long, standing vertically, in the center of which hung an insulated wire connected to the high-tension pulsating unidirectional current from a transformer and rotary switch, the precipitator being operated in the standard manner. Five-inch pipes are in fairly large-scale use at the present time and the precipitator might well have been a 5-in. pipe instead of 4-in., but as it was it represented commercial conditions fairly well for the purposes of testing the possibility of precipitation of lead-chloride fumes. All visible fume was removed when gas velocities ranged from 4 ft. to 19 ft. per second, and sodium-sulphide paper held for 15 min. in the exit gases failed to show perceptible blackening when gas velocities of not over 10 ft. per second were used. The temperatures of the gases in the small precipitator varied widely, but were usually close to 100° C.

The use of an ordinary reverberatory roasting furnace in place of a blast roaster for carrying out the first step of the volatilization process is permissible. It has been found that the mixture of salt and ore in passing through such a furnace gives perhaps even higher extractions, although the cost of such a roast is higher, due to higher fuel consumption and greater length of time during which the ore is in the furnace. With a downdraft roaster, such as the Dwight-Lloyd sintering machine, the ore is in the roaster only about 15 min., so that large capacities are obtained in relatively small floor space and with relatively low first cost. As a substitute for the Cottrell precipitator, the bag house is acceptable, although for the same capacity the installation of a bag house costs more than a Cottrell precipitator and the upkeep of bags is a greater expense than the upkeep of the pipes or plates of the electric precipitation method.

Comparing the volatilization method with the leaching method above described, and with ordinary smelting methods for low-grade lead ores, it can be seen that the volatilization process has many great advantages. It is independent of the composition of the gangue, and non-fluxing ores are to be preferred on account of the fact that they do not tend to fuse in the furnaces. The hydro-metallurgical process is often hampered by the formation of slimes during leaching. This and the numerous other woes of hydro-metallurgy are all avoided by the volatilization process. Figuratively speaking, the volatilization process may be regarded as one in which gases are used for leaching ores. Gases are much more cheaply pumped and filtered than are solutions. The Cottrell precipitator corresponds to electrolytic precipitation from leach solutions, while the bag house corresponds to an ordinary filter. The downdraft roaster corresponds to the leaching tank.

We have considered the evolution of this process as one of the most promising steps made in the development of a new metallurgy of lead.

The questions as to the further treating of the lead-chloride fume, which might have proved very difficult, fortunately gave us no trouble.

It was found that the method of handling copper chloride in use at Chuquicamata, was applicable to the lead-chloride fume from the Cottrell precipitator. Upon mixing the lead chloride with between 5 and 10 per cent. of coal fuel and its chemical equivalent of lime, and heating to a dull red heat in a reducing atmosphere, lead chloride is easily transformed into metallic lead with the formation of a calcium-chloride slag. The calcium chloride was found to be a suitable substitute for sodium chloride in the first step of the process, as the silicate of calcium is less fusible than that of sodium, thus decreasing the tendency toward sintering. Thus it would be possible to recover somewhat over 50 per cent. of the chlorine for the first step of the process. Careful tests have shown that from 105 to 150 per cent. of the theoretical NaCl or mixture of NaCl and $CaCl_2$ is necessary during the volatilization step of the process.

This process promises to be one of the most powerful processes on account of the wide range of ores to which it can be applied, and on account of the high extractions of lead possible. Particularly, as compared to concentration followed by smelting, it gives a much higher percentage of extraction in fewer steps.

Oxidized Ores of Lead Containing Precious Metals

In case silver is present in oxidized lead ore, it is very often in the form of silver chloride or silver chloro-bromide. Both lead and silver chlorides are soluble in strong brines, so that the leaching process for lead-carbonate ores can also be applied to some of the ores containing silver. The silver sulphides dissolve so slowly in an acidified brine that the method is not applicable to them. Leaching of such ores depends upon the mineralogical association of the silver minerals. Most of the gold which is often found in such ores is likewise insoluble.

It was found that chloridizing roasting and brine leaching of lead ores containing precious metals could be accomplished, although considerable difficulty was met in the extraction of the lead due to the formation of a lead silicate or other compound which is insoluble in brine. Most of the gold and silver could be extracted after a chloridizing roast. However, it was found preferable to raise the temperature so that from 70 to 90 per cent. of the lead volatilized as chloride leaving gold and silver chlorides in the ore to be leached with a brine. Some of the gold and silver was volatilized with the lead.

While this process was found to be successful and to promise commercial success as well, it was found that on still further raising the temperature it was possible to volatilize all of the lead, gold, and silver. At 900° C., in a reverberatory furnace, giving the ore at least an hour's roast, it was possible to volatilize out the chlorides of lead, gold, and silver

to such an extent that only a trace of lead was left in the calcine and 10 to 20 per cent. of the gold and silver. In other words, the extractions were 99 per cent. of the lead, and 80 to 90 per cent. of the gold and silver.

In no case have such ores been tested carrying more than 10 per cent. of lead. These are known as "dry ores" in lead smelting, as at least that much lead is desirable to allow good extraction of gold and silver. The volatilization process seems to be especially adapted to this type of ore. Since no complex machinery is involved, it would seem that small inexpensive units can easily be erected to treat such low-grade ores economically. It is known that many mines in the arid and semi-arid regions need a process of this description.

Occasionally some copper is present in such ores, and it has been found that extractions approximately as high as those of lead are possible. Zinc, on the other hand, does not yield to this process, the volatilization varying widely for different ores, and being entirely uncontrollable. Some ores have given a maximum volatilization of 75 per cent. while others have given a minimum of 5 per cent., but in no case has it been possible for us to control or predict the amount. A mixture of gold, silver, lead, and copper in an ore is not a hard thing to handle by present standard processes, but the presence of zinc has long been a problem in the lead and copper smelters of the intermountain states. This new process seems to be much better adapted to treating such ores than the present smelting processes.

This process is now being tested on its silicious oxidized lead-silver ore in a semi-commercial plant by the Chief Consolidated Mining Co. of Eureka, Utah. Other installations are contemplated on a larger scale. Some similar work is now in progress in our own laboratory in an attempt to determine what form of furnace can be used to best advantage.

The only limitation to this process is that the ores treated must usually be low in sulphur and not too near to being self-fluxing. From 2 to 3 per cent. sulphur present in the ore usually gives low silver extraction. It is possible that preroasting of sulphide ore followed by the addition of salt will improve this condition, although a few roasts of sulphide ores carried on in our laboratory to test this idea have not given good extractions. However, for the low-grade oxidized ore problem which was met in the intermountain states, this process seems to be a solution.

Oxidized Lead-Zinc Ores

Most of the oxidized ores containing zinc, when leached with sulphuric acid, form enough silicic acid to prevent filtration of the solution. To overcome this, it was found best to apply strong sulphuric acid in sufficient amount to form sulphates of zinc and lead, the mixture of ore and

acid forming a thick mud. On passing this mud through a furnace at about 600° C., all silicic acid is dehydrated and all iron sulphates are broken up, leaving only zinc sulphate soluble in water. The lead sulphate formed is left in the residue from the water leaching of zinc sulphate and can then be removed with a strong brine, although lead sulphate which has been heated to the point of formation of basic lead sulphate is very hard to extract with a brine or an acid brine.

A better method of handling such an ore seems to be to use the volatilization method which has been mentioned above, since heating the mixture of salt and ore causes volatilization of lead chloride, usually leaving most of the zinc untouched. The zinc can then be leached by ordinary methods. By conducting the volatilization at over 900° C. for nearly 2 hr., it was found that only minute traces of chlorine were left in the calcine, as all the chlorides volatilized. Sulphuric-acid leaching and electrical precipitation of the zinc from the solution is possible, although chlorine is known to be an element undesirable for electrolytic precipitation of sulphate solutions.

In case a large amount of lime or similar gangue is present in an ore of lead and zinc carbonate, it is perhaps best to treat it by the standard igneous method for making of a leaded zinc-oxide pigment. The plant involved in such work is rather small and need not be expensive, but since such a process does not use salt it need not be discussed in this paper.

Sulphide Ores

For the treatment of sulphide ores containing lead, the use of a salt metallurgy would also seem to present some advantages. In the case of low-grade galena ores needing concentration before smelting, the methods of the past have often been wasteful, as concentration usually did not give high recoveries. Most of the losses were in the slimes and have now been corrected by the use of flotation, although extractions are often still far from satisfactory. The volatilization method mentioned for oxidized ores can be used on these ores, and its great advantage is that it is possible to produce metallic lead as finished product when treating a low-grade ore, and at the same time make a much higher extraction than was possible by the combined concentration, flotation, and smelting methods. This high extraction is to be balanced against somewhat higher costs.

It has been found possible to roast such ores at lower temperatures, and form lead sulphate which is soluble in saturated brine, and with the presence of some salt the roasting is facilitated so that it may be carried on at still lower temperatures. Very little lead is volatilized, and all gold and silver present can be chloridized so that the lead, gold, and silver can be leached with strong brine. This contains the basis of the leaching

process which has recently been tested on a semi-commercial scale by the Bunker Hill & Sullivan Co. at Kellogg, Idaho, and is still under consideration by them.

In the southeastern Missouri district there are always some iron middlings formed in the milling of the lead-sulphide ores. All the iron middlings contain 10 to 15 per cent. of lead, and occasionally some zinc. A mixture of salt with this material, passed over a downdraft roaster with the shallow bed and a low draft, allows the volatilization of a fume of lead chloride mixed with elemental sulphur from the pyrite. The pyrite acts as the fuel and continues to burn after the lead chloride has been volatilized, but it is not necessary thus to burn up all of the pyrite. It is possible in a 15-min. treatment to send off most of the lead and leave a partly roasted pyrite which can be discharged from the downdraft roaster while still ignited, and quenched with water. No work has as yet been done with the mixed sulphur-lead chloride fume to see if it can be successfully handled. If zinc is present in the form of sulphide, there is not so much danger of overheating the mass, resulting in the matting of the iron sulphide, and good extractions of lead are thus made possible. Under no condition has it been found possible to volatilize out any silver present in mixed sulphide ores. The zinc is unaffected by the roast if it is stopped as soon as the lead is volatilized. It has occasionally been possible to grind the quenched calcine for flotation of the zinc sulphide to form a concentrate containing 40 to 45 per cent. zinc. The proper roast that will allow this is rather too difficult at present for commercial application, but the idea is a good one with which further work might be done.

For the treatment of complex sulphides of zinc, lead, iron, copper, silver, and gold, a salt metallurgy does not seem to present a solution. It is possible to roast such ores with salt and save part or all of the lead, leaving the other metals in the residue, but it can be seen that this does not constitute a solution of the problem, and bears out our statements that the large field of application of these processes comes in the treatment of oxidized ores.

Summarizing, we may say: 1. For the treatment of oxidized ores of lead two methods are available. One involves the leaching of the lead with a saturated salt solution acidified with sulphuric acid, followed by either electrolytic precipitation of metallic lead, or lime precipitation of the basic lead hydroxide with the concomitant regeneration of the salt solution. The other method involves mixing salt with the ore and treating in a blast roaster, or a reverberatory furnace, at a temperature of 800° C., or more. At these temperatures lead-chloride fumes are formed which can be caught in a Cottrell precipitator, and later reduced to the metallic form by treatment with lime and carbon in either a melting pot or a small reverberatory furnace, with the formation of calcium-chloride slag which can be used as a substitute for sodium chloride in the volatiliza-

tion step of the process, and means the recovery of from 50 to 70 per cent. of the chlorine.

2. For oxidized lead ores containing precious metals or copper, the leaching process can be used after a chloridizing roast at slightly over 600° C. The volatilization process can be used provided temperatures of over 900° C. are used, the ore does not tend to form a slag, and the percentage of sulphur in the ore is low. The zinc in most ores is only slightly affected by this latter process.

3. Lead can be recovered from practically any sulphide ore by the volatilization method, but precious metals and copper cannot be extracted. Precious metals and copper are leached after a chloridizing roast, although the roast is more difficult to accomplish than with oxidized ores.

4. The volatilization process promises the solution of the vexing problem of oxidized siliceous lead ores so common in the Western arid regions and has the further advantage of requiring very little water. The chloridizing, roasting, and leaching method promises close competition with treatment of low-grade sulphide ores containing precious metals by gravity and flotation concentration followed by smelting, because of the necessity for only a few simple operations, the production of finished metal, and the higher total extractions usually possible. Neither of the salt processes seems to offer a fundamental solution of the "complex sulphide" problem.

5. Installations of the volatilization method for oxidized ores and of the roasting and leaching method for one sulphide ore are contemplated. The exact data on the ores and processes in these plants will probably soon be available.

Discussion

E. L. Blossom, New York, N. Y.—That this paper deals with a real problem is illustrated by a statement made to the speaker a few months ago by the manager of a silver-lead property: "Our ore contains 60 per cent. silica on which the smelters are penalizing us heavily, and they are not able to take all our stuff even then. We have simply got to find some method other than smelting for getting the silver and lead out of our ores. We have had experts down here and have investigated flotation, leaching and volatilization, of which processes the latter is by far the most promising. We are continuing investigation."

I was somewhat concerned in certain chloride volatilization experiments which we carried on in Denver in 1903, working on a carload of ore from Butte. Now being connected with the Cottrell process, by which method the fumes will undoubtedly be recovered if they solve the volatilization problem, I have looked into the matter again, and believe it can be developed into a commercial process.

On a laboratory scale, working with silver-lead ores, experimenters have obtained a volatilization of over 90 per cent. of the total values in 30 to 60 min. on quantities of 1 to 5 lb., using a roasting dish and a muffle. That looks very good, and would be entirely commercial. On attempting to duplicate these results in a reverberatory furnace, they found that it is more likely to take 8 hr. firing to yield 75 per cent. extraction, which is not commercial at all, as it requires too much time, labor, and fuel, and the whole process is too slow. Of course it can be done, because we have done it in the laboratory.

As to how this may be developed on a commercial scale, I will offer a few of my own ideas. First as to the mechanical aspects: In my opinion, the probable reason for the slowness of reaction in the reverberatory roaster is lack of ventilation of the charge; the metallic vapors hang behind in the interstices of the ore and by their vapor pressure retard the volatilization of the unvaporized particles; and although we stir and rabble, it is practically impossible to keep the charge ventilated. With small charges in a muffle, of course, this can be done; a moment's stirring will give good aeration. One reason for thinking that this is the real explanation is that the results obtained with a revolving kiln of the White-Howell type have shown a higher rate of volatilization; there is a better chance for the stream of gases to sweep away the metallic vapors and bring fresh gas into contact with the material. This experiment was made in 1903, and is now being repeated.

Personally, I believe that the highest rate of extraction will be attained by some form of blast roasting, perhaps down-draft, the mechanical details of which I cannot indicate at present, as nobody has worked them out. By this means, we shall be able, with the minimum amount of gas, completely to sweep away and displace the metallic vapors, just as we wash tailings in a cyanide tank. If we can do it by displacement, we shall need much less volume of gas than if we rely on dilution, and inasmuch as this volume of gas has to be settled or filtered, perhaps by the Cottrell, or bag-house, or some other method, it is important to keep the volume at a minimum.

Next, as to the chemical standpoint: It seems to me that Mr. Ralston has dwelt too little on the importance of metallic sulphates in the chloridization process. We all know, for instance, that if we mix salt and copper oxide and try to chloridize, our chloridization is nil. If we add some pyrite, it sulphatizes in the furnace to sulphate, the acid radicle breaks up the salt and sets free its chlorine, which then chloridizes the copper, and everything goes well. At any rate, something is required to decompose the salt or other chloride (whatever we are using), and for this purpose these gentlemen appear to have relied on silica. I believe that the metallic sulphates have performed this function more efficiently than silica. In our 1903 experiments, we used to mix pyrites with our

charge before roasting, and frequently, just at the finish of the roast, we would add 2½ per cent. more pyrite, stirring it in, with good effect.

Two other points deserve study in this connection, which I can merely suggest, as I have not tested them. First, the possible beneficial effect of giving the charge a sulphated wash before putting it into the furnace. Sulphate solution is often wasted aplenty in concentration plants. This could be evaporated and the impure residium mixed with the charge, either wet or dry, and might do a lot of good. Secondly, the effect of water vapor in stimulating chloridization. I fancy I observed that when moisture was going into the furnace, the process worked a little better, but my observations on that matter were not close enough to quote. In conclusion, I would say that the problem seems ripe for solution, and it looks easy to me.

The Chairman (Arthur Thacher, St. Louis, Mo.).—This problem carries us back to the early days of metallurgy. Probably we all remember, at least in our school days, when studying the leaching of ores, the Ziervogel, Augustine, and Von Patera processes, as they were called. In that connection, the book that I found the most instructive was Platner's *Roast Process*, published in German, probably in 1840. Of course, it did not go into some of the details that we now know, but it discussed more intelligently the real reactions and it has been of great help to me in all my connection with roasting.

My first experience was with hyposulphite leaching, in Mexico. It was called the Von Patera process abroad, and was supposed to employ hyposulphite of soda, but, soda being expensive, they used hyposulphite of lime. This was naturally preceded by a chloridizing roast. Of course, we were not aiming to volatilize anything, which is the essence of the process now under discussion, but we were simply trying to get the silver into the state of soluble chloride. That would appear to be one of the most delicate operations that could be performed in a furnace; for, as we are all aware, when chloride of silver has been formed, a little more heat will blacken it or make it insoluble in hyposulphite. Still, delicate as it was, the process gave an extraction of 90 per cent. and over; in small runs as high as 98 per cent. of the silver has been extracted from the ore, and, curiously enough, quite a percentage of gold. Theoretically, we should not have recovered any gold at that stage, because the soluble gold chloride, if it were present, should have been removed by the water washing which always preceded the leaching; but in practice we extracted a higher percentage of the gold than the amalgamation process recovered.

The question of roasting was naturally important and our experience was the same as Mr. Blossom has indicated, that the reverberatory was very slow; the White & Howell furnace was much more rapid, but the

one that did the work best and most rapidly was the Stedefeldt. This was a shaft furnace, in which the ore was showered down, and although the particles remained in the furnace for only a few seconds, they were so isolated that they chloridized very rapidly. I have always felt that the Stedefeldt furnace in later years has not been given the opportunity that it deserved, and I am still convinced that a good deal of our light roasting should be done in that rapid manner.

There is no question about sulphur having to be present to decompose the salt, but most of our ores contained pyrite or other sulphide which acted in that way.

Stuart Croasdale, Denver, Colo. (written discussion*).—While working on the chlorination of Cripple Creek gold ores in 1893, I discovered that gold could be easily and completely (over 95 per cent.), volatilized from the ore by roasting with salt. That heavy losses are sustained when gold ores are treated in this manner had been known before and had been published by Aaron, but so far as I know, this treatment had never been carried to the extent of complete volatilization of the gold with the idea of making it a commercial process. Dull times followed the panic of 1893, and I was unable to develop this idea until 1897–98, when I was chief chemist for the Globe Smelting and Refining Co., now the Globe plant of the American Smelting and Refining Co. At that time, Edwin C. Pohlé was chief assayer for the same company. Some years previous he had made similar discoveries with reference to silver, while searching for a process to treat the heavy baryta ores from Aspen, Colo. We compared notes and began our investigations with the idea of developing a process that would treat all ores containing metals that were volatile as chlorides. We spent a great deal of time on low-grade gold-silver-copper ores, for which, at that time, there was no satisfactory method of treatment. We worked also on complex lead-zinc ores, trying to make a preferential volatilization of the gold, silver and lead from the zinc, which seems possible, but under exacting conditions it may be difficult to reach in practice.

Our process was patented in all countries where mining is done and patents obtainable. I think we had 23 in all. In the United States, we were confronted immediately by a fraudulent interference which necessitated fighting our case through the highest courts. We were sustained in every instance, and finished our litigation with a legally established basic patent granted to us for the treatment of ores by volatilizing their metals as chlorides.

These patents were published in the *Engineering & Mining Journal* of Oct. 31, 1903. The litigation, of course, delayed the publication of our results, but they were finally published in detail in the *Engineering &*

* Received Sept. 22, 1917.

Mining Journal of Aug. 29, 1903, followed by more or less discussion on Sept. 19, 1903, and subsequent issues. At that time, the process attracted considerable attention in this country and in Europe, and it seemed to be known almost universally among mining men and metallurgists. I was asked about it for years afterward during my professional travels throughout the mining country. My attention has been called to it in Rose's *Metallurgy of Gold*. In passing, I might say that in the original article mentioned above, two references to tables are reversed and there are one or two typographical errors, all of which were beyond my control to correct.

This process was "rediscovered" in 1912 by Ben Howe, who tried to use it on the ores of the Gwalia Consolidated gold mine in Australia. He had the attention and financial backing of the foremost mine operating company in England, but the local conditions were against him. His results were published, as something new, in the *Mining Magazine* of London, December, 1912. This article was published in abridged form, or as an abstract, in all of the mining and metallurgical papers in the United States during 1913 and created more or less discussion. It was also published in the Australian mining papers. These articles were answered by H. C. Parmelee, Western Editor of *Metallurgical & Chemical Engineering*, and by myself. My answer to Mr. Howe's original article appeared in the *Mining Magazine* of London in April, 1914.

Aside from the references made to later processes and inventions like the Cottrell precipitator, the Dwight-Lloyd machine, and other roasters, the information given by Mr. Howe and by the authors of this paper concerning the volatilization process, might have been obtained in every detail and in greater detail by referring to my original article.

I greatly appreciate the endorsement of the process and the confirmation of my results, obtained nearly 20 years ago. I hope the authors will continue their work in this direction, for there is ample room for improvement in the metallurgical treatment of low-grade complex ores, especially those containing lead and zinc. Flotation processes will frequently enable one to bring the problem to a more favorable locality for treatment in a concentrated form, but it is still complex. I presume our patent is still in force. It is controlled by the company organized for its exploitation, but I think I can assure the mining and metallurgical industry that it will never be used to hinder the progress of the art. I am always ready to assist in any way that I can to further the development of this really fascinating process.

E. L. Blossom.—It was with the Metal Volatilization Co., of Denver, of which Mr. Hawkins was manager, that I gained experience in this matter in 1903. We had a second-class grade of ore in Butte, on which, owing to fine crystallization, the concentrating losses were high; it was

also very siliceous and as our furnaces were already choked with silica we were looking for some other method. As a result of our tests at Denver, I figured that the volatilization would have to be about eight times faster to make the process commercial. It required 1000 cu. ft. of air to contain 1 oz. of copper, and at that time we did not have the Cottrell process, but only the elaborate, troublesome, wet-washing spray tower, and submerged filter, which were entirely inadequate. The chance for developing the process now is very much better, and also the need of it is greater.

Clarence L. Larson, Kellogg, Idaho (written discussion*).—This paper has been of much interest to us at the Bunker Hill & Sullivan because of our direct concern in the matter. In its reference to the Bunker Hill work, the paper may mislead some readers, as reference is made under both the oxide and sulphide divisions. In the oxide division, it is stated that iron does not build up very much in brine lixiviant during cyclic operation, but that occasionally excess iron may have to be precipitated with lime, and that such a step has been tested out in the Bunker Hill work. This is not the case. In order to explain this error, I shall give a brief description of the Bunker Hill process.

The ore contains 8 to 10 per cent. lead, as sulphide, 16 to 20 per cent. iron, as carbonate, 50 per cent. silica, 3 per cent. sulphur, 1.5 per cent. manganese, 2.5 per cent. alumina and 3.5 to 4 oz. silver. It is first given a roast in the presence of 3 to 5 per cent. salt, and the roasted product is then leached with saturated brine which has been slightly acidified, the leach being carried almost to neutrality. Silver and copper are then removed from the leach solution, before sending it to cells for the electrodeposition of its lead. Either soluble iron anodes or insoluble anodes may be used.

The brine is used cyclically and in such use iron and sulphur are found to build up. In case iron anodes are used, iron builds up steadily from their dissolution. If insoluble anodes are used, iron builds up very slowly, as its source is then confined to the small soluble amounts in the roasted product. As the iron approaches a concentration of 20 grams per liter, it precipitates out upon the calcine during the leaching step and the lixiviant maintains an iron concentration of 20 to 22 grams per liter, despite long continuance of the cyclic operation. No evidence has ever been noted that such iron content of the lixiviant brine has any effect, either harmful or beneficial, on the process.

We find, however, that sulphur has an obstructing effect; it builds up steadily and rapidly from the soluble sulphates of the calcine. Degeneration of the lixiviant becomes noticeable as soon as the sulphur in-

* Received Oct. 9, 1917.

creases to 5 or 6 grams per liter, and by the time it reaches 10 to 12 grams per liter leaching becomes infinitely slow. Regeneration of such solution immediately follows the removal of this sulphur by precipitation.

In case insoluble anodes are used, sulphur is eliminated by means of calcium chloride. In case the iron anode is used, a choice is given between the use of calcium chloride and the following scheme, which would probably be the more economical one in most localities. Burned lime rock is crushed into the brine, after all its lead has been removed by electrolysis, and the following reactions take place:

$$CaO + H_2O = Ca(OH)_2$$
$$Ca(OH)_2 + FeCl_2 = CaCl_2 + Fe(OH)_2$$
$$CaCl_2 + Na_2SO_4 = CaSO_4 + 2NaCl$$

The iron precipitation is instantaneous, but the precipitation of the sulphur requires several hours of agitation. I trust that this explanation will make clear the reason for our precipitating out the innocent iron content of the solutions. Inasmuch as sulphur is objectionable and has to be eliminated, it will be usually found economical to give calcines a water wash preliminary to brine leaching. Average calcine will thus allow elimination of about one-half of the total brine-soluble sulphur.

The paper states that the use of insoluble anodes is unsuccessful. Although at the Bunker Hill we obtained a current efficiency of only 40 per cent. and a yield of but 1¼ lb. of lead per kilowatt-hour, insoluble anodes might have favor under certain conditions. For instance, the present high cost of iron cuts deeply into the economy of using the iron anode. Coupled to this, the conditions of cheap calcium chloride and cheap power might make the insoluble anode the economical one.

Because of the low concentration of lead possible in brine, a very efficient agitation of electrolyte is required. Perhaps the most suitable method would be the use of air bubbling through canvas or other porous-bottom cells. The deposited lead is finely crystalline and adheres for the most part to the cathodes unless the deposit is allowed to accumulate to too great a thickness, when it falls to the bottom. Such lead is spongy and presses between the fingers like an amalgam. In the event of inadequate circulation, a light, fluffy, floating lead results, which is oozy and worthless, as it cannot be recovered or pressed. Sponge lead is pressed as soon as it is removed from the solution. Otherwise it rapidly oxidizes and then requires a reducer in its melting. For this reason, sponge lead should not be dried, as is suggested in the paper under discussion. Pressed sponge lead does not occasion much of this difficulty.

Although the use of lime as a precipitant of lead from brine solutions may be advisable under certain conditions in preference to electro-deposition, I believe the paper under discussion has given its use too favorable a notice. The Bunker Hill attempted such lime precipitation

about 3 years ago on a semi-commercial scale, with very unfavorable results. Since that time some of the early faults have been remedied, but the method still has difficult features. The addition of lime to brine containing lead demands a rather exacting condition if lead hydrate alone is to result. The proper amount of lime must be brought into immediate reaction with the proper bulk of solution. Otherwise oxy- or basic chlorides of lead are formed in appreciable amounts.

The greater the quantity of solution to be handled, the greater the difficulty of carrying out the precipitation efficiently. It is not sufficient to dump the necessary amount of lime into the top of a large tank; it is advisable to attain the ideal condition of a small stream passing a certain point, where the proper amount of lime is added constantly; this is not an easy operation. The presence of oxy-chlorides reduces the grade of the precipitate, makes its reduction to bullion more difficult, and increases volatilization losses. Also, chlorine is lost to the brine cycle. Settling of the precipitate is readily carried out, but filtration is difficult and requires a large mechanical capacity. The washing of the precipitate presents the most serious problem, as the loss of salt would be a waste and would also seriously lower the precipitate grade. The washing out of this salt is very difficult and inefficient at best, due to the excessive cracking of the filter cake.

The paper indicates that in leaching work it is advisable to control the ratio of solution to solids so as to make the solutions carry but 1 per cent. of lead. To insure complete extractions in charge leaching, this may be true, but I believe that practice would demand countercurrent work. At the Bunker Hill, countercurrent work gave the best extractions and allowed us to carry maximum lead concentrations (15 to 18 grams per liter at 20° C.).

The paper limits the current density for electro-deposition of lead from brine to 60 amp. per square foot. At the Bunker Hill, we find that the degree of electrolyte agitation is the measure of the allowable current density, if the latter be considered the highest density at which the quality of the deposited lead is suitable for recovery, pressing, and melting. Under ideal agitation conditions, we have secured satisfactory sponge lead at 100 amp. per square foot while that secured at 150 amp. per square foot was not amenable to pressing. The best density, however, would probably be much below 100 amp. per square foot.

In the sulphide division of the paper, it is stated in reference to a gold-silver-lead ore, that the use of salt in the roast chloridizes all of the gold and silver, as well as permits the roasting of the lead at lower temperatures. I question this statement. My experience has been that the roasting of lead sulphide ores is independent of temperature between 300 and 750° C. in so far as lead extractions are concerned. Salt in the roast aids the lead extractions by 10 to 15 per cent. in case the ore is low

in sulphur. If the ore contains sufficient sulphur in excess of that combined with the lead, zinc and copper (in other words, pyritic sulphur), the extractions are very nearly complete and salt becomes unnecessary. Silver and gold values, however, always require the presence of salt in the roast. Silver extraction is not so complete as that of lead and is likely to vary considerably, due to the fact that silver is precipitated out of brine by compounds found in calcine, such as PbO, PbS, ZnS, and FeS_2. However, very satisfactory results are possible on lead-silver ores. In the case of lead-silver-copper ores, the copper compounds respond to low temperatures better than high temperatures, while the opposite is true of silver minerals. But as in the case with the cleaner lead-silver combination, very satisfactory results are possible.

The presence of gold in appreciable amount practically prohibits the use of a brine process alone. Rarely will 40 per cent. of the gold be extracted. A combination process then becomes necessary. On a certain lead-silver-gold concentrate, roasting to volatilization of the lead, followed by cyaniding, has proved very efficient in small-scale work. On this same material, roasting, brine leaching and cyaniding proved very effective. Some recent work at the Bunker Hill has been very interesting in that it points to the handling of a zinc-lead-silver-copper combination by an adaptation of the brine process.

O. C. Ralston (written discussion*).—We note with interest Mr. Blossom's agreement that the treatment of siliceous oxidized lead-silver ores is a serious metallurgical problem and also that our chloride volatilization method seems to present the solution of the problem. Since the paper was written, we have tested a rotary kiln of small size for the application of the above method. Most of this work has been done by Messrs. John C. Morgan and Louis G. Gerhart. A small kiln built in our laboratory was of about the following dimensions—7 by 1 ft. (2.13 by 0.3 m.) with a 12-in. (30.48 cm.) firebrick lining, in which two of the longitudinal rows of brick were set out 2 in. (51 mm.) to form lifters for the ore, in order that it might be more efficiently showered down through the heated gases from a large Hauck oil burner which was operated under pressure. The ore passed through this kiln in about 30 min. and the residue was assayed for the various metals that the ore contained. It was found that the lead, copper, and gold were extracted to about the same extent as was possible with a 2-hr. treatment in the assay muffle. With the silver, only about 90 per cent. of the muffle extractions was obtained, except in runs where the temperature of the Hauck burner happened to go higher than had been planned, and in these cases the silver extractions were satisfactory. With a kiln nearer the dimensions of a cement-making kiln, say 100 by 6 ft. (30 by 1.8 m.), there is no doubt

* Received Nov. 27, 1917.

that it would be possible to give the ore a longer time of passage through the heated zone and thereby obtain satisfactory silver extractions without having to resort to higher temperatures.

This justified our hopes and confirms Mr. Blossom's conjectures that some such furnace as the White-Howell would be the proper one, on account of the better ventilation of the ore while being roasted. Although we used lifters in the rotating kiln, our observations were that the charge seemed to be sufficiently agitated without their presence, and that they caused too much dusting.

We also built a small brick-dust chamber and a Cottrell precipitator to catch the volatilized chlorides. Due to imperfect connections, a great part of the gases did not pass through the precipitator, so that to date we have no data on the quantitative recovery of the precipitator. The dust chamber proved to be too small to cool the gases sufficiently for the precipitator, since they entered the latter at over 400° C. A water spray introduced into the flue at the end of the dust chamber cooled the gases to 120° C. before they entered the precipitator. Under these conditions the gases discharged from the precipitator were practically completely freed from suspended chloride particles. Six 4-in. pipes in the precipitator sufficed for handling the gases, but the leaks in the apparatus have not yet been sufficiently stopped up to make possible measurements of the volume of gases treated, and their velocity in the treater. However, plenty of fume has been collected and its analysis seems to bear out our smaller-scale laboratory work. From low-grade ores, only a low-grade fume is obtained, as dust gets through the dust chamber into the precipitator and thus dilutes the fume; from higher-grade ore much higher-grade fume is obtained.

At the time of writing, a 30-ft. kiln is under test at the Chief Consolidated mine in Eureka, Utah, under the direction of G. H. Wigton. It will probably have a capacity of 25 to 50 tons of ore per 24 hours.

We believe that the data so far collected justify us in stating that the chloride volatilization in a rotating kiln is sufficiently rapid to meet the conditions mentioned by Mr. Blossom, and that in every other way such a kiln meets the mechanical and chemical conditions called for by the reactions involved. The gases are confined as they are in no other kind of furnace; a fair heat exchange between the gases and the fresh ore should be obtained in a long kiln before they discharge into the flues; and besides, the furnace gives continuous stirring of the ore without having any rabble arms or other iron parts to be burned and corroded at the rather high temperatures required by the process.

We have devoted our attention rather to the engineering phases of the situation than to the development of a new process. The chloride volatilization idea is very old, and all we have done is to find out what machines will adapt themselves to the conditions required. The only

thing in the nature of a discovery was our finding that sulphides and sulphates could be omitted in the roasting of a completely oxidized ore, and nevertheless good extractions of the metals be obtained by chloridizing. Mr. Blossom assumes that we have been able to obtain chloridizing, in such cases as these, by the substitution of silica for the oxides of sulphur in liberating the chlorine from the salt. We would call attention to the fact that several ores either high in lime or in iron, and with practically no silica present, gave equally good extractions of the metals, but that the necessary temperatures for this work were well above the melting point of sodium chloride; sodium chloride also vaporizes quite noticeably at these temperatures, although it does not boil at as low temperatures as do the chlorides of most of the desirable metals involved. It would seem that the chloridizing agent in this case was probably the vapor of sodium chloride, and the following reversible reaction proceeds to completion due to the fact that the products of the reaction are continually swept away: $2NaCl + MO \leftrightarrows MCl_2 + Na_2O$ (where M represents a divalent metal like lead).

We have found that when the gangue of an ore is easily fusible and, therefore, a low temperature of operation is desirable, the addition of various sulphates is beneficial, especially in the case of zinc and silver ores. In fact, where the zinc extractions are erratic with salt alone, the addition of sodium or calcium sulphates to the salt will permit higher extractions of the zinc, and these can be steadily maintained. Therefore Mr. Blossom's suggestion that these problems be tested is a good one.

Mr. Thacher has called our attention to the old Stedefeldt chloridizing furnace. Frankly, we had not considered it up to the present date and are not prepared to say whether it would give satisfactory chloride volatilization or not. The suggestion is worthy of a trial, because great rapidity of the reaction will allow the use of a smaller amount of combustion gases per ton of ore, and hence a smaller Cottrell precipitator.

Mr. Larson has discussed the brine-leaching portion of our paper quite frankly. Our only regret is that we were unable to induce him to publish a paper on the Bunker Hill & Sullivan work, coincidently with our own paper. He was associated with the earliest part of our work on salt in the metallurgy of lead, and left our laboratory to continue this work with the Bunker Hill & Sullivan company. Most of the work that we have described was done after he left us, and while he worked almost entirely on sulphide ore, the greater portion of our own work has been on oxidized ores. This has brought about some differences of opinion as to the best way to handle the resulting solutions, as can be understood from his discussion, but in the main we are still willing to stand by the points which he has questioned, by saying that our paper simply stated our experience.

In further comment on the work at the Bunker Hill & Sullivan, it would seem that only their peculiar business necessity caused their building of a lead smelter. The need for immediate action resulted in the smelter, and meanwhile the chloridizing method perfected by Mr. Larson was being tested. While such a process might easily cost more than gravity concentration and flotation followed by smelting, the higher extractions made possible by a hydrometallurgical process tend to balance it, if not speak in its favor; and even at the same profit per ton of ore, the greater extractions made in a hydrometallurgical process would indicate that the interests of conservation demand the use of this process. We earnestly commend this and the volatilization process to the attention of metallurgists.

We are glad that our paper has brought from Mr. Croasdale the interesting historical items which he has contributed, and we hope that time will see both himself and his associates rewarded for the excellent work which they did so many years ago. We notice that Mr. Croasdale is evidently under the impression that his patents cover all of the volatilization methods mentioned in our paper. We do not believe that any one previously knew that good chloride volatilization could be carried on with a completely oxidized ore, without the presence of any sulphur. We have not read Mr. Croasdale's patents with close attention to this point, but our impressions are that practically all previous patents mentioned the necessity of having sulphide ores, in order to obtain satisfactory chloridizing.

The Zinc Ores of the Joplin District. Their Composition, Character and Variation

BY W. GEO. WARING,* M. SC., WEBB CITY, MO.

(St. Louis Meeting, October, 1917)

Introduction

The winning of zinc and lead ores from the comparatively shallow deposits of the Joplin district presents few such problems for the mining engineer as are encountered in deep ore mining and in the handling of complex ores.

However, for those who are interested in the origin and nature of ore deposits in general, there is much here to investigate, and upon studies in this direction will depend, in no small measure, the future of an industry that has produced during the year 1916, in southwest Missouri, lead and zinc ores to the value of $26,319,383; adding $46,022,164 to the metallic wealth of the nation. Including the adjacent mines in Ottawa County, Oklahoma, and Cherokee County, Kansas, which are really continuations of the Joplin district, the total value of ores produced reached the sum of $34,089,503, and that of the metals, $58,927,941.[1]

The ores produced in the Joplin district are remarkable on account of their almost uniformly high metal content in both lead and zinc. Their superiority in this regard is unparalleled, and is doubtless due to certain geological conditions that are now possibly explicable, provided that the artesian circulation = ascension hypothesis of the origin of the ore deposits is proved beyond question to be the only tenable explanation of the facts.

These, as well as other valuable zinc-ore deposits, are bound to be exhausted in time, for ordinary mining purposes, and it concerns the mining engineer that the conditions under which ores of this character were actually deposited in the form in which they occur, may, if possible, be ascertained, in order that he may learn where to look for similar deposits in other regions, or even how to reproduce artificially such condi-

* Analytical Chemist

[1] *U. S. Geological Survey, Bulletin* No. 320 (June, 1917).

tions as may render available other low-grade sources that cannot be exploited by present methods.

The purpose of this memoir is to contribute certain specific data relative to the character and composition of the ores produced in the Joplin district, in the hope and belief that it may be of service in future studies of the problem just now suggested.

Without further prelude I therefore append here:

1. A table exhibiting variations in the commercial grades of concentrates produced from 34 of the principal mines of the district during 3 months in 1916. This table is only tentative and is incomplete for the reason that few mines keep their first- and second-grade ores separated, while very few have all the products assayed for lead and lime, and none for cadmium.

2. To remedy in a measure the last-named deficiency, a table giving a series of analyses of concentrates from a number of mines throughout the district, showing in particular the variations in the cadmium content of the concentrate. This table shows also the zinc, lead, and iron assays and, in 18 cases, the copper. Only the cadmium, however, is an essential or molecular constituent of the sphalerite, the other metals being adventitious; part of the iron coming from the crushing plates and rolls in the mill, the rest, with the copper and lead, being derived from the minerals marcasite, chalcopyrite, and galenite, associated with the sphalerite in the crude ore or "mine dirt."

3. Supplementary analyses, in more or less detail, of typical calamine, blende, and lead ores, mine water, leach, and flue dust from the calcination of Webb City—Carterville blende concentrates.

The analyses of calamine, and of mixed blende and calamine, show the extent to which they are separated from each other by the ordinary jig process. The calamines here consist almost entirely of the silicate (hemimorphite or true calamine), with very little smithsonite.

The analyses of lead ores (low-grade and average) show the character of the impurities. A considerable portion of the iron abraded from the crushing rolls finds its way into the galena or lead concentrates because it goes through the screen of the first cell of the jig, where all the lead is collected (see footnote to analysis *f*, Table 3).

An analysis of a high-grade, pure honey-yellow, soft flaky sphalerite from the Cottonwood district in Utah, associated with galenite, greenockite, rhodochrosite, calcite, etc., is given, with permission of the U. S. Geological Survey, for comparison with two full analyses of blende concentrates from the Joplin district.

The analysis of acid mine water is typical of the mine waters in certain areas about Galena, Joplin, and the Carterville districts. The mine waters of some other areas are alkaline. The leach water from tailing dumps was collected in April, 1917, after a rain.

The Data

TABLE 1.—*A Tabulated Summary Showing the Highest, Lowest, and Average Assays of* 426 *Shipments of Zinc and Lead Ore Concentrate from* 34 *Principal Producing Mines in the Joplin District (Including Seven Miami, Okla., Mines) During January, February, and March,* 1916. *From Settlement Assays.*

Mine No.				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Zinc-blende Concentrate	First-grade (Jig Stuff)	Highest	Zn	63.40	63.40	61.10	60.80	59.50	63.00	63.40	63.50	63.60	55.00	56.80	63.00	57.90	61.60	60.50	58.90	57.90
			Fe	2.40	2.30	1.90	4.10	3.10	1.30	2.10	2.40	1.70	5.40	3.40	1.10	3.90	2.05	3.90	2.20	3.90
			Pb	0.40	0.85	.....	1.00	0.45	1.10	0.68	0.15	0.12	.....	.....	0.10	3.00	0.45	1.90	2.00	
		Lowest	Zn	62.80	62.00	60.20	56.90	58.60	62.20	62.20	61.30	60.60	52.60	51.20	62.20	56.00	61.50	58.00	56.90	56.00
			Fe	1.80	1.70	1.30	1.90	2.40	0.90	1.10	1.70	0.70	3.50	2.10	0.80	2.90	2.10	1.90	1.40	2.90
			Pb	1.50	0.20	.....	0.50	0.18	0.40	0.21	0.02	0.04	.....	.....	0.05	1.40	0.32	0.30	1.00	0.10
		Average	Zn	63.09	62.53	60.65	57.78	59.03	62.60	62.80	62.46	61.16	53.90	53.78	62.72	57.14	61.52	58.96	57.68	57.14
			Fe	2.04	1.96	1.55	3.29	2.84	1.05	1.39	2.02	1.28	4.30	2.70	0.90	3.29	2.01	2.72	1.56	3.29
			Pb	0.25	0.47	.....	0.76	0.31	0.80	0.45	0.12	0.06	0.84	2.86	0.08	1.97	0.25	0.87	1.41	1.97
			CaO	.....	.....	.....	.....	.....	.....	.....	.....	1.55	.....	0.47						
	Second-grade, Mixed, Table Concentrate, and Tailing Concentrate, Average		Zn	53.40	58.37	56.43	57.04	51.64	54.70	57.70	52.95	53.50	48.62	49.30	51.80					
			Fe	5.00	3.48	2.51	2.45	6.63	2.72	2.21	6.25	2.40	5.70	3.40	3.60					
			Pb	0.97	1.20	1.30	2.50	0.85	1.45	0.61	1.12	0.04	1.51	3.83	1.00					
			CaO	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.55							
Galena Conc.	Jig Concentrate	Highest	Pb	.....	79.20	.....	81.80	.....	77.20	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	81.90
		Lowest	Pb	.....	75.70	.....	80.30	.....	75.30	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	81.00
		Average	Pb	.....	78.18	.....	81.00	.....	76.67	.....	.....	.....	.....	80.85	.....	81.80	.....	.....	82.80	81.53
	Table Concentrate Average		Pb	.....	.....	.....	.....	.....	72.50	69.20	.....	.....	.....	70.90						

Mine No.			18	19	20	21	22	23	24	25	26	27	28	29	30	Calamine 31	Calamine 32	Calamine 33	Calamine 34
First-grade (Jig Stuff)	Highest	Zn	56.00	62.20	61.30	56.00	63.30	54.90	60.30	55.70	60.50	47.60	57.70	59.40	59.70	51.70	41.30		
		Fe	2.90	1.60	3.30	3.80	2.70	7.80	2.60	2.40	2.90	13.80	2.30	5.20	3.10				
		Pb	4.00	0.40	0.50	.....	0.30	0.23	0.29	.....	.....	1.00	1.00	1.40					
	Lowest	Zn	52.80	60.80	55.90	54.00	60.80	51.20	58.80	53.00	57.80	36.70	55.80	52.70	59.00	48.10	32.00		
		Fe	2.50	1.30	1.20	2.90	1.40	2.70	0.70	1.60	2.10	7.20	2.10	1.50	2.50				
		Pb	2.65	0.10	0.15	.....	0.10	0.03	0.03	.....	.....	0.36	0.50	0.80					
	Average	Zn	54.57	61.82	59.05	54.93	62.02	53.41	59.63	54.32	59.85	42.10	56.31	56.98	59.25	50.08	38.05	38.50	42.80
		Fe	2.69	1.48	1.97	3.31	2.06	5.97	1.80	1.94	2.46	10.75	2.16	2.80	2.75				
		Pb	3.34	0.16	0.22	1.67	0.16	0.12	0.17	1.15	.....	0.62	0.65	1.12	0.37				
		CaO	.....	.....	.....	.....	.....	1.17	1.64	0.15	.....	0.32							
Jig Concentrate, Average		Pb	.....	.....	.....	.....	.....	.....	74.35	80.30	.....	80.62	78.00	73.70					
Table Concentrate, Average		Pb	.....	.....	.....	.....	54.40	.....	58.10	.....	.....	53.70							

TABLE 2.—*The Relative Content in Zinc, Cadmium, Copper, Lead, and Iron, of Various Zinc Ore Concentrates (Including One Sample of Rich Crude Ore or "Mine Dirt") as Shipped from the Joplin District,* 1902–1916

Name of Mine	Locality	Description	Per Cent., Zn	Per Cent., Cd	Per Cent., Cu	Per Cent., Pb	Per Cent., Fe
Standard......	Fortuna, Mo.	Rosin jack	61.97	0.436	0.133	0.815	0.55
Standard and Gundling (2 cars)........	Fortuna, Mo.	Rosin jack	63.50	0.530	0.107	0.323	0.90
Big Six (3 cars)	Aurora, Mo.	White jack	56.75	0.018	0.004	None	1.88
Uncle Sam	Aurora, Mo.	White jack	61.20	0.590	0.014	None	0.75
Ayres and Messick.....	Stratford, Mo.	Lead gray, pebble jack	64.05	0.841	0.015	0.625	0.80
Hudson.......	Pleasant Valley, Mo.	Ruby red pebble jack	62.05	0.322	0.030	None	0.61
Jack Rose.....	Alba, Mo.	Yellow blende	54.70	0.225	.....	1.025	1.36
Sphinx........	Neck City, Mo.	Yellow blende.	65.77	0.135	0.077	None	0.55
Willard........	Neck City, Mo.......	Yellow blende.	63.30	0.320	0.064	0.410	0.80
Big Circle.....	Oronogo, Mo.	Yellow blende	56.90	0.110	Tr.	1.510	1.64
Oronogo Mutual	Oronogo, Mo.	Yellow blende	61.87	0.410	0.018	0.265	1.38
McKinley.....	Carterville-Prosperity	Dark brown rosin blende	57.20	0.550	None	5.290	1.25
Lucky Budge..	Carterville-Prosperity	Dark brown rosin blende	59.90	0.380	......	0.990	1.06
Welton.	Carterville-Prosperity	Dark brown rosin blende	57.40	0.410	0.036	1.340	3.05
Holy Smoke...	Carterville-Prosperity	Dark brown rosin blende	60.13	None	0.070	0.102	1.20
Ramage No. 1.	Carterville-Prosperity	Dark brown rosin blende	63.60	0.410	0.018	0.265	1.38
Twin Cities....	Carterville-Propserity	Dark brown rosin blende	46.00	0.270	......	0.600	8.60
McDonald.....	Carterville-Prosperity	Dark brown rosin blende	57.27	0.277	0.030	1.500	1.56
Edgar Zinc Co.	Carterville-Prosperity	Dark brown rosin blende	61.30	0.300	......	0.134	1.00
Edgar Zinc Co.	Carterville-Prosperity	Dark brown rosin blende	62.45	0.490	......	0.003	0.90
Edgar Zinc Co.	Carterville-Prosperity	Dark brown rosin blende	59.60	0.200	......	0.033	1.32
Munson (Continental)	Carterville-Prosperity	Dark brown rosin blende	59.20	0.068	0.054	0.293	1.90
Maude B.	Webb City	Rosin jack	55.70	0.227	Tr.	Tr.	4.90
October.......	Webb City	Rosin jack	58.10	0.460	None	0.590	2.74
Majestic.......	Webb City	Rosin jack	57.05	0.330	None	1.610	1.82
Three Shaft...	Webb City	Rosin jack	60.20	0.140	0.086	0.887	2.08
Bertha A.......	Webb City	Rosin jack	59.80	0.220	......	0.650	3.40
Wingfield......	Webb City	Rosin jack	60.00	0.375	......	0.600	3.00
Little Butler...	Webb City	Rosin jack	56.90	0.327	......	1.610	2.20
Incline........	Webb City	Rosin jack	55.60	0.083	......	1.400	4.50
White Dog....	Webb City	Rosin jack	60.00	0.410	......	1.070	1.70
Av. of 5 cars, misc.........	Webb City	Rosin jack	60.28	0.520	0.040	0.372	2.00
Yellow Dog....	Webb City	Rosin jack	57.95	0.710	......	1.620	1.60
Yellow Dog, "Mine Dirt"	Webb City	Crude Ore	12.78	0.170	......	0.293	1.90

The very high average metallic content of the zinc and lead jig concentrates is evident from an inspection of Tables 1 and 2.

These concentrates are produced from a heterogeneous mixture of galena and sphalerite, with jasperoid flint and chert gangue, along with clay, shale, limestone, and varying amounts of calcite, dolomite, marcasite, and chalcopyrite, and in some places calamines and oxidized lead minerals, with no other preparation for the jig process than crushing it, just as it is blasted down in the mine, to pass a ⅜ or ½-in. mesh screen. It then runs over a roughing jig to carry off the coarser barren matter and

the "smittem" from the "rougher" finally passes over a 5 or 6-cell cleaning jig which yields the clean galena from the first cell, return middlings from the second and clean blende from the remaining cells.

The explanation for the production of such surprising results from a seemingly crude and simple process is to be found, apart from the skill of the jig operators, in the physical condition of the mineral and gangue.

The studies of the ore deposits of the Joplin region, made by Siebenthal[2] during 1902–1914, have shown pretty conclusively that these ores of zinc and lead were most probably deposited during the present geological epoch, from circulating artesian waters of alkaline composition, which had become charged with H_2CO_3, $Si(OH)_4$ and H_2S during their circulation under pressure through the Cambrian and Ordovician strata, whereby they were fitted to dissolve the metallic sulphides that abound in the rocks of those formations, and to transport them in solution west-

Table 3.—*Supplementary Analyses*
Blende and Calamine Concentrates
Jig Separation

Mine	Blende	Cal.	Blende	Cal.	Cal.
	a	*a*	*b*	*b*	*c*
Total Zn, per cent	56.40	57.30	58.40	51.80	40.00
Zn as ZnS, per cent	39.80	18.20	40.10	8.65	2.91
Zn as Calamine, per cent	16.60	33.10	18.30	43.15	37.09
Fe, per cent	2.20	1.50	3.40	1.30	0.80
CaO, per cent	0.45	0.42	0.15	0.40	4.05
MgO, per cent	Tr	.....	Tr	.....	1.44
Pb, per cent	0.34	None	0.52	None	None
Insoluble, per cent	11.67				

a and *b* from Wentworth.
c from Princeton.

Lead Concentrate

	d	*e*	*f*
Pb, per cent	74.30	80.00	77.09
Zn, per cent	5.20	5.16	3.05
Fe, per cent	4.20	3.90	4.32*
Insol. per cent	1.30	.....	0.91

d Carbonate ore, Granby.
e Galena, Granby, containing 6.5 per cent. Pb as $PbCO_3$.
f Composite sample of concentrate shipped from all parts of district.
* Metallic Fe = 1.28 per cent. + Fe as FeS_2 = 3.04 per cent.

[2] C. E. Siebenthal: Origin of the Zinc and Lead Deposits of the Joplin Region. *U. S. Geological Survey, Bulletin* No. 606 (1915).

TABLE 3.—*Supplementary Analyses.*—(*Continued*)

Blende Concentrate	*g*	*h*	Utah Sphalerite	*i*	*j*
Zn, per cent.....	58.260	59.200	Zn, per cent...	48.200	65.020
Cd, per cent....	0.304	0.058	Cd, per cent..	0.903	0.112
Pb, per cent....	0.700	0.293	Pb, per cent...	11.020	0.505
Cu, per cent....	0.049	0.054	Cu, per cent....	0.220	0.440
Fe, per cent......	2.230	1.900	As, per cent...	Heavy	0.013
Mn, per cent....	0.010	None	Sb, per cent...	Tr.	0.021
$CaCO_3$, per cent.	1.880	Tr.	Fe, per cent...	0.970	0.032
$MgCO_3$, per cent.	0.850	None	Mn, per cent...	1.740	0.406
$BaSO_4$, per cent.	0.820	Tr.	Ca, per cent...	1.910	0.082
SiO_2, per cent....	3.950	6.971	Mg, per cent..	0.340	Tr.
Ni,Co, per cent..	......	Tr.	Ba, per cent...	......	0.112
S, per cent......	30.420	31.301	SO_4, per cent..	......	0.078
			S, per cent.....	......	32.120
	99.773	99.777	CO_2, per cent.	......	0.320
			SiO_2, per cent.	......	0.287
					99.548

g Composite sample of 3800 car shipments from the Joplin district.
h A typical sample of sheet-ground blende concentrate from Mine No. 13.
i Specimen from Cottonwood district, Utah; composed of honey-yellow sphalerite, with pink spar (rhodochrosite) galena, calcite, etc.
j Selected clean honey-yellow sphalerite from *i*.

Flue Dust from the Calcination of Webb City-Carterville Blende Concentrates

	Water Soluble, Per Cent.	Total, Per Cent.
Zn......................	22.526	29.876
Cd......................	2.726	2.850
Pb......................	Tr.	2.661
Cu......................	......	Tr.
Fe......................	0.120	3.650
CaO....................	......	1.040
MgO....................	......	0.440
SiO_2....................	......	10.264
SO_4....................	37.240	37.240
S......................	0	3.040
Tl......................	0.030	0.030
	62.642	91.091

wardly until, arriving at localities where structural conditions, namely, the absence of the impervious covering of the Chattanooga shale and the presence of fissures communicating with the overlying subcarboniferous limestone beds, enabled the solution, under heavy pressure, to rise to

TABLE 3.—*Supplementary Analyses.*—(*Continued*)

Mine Water (A Typical Acid Mine Water from Duenweg)	Parts Per Million	Surface Water from the Leaching of a Tailing Dump Pile	Parts per Million
Free H_2SO_4	29.40	Free H_2SO_4	None
$FeSO_4$	157.40	$FeSO_4$	1,033.6
$Fe_2(SO_4)_3$	179.00	$Fe_2(SO_4)_3$	Trace
$ZnSO_4$	2,840.50	$ZnSO_4$	9,092.7
Pb	1.90	Cd	3.7
Cd	5.25	Cu	3.2
Cu	0.30	Mn, Ni, Co, etc.	None
Mn	24.50		
Ni, Co	2.10		
$CaSO_4$	Saturated		

within say 300 ft. (91 m.) of the surface terrain, where exits to the surface were found at points not completely covered by impervious strata.

At such points the diminution in pressure permitted the gases, H_2S and CO_2, to be released, so that the dissolved silica precipitated as a gel, forming amorphous jasperoid when hardened, in place of the limestone taken into solution as bicarbonate, while the sulphides slowly crystallized out, selectively, in the forms in which they are found here. That is, each mineral species was deposited in separate crystals or crystalline masses with little or no interpenetration, the galena usually in the roof of an opening and the sphalerite on the floor. Certain faces of sphalerite masses are well oriented, and the chalcopyrite, where it is found, is almost always perfectly oriented throughout.

In that member of the lower Mississippian known as the Grand Falls chert, in the vicinity of Joplin, and in what is known as the Duenweg-Oronogo ore channel, passing between Webb City and Carterville, is found the most productive "run" of the now famous "sheet-ground" deposits. The Grand Falls chert formation consists, where it is not ore bearing, of thin bands of alternating chert and limestone. Where it is ore bearing (as in the Duenweg-Oronogo ore channel, which follows the direction of a nearly vertical fissure 3 or 4 ft. wide, now entirely closed, but perhaps not everywhere, by ribbons or upright layers of jasperoid flint), the limestone bands have been replaced with the sulphides, and in places with calcite, and jasperoid, on each side of this fissure for distances varying from several hundred feet to half a mile or more. The wider extensions of the ore-bearing ground appear to coincide with oblique cross-fractures, some of which show slight displacement. The main fissure of this particular channel is well defined in the lower workings of the Oronogo Mutual mine, and in the Oronogo Circle mine it is visible almost to the surface. It has

been traced at other points, from Duenweg to near Georgia City, for a distance of 15 miles.

Over the sheet-ground horizon, large cavernous deposits have been mined, which were undoubtedly outlets of the artesian mineral solutions, for they were filled with irregular deposits of blende and galena and other minerals in a gangue of brecciated chert associated with clay and fragmentary débris resulting from the tumbling in of surface strata. These form the "open-ground" or soft-ground deposits, and were the first to be mined in this district.

The conditions which obtain in other portions of the region differ in some details, but the hypothesis as to their probable origin applies to all.

The sulphides, then, in these deposits are hard and crystalline, and when they are comminuted, as in the processes of milling, each minute fragment remains approximately an isometric crystal; the galena particle a cube, the sphalerite a hextetrahedron, and the chalcopyrite a perfect tetrahedron. The marcasite, being orthorhombic instead of isometric, gives fragments which are sufficiently near to being roughly spherical to answer for practical purposes in concentration.

The chert and jasperoid flint gangue, which makes up more than 90 per cent. of the "mine dirt" in the sheet-ground mines, splits up into conchoidal splinters and flakes that are peculiarly adapted to jig work, facilitating a rapid and complete separation of the ore.

Variations in the color and physical appearance of the sphalerite are to be seen, which can hardly be accounted for satisfactorily by difference in chemical composition or in the composition or nature of the associated gangue or of the rock inclosing the deposit. Light yellow sphalerites were formerly preferred by ore buyers as being "softer," and therefore more easily calcined and retorted than the supposedly harder red ores or the hard "pebble jack," the name given to small individual hextetrahedrons scattered through clay or other soft matrix.

The ores of the Joplin district differ considerably from those of the Wisconsin region, although their origin is supposed to be of similar nature. The Wisconsin ores contain a notable amount of arsenic and antimony, two elements that have not been found in the Joplin ores. The marcasite is associated with the Wisconsin ores to an extent quite unknown anywhere in the Joplin district. I have no exact data as to the relative proportion of cadmium in the Wisconsin and the Missouri zinc ores, but I am inclined to believe that the ratio of cadmium to zinc is somewhat higher in the Joplin ores than in those of other zinc-producing regions, and particularly in those ores that are recovered from deeper-lying deposits.

Zinc, cadmium, and tin, along with the rare metals gallium, indium, thallium, and germanium, are closely related in the periodic group system of Mendeléeff.

Lecoq, in 1875, with an assistant, Jungfleisch, extracted 62 grams of the fluid mercury-like metal gallium from 2400 kg. (5291 lb.) of Pierrefitte blende, and had exhibited it in the Paris Exposition of 1878.

In 1915, a very observant watchman at the zinc smeltery of the Bartlesville Zinc Co., in Oklahoma, discovered globules like mercury, that had oozed out in warm weather upon the drusy surfaces of the cakes of lead residuum that had been taken out of retorts used for the redistillation of crude spelter, at a temperature of about 350° C. below the highest temperature attained in the ore distillation. He collected a handful of the liquid metal and submitted it to F. G. McCutcheon, the chief chemist, who found that it gave the reactions of both gallium and indium, and it was later proved to be an alloy of those elements.[3]

In 1916, G. H. Buchanan,[4] chemist of the New Jersey Zinc Co., taking advantage of the volatility of the higher chloride as predicted by Mendeléeff in 1869 for the hypothetical *ekasilicon*, succeeded in proving quantitatively the presence of the extremely rare metal germanium in Joplin ore, as well as in some products of zinc metallurgy.

Still more recently, I have found thallium to be a constituent of zinc ores in the Webb City-Carterville district.[5]

Practical methods for extraction of the rarer metals from zinc-blende ores are as yet lacking. It is necessary that the development of such methods shall be controlled by very delicate spectroscopic tests, requiring the use of the expensive high-grade quartz prisms and lenses, such as are provided in the new Hilger spectrographs, because the important spectral lines of germanium and gallium lie mostly in the ultraviolet. On account of the high temperatures required for the volatilization of these elements, only the spark and arc spectra are available, using resublimed graphite electrodes. A good beginning, however, has been made since Urbain pointed the way several years ago, and a special research is now being considered under the direction of the U. S. Bureau of Standards.

In this connection, it should be noted that Urbain[6], in examining by this method 64 blendes from different localities, found that all of them contained Cd and Pb; 59 contained Ga, Ag, and Cu; 41 contained In; 38 contained Ge; 32 contained Sn; 26 contained Sb; 10 contained Bi; 9 contained As; and 5 contained Mo.

It is also of interest to note the progress and surprising results that are being obtained by the application of a new instrument of research

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[3] W. F. Hillebrand and J. A. Scherrer: Recovery of Gallium from Spelter in the United States. *Journal of Industrial and Engineering Chemistry* (1916), **8**, 225.

[4] The Occurrence of Germanium in Zinc Materials. *Journal of Industrial and Engineering Chemistry* (1916), **8**, 585.

[5] See flue dust analysis.

[6] *Revista real academia cientifica*, Madrid, **8**, 49–63.

—the Roentgen rays—to the crystal surfaces of zinc blende, etc., and the analysis of the remarkable interference phenomena which are produced.[7]

To return to the question of extracting the rare elements from zinc ores, I have to suggest that no particular difficulty lies in the way of separating the cadmium, thallium, and indium along with copper, etc., in metallic form, from the leach liquors of the electrolytic process by cementation upon granulated spelter, while gallium and germanium would doubtless concentrate in the electrolyte until their precipitation by some method yet to be devised could be effected.

The analysis of flue dust from Webb City and Carterville ore calcination, showing an increase in the cadmium-zinc ratio to 15 times that of the original ore, and the conversion of 95.4 per cent. of the cadmium from CdS into soluble $CdSO_4$, suggests that the extraction of the cadmium from such material should be made by hot water, followed by cementation upon zinc, instead of returning the flue dust, as is commonly done, to the calcining furnace and ultimately compelling the unwelcome element to enter into the spelter or be dissipated in fume. As about 75.5 per cent. of the zinc in the flue dust is also soluble in water, it should be precipitated by lime after the separation of the cadmium, and the lime-zinc precipitate distilled by a special method for the recovery of the zinc. This has been proved to be both feasible and economical. The recovery of the cadmium from the retort fumes is not difficult, according to methods already in use, but the fumes could also be leached like the flue dust, using an acid solvent, and the cadmium could then be recovered by cementation.

Discussion

V. H. Gottschalk, Rolla, Mo. (written discussion*).—In connection with Mr. Waring's quotation of Urbain's work, attention may be drawn to a remark found in the report of the session of the Société de Chimie Physique,[1] held in December, 1909, that, while all blendes contain gallium in larger or smaller quantities, those blendes carrying indium are generally free from germanium, and *vice versa*. Urbain, furthermore, expresses his conviction that this behavior, being related to the conditions of deposition of the ore, should be an important factor in determining the origin and mode of formation of zinc blende deposits. In searching the literature for the naturally expected sequel to these remarks, there is found a report on the extraction of germanium from

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[7] For a full description of the methods used, and the manner in which the solution of the molecular structure of blende, fluorite, pyrite, and calcite is effected by the X-ray spectrometric method, see the memoir of Prof. W. L. Bragg in *Proceedings of the Royal Society of London* (A), **89**, 468–489.

* Received Oct. 6, 1917.

[1] *Journal de Chimie Physique* (1910), **8**, 107–9.

blende, after many months' work, on a commercial scale;[2] then a paper[3] on some Spanish blendes, all containing cadmium, examined by Urbain's method (this paper contains, also, very important work on the composition and color of blendes, as well as on the occurrence of organic matter and of inclusions in such blendes), and a second paper on Spanish blendes in general, in which this original point of Urbain's concerning the relative quantities of the three rare elements is taken up.[4] Angel del Campo finds that of 68 specimens examined, four only were entirely free of germanium, gallium, and indium; 38 had small proportions, while 19 were notably, and seven relatively, rich in the three rare metals. He found the relative frequency to be: gallium in 60, germanium in 50, and indium in 38; all three occurred in 29 cases; germanium and gallium together in 18; gallium and indium together in four; germanium and indium together without gallium, in none; four had gallium only, three had germanium only, and one had indium only; in 23 it appeared feasible to extract these elements. Finally, it may be worth mentioning that the discovery of gallium in commercial aluminum[5] brought forth two notices[6] regarding the distribution of gallium, one claiming that the richest source of this rare metal is the cast iron made at the Middlesborough furnace in England. There is also a note on the extraction of germanium from Vichy water[7] and a discussion of the distribution of indium.[8]

Since the publication of Urbain's original remark in 1909, it has seemed of peculiar interest and importance to examine our American blendes in a similar manner, and now that the war conditions have led to such a spectacular demonstration, in McCutcheon's gallium-indium specimens, of the vivid reality of the existence of these rare metals in our mid-West zinc ores, it is to be earnestly hoped that the research, said by Mr. Waring to be under consideration by the U. S. Bureau of Standards, will be undertaken as soon as conditions permit.

Supplementing Mr. Waring's very valuable cadmium analyses in Missouri zinc products, the following references, while probably of historical interest only, may nevertheless be worth while. Regis Chauve-

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[2] G. Urbain, M. Blondel and M. Obiedoff: Extraction of Germanium from Blende. *Comptes rendus* (1910), **150**, 1758.

[3] Ramon Llord y Gambóa: Composition of Blende from Picos de Europa *Anal. Fis. Quim.* (1910), **8**, 413–21.

[4] Angel del Campo y Cerdán: Spectrographic Study of Spanish Blendes. *Anal. Fis. Quim.* (1914), **12**, 80–96.

[5] Ch. Boulanger and J. Bardet: Gallium in Commercial Aluminum and Its Separation. *Comptes Rendus* (1913), **157**, 718–9.

[6] Hugh Ramage: *Chemical News* (1913), **108**, 280; F. H. Loring: *Idem.*, 305.

[7] J. Bardet: Extraction du germanium des eaux de Vichy. *Comptes Rendus* (1914), **158**, 1278–80.

[8] V. Vernadskii: *Bulletin Académie de St. Petersburg* (1911), 187–93.

net has given the cadmium content in two blendes from Missouri[9] as 0.509 per cent. in the ore from Porter Diggings, Joplin, and 0.723 per cent. in that from Leadville, southwest Missouri. The only other published results (to my knowledge) on analyses of Missouri zinc ores are those of Spencer,[10] of which the following is an abstract.

	Zinc, Per cent.	Lead, Per Cent.	Iron, Per Cent.	Cadmium, Per Cent.
Arkansas Belle	62.045	0.382	0.194	0.031
Gordon Allen	56.089	0.371	0.756	0.024
Empire	64.044	....	1.002	0.052
Stevenson and Wampler	58.308	0.840	1.341	0.092
Magnet	60.068	1.674	0.137	0.132
Newell	55.097	1.238	3.588	1.348
Constant Mining Co.	59.692	1.800	1.884	0.612
North Star	60.855	0.102	1.495	0.131

These are the only figures that are accessible; Jensch,[11] who gives analyses of ores from even minor European zinc districts, includes none from America except mention of one from Eaton, N. H.

Analyses of spelter seem not to find their way into the literature, although a number of zinc laboratories make such analyses regularly. Steger[12] gives many complete analyses of spelter but, curiously enough, the cadmium content of the one Missouri spelter there quoted is not given among the seven constituents other than zinc. Suppan[13] gives the following for the cadmium percentages in spelter: 0.0011 per cent. from Nevada, Mo.; 0.0056 per cent. from Glendale Zinc Works, St. Louis; 0.0188 per cent. from La Salle, Ill.; six other American samples contained only traces, and two foreign spelters gave 0.0245 per cent. and 0.054 per cent. cadmium respectively.

Finally, it may not be amiss to point out the strange coincidence in the occurrence of cadmium in the Missouri zinc ores and the almost unique position of salts of cadmium as addition agents in flotation ex-

[9] *Report of the Geological Survey of Missouri for* 1873–74, 392, in connection with Dr. Adolf Schmidt's study of the lead and zinc regions of Missouri.

See also Dr. Adolf Schmidt: *Die Blei-und Zinklagerstaetten von Suedwest Missouri,* Heidelberg, 1876.

[10] H. G. Spencer: Composition of Cleaned Ores of Southwest Missouri. *Bulletin of the Missouri Mining Club* (Rolla) (June, 1895), **1**, No. 2, 51.

[11] Edmund Jensch: Das Cadmium, sein Vorkommen, seine Darstellung und Verwendung, in Ahrens' *Sammlung chemischer und chemisch-technischer Vortraege* (1898), **3**, 207–32, Ferdinand Enke, Stuttgart.

[12] Victor Steger: Verdichtung der Metalldaempfe in Zinkhuetten, in Ahrens' *Sammlung chemischer und Chemisch-technischer Vortraege* (1896), **1**, 47–88, Ferdinand Enke, Stuttgart.

[13] L. R. A. Suppan: American and Foreign Spelters. *Bulletin of the Missouri Mining Club* (Rolla) (June, 1895).

periments,[14] a discovery the details of which will be published shortly in a Technical Bulletin from the Experiment Station at the Missouri School of Mines. Even should this triple coincidence—namely, the occurrence of cadmium in Joplin zinc ores, the difficulty in flotation of those ores, and the singular behavior of cadmium in small quantities on flotation—turn out to be only fortuitous, the action of the cadmium must still be acknowledged as a striking property of these compounds and as a fact to be considered in any theory of the flotation of heavy sulphides.

In conclusion, now that Mr. Waring has again directed attention to the cadmium in these large Missouri deposits, a word concerning the economic significance of this metal seems warranted. I refer to the possibility that the lack of a market for cadmium rests less on the rarity of that metal than on our ignorance of the properties of the metal, its alloys and compounds, and that an investigation, with a view to the commerical exploitation of cadmium products, would be both timely and of industrial interest.

J. W. Richards, South Bethlehem, Pa.—The analyses show that the dark-brown rosin blende contains frequently less iron than the white jack. I have been guilty of teaching my students that the dark color of the blende is due to intermingled iron sulphide; it now appears that this is not the case, for eight out of 10 analyses of dark-brown blende show less iron than a sample of the white jack. Neither can it be due to lead sulphide, because one of these samples of dark-brown rosin blende is low in lead as well as in iron. The essential reason for the color of the dark-brown rosin blende appears to be still undetermined.

The Chairman (G. C. Stone, New York, N. Y.).—While for the Joplin ores, Mr. Waring has shown that the iron is not in proportion to the depth of color, for the ores of the whole country, the darker blende, as a rule, contains more iron. We find that in the West what they call the "black blendes," the very dark ones, are quite strongly magnetic and can be lifted by the magnet, while the light-colored blendes do not contain so much iron and cannot be lifted.

W. Geo. Waring (written discussion*).—The iron in the Joplin concentrate is in part the result of abrasion in grinding, and in part due to intermingled marcasite. It therefore bears no relation to the color of the zinc-blende.

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[14] M. H. Thornberry: The Selective Action of Cadmium Salts on Lead and Zinc Sulphides in Flotation, in the program of the Metallurgical Symposium at the 55th meeting of the American Chemical Society. *Journal of Industrial and Engineering Chemistry* (1917), **9**, 986.

* Received Jan. 29, 1918.

The dark colored "black jack" of the Rocky Mountain ores—so-called marmatite—is not found here.

The Missouri sphalerite varies from very light straw color, through all shades of yellow and brown, to ruby red; the streak, from light straw to orange yellow.

I have seen somewhere the suggestion that the dark brown color may be due to a hydrocarbon, but I think this is only a conjecture based upon the rather common occurrence of bituminous pitch in the ore deposits of some mines. Very little is really known about the origin of the characteristic colors of zinc-blende. I hope before long to investigate the subject, working upon clean mineral.

Development and Underground Mining Practice in the Joplin District

BY H. I. YOUNG,* CARTERVILLE, MO.

(St. Louis Meeting, October, 1917)

INASMUCH as there has been a great deal of activity in this district recently, a paper of this kind should treat of all the various phases of mining, namely, prospecting, developing, and operating.

Prospecting

Where ore is found at a depth of 100 ft. (30 m.) or more, the prospect work is done by churn drilling. Several makes of churn drills are used in this work, both steam- and gasoline-driven machines being used. The drilling is usually started with a 6-in. bit, and the size of the bit is reduced, if necessary, as the hole is advanced. It is general practice to drill from 100- to 400-ft. holes, but this is governed entirely by the location and formation of the ground. In drilling sheet ground, close spacing of drill holes is unnecessary, but in narrow orebodies, the holes are spaced from 15 to 50 ft. (4.6 to 15 m.) apart. Ten years ago, very few assays were kept of the drilling, but during recent years records have been kept very accurately, showing the location, elevation, assays, and also the vertical section of the hole.

After the orebody is reached, cuttings are taken every 2 or 3 ft. (0.6 to 0.9 m.) samples being sometimes taken by pouring the water off the cuttings and taking a portion of the coarse material for assay. It has been found, however, that very often the sludge material poured off with the water contains high mineral values. To overcome this, cuttings from each 2 or 3 ft. are put in a container, and, after the water is evaporated, sample for assay is taken by means of Jones samplers.

Permanent records of cuttings showing various formations are kept in glass jars or glued on large cards. Drilling at present is costing from $1.25 to $1.50 per foot. In pre-war times, the drilling costs ran from 75c. to $1 per foot.

Development

After proving property by drilling, shafts are put down to the orebody. These are usually vertical shafts, either single or double compart-

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ment. The dimensions of the one-compartment shafts vary usually from 4 by 5 ft. (1.22 by 1.52 m.) to 5 by 7 ft. (1.52 by 2.2 m.) in the clear, while those of the two-compartment shafts run from 5 by 10 ft. to 7 by 12 ft. in the clear. Recently a few three-compartment shafts have been sunk. The shafts are usually put down near the edge of the orebody, so that very little drifting is required to open up the ground for operating. The shafts are usually cribbed with 2 by 4-in. or 2 by 6-in. (50.8 by 101.6 mm. or 50.8 by 152.4 mm.) sawed timbers (very few square sets being used because of the nature of the ground), and extend several feet below the top of the solid rock. The cribbing is lined or laced vertically with 1-in. boards, which help to make the shaft free from small rocks and water, and permit faster hoisting. The cost of sinking shafts varies according to the nature of the ground, the amount of water encountered, and the size and depth of the shaft. The average cost of a 5 by 7-ft. shaft, at present, where from 100 to 500 gal. (379 to 1893 l.) of water per minute is handled, to a depth of from 200 to 250 ft. (60.9 to 76 m.) is from $25 to $30 per foot. Usually two or more shafts are sunk on each operation. These shafts are connected with an air drift 5 by 7 ft. which gives good ventilation. The cost of driving a drift varies from $6 to $9 per foot.

In the Granby, Aurora, and Galena camps, where ore is found at from 40 to 100 ft. (12 to 30.5 m.) the ordinary practice of developing ground is by sinking small shafts; very little drilling is done in proving up the shallow orebodies.

A good way to increase tonnage where working faces are limited is by driving a 7 by 8-ft. drift ahead of the working faces to a distance of 300 to 600 ft. and opening up the ground so as to permit the use of several machines. Drill holes are usually necessary for proper ventilation where this method is used.

Operating

General Practice

The methods employed in the underground work vary with the nature of the deposits, and have been developed to suit the local conditions encountered in the various mines. The ore deposits lie in a horizontal plane, and are usually found on one level, varying in heights from 8 to 40 ft. (2.5 to 12 m.). In some parts of the district, however, two distinct levels are found and in a few instances as many as four separate ore strata have been found, these being separated by a relatively thick stratum of hard, barren rock, and the several runs are mined independently of each other.

In the mines where the orebody is higher than 16 ft. (4.9 m.) "underhand stoping" is practiced, especially in the hard-ore sheet-ground mines. Pillars are left at frequent intervals, the distance between pillars and

thickness depending on the nature of the ground and the height of the orebody. The ore is usually found in hard flint formation, known as the "Grand Falls Chert," which lies between beds of limestone. The lime

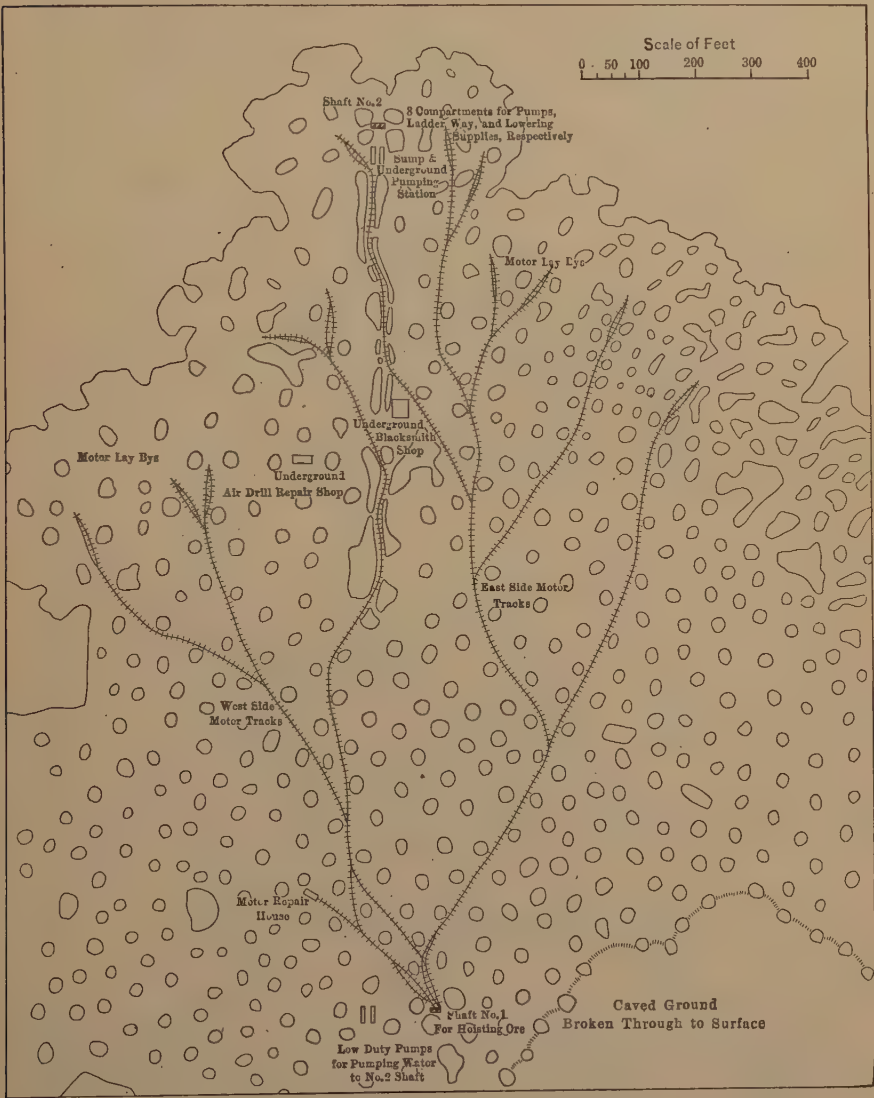

FIG. 1.—UNDERGROUND WORKINGS OF A SHEET-GROUND MINE.

formation is known as the "Mississippian," and is overlain by the "Pennsylvanian" series of shales and sandstones.

A comparatively small amount of timbering is required in the mining operations of the district, as there is a cap rock of flint varying in thick-

ness from 1 to 5 ft. (0.3 to 1.5 m.), which makes a good roof in the majority of mines. In some cases, this flint decomposes into what is commonly known as "cotton rock." This is soft and white, and where it occurs pillars must be placed very closely, and the roof arched. In sheet-ground, from 10 to 20 per cent. of the ore formation is left in the form of pillars. These are sometimes trimmed, and many of them removed after the ore-body has been worked out. Fig. 1 shows the pillar system in a typical sheet-ground mine.

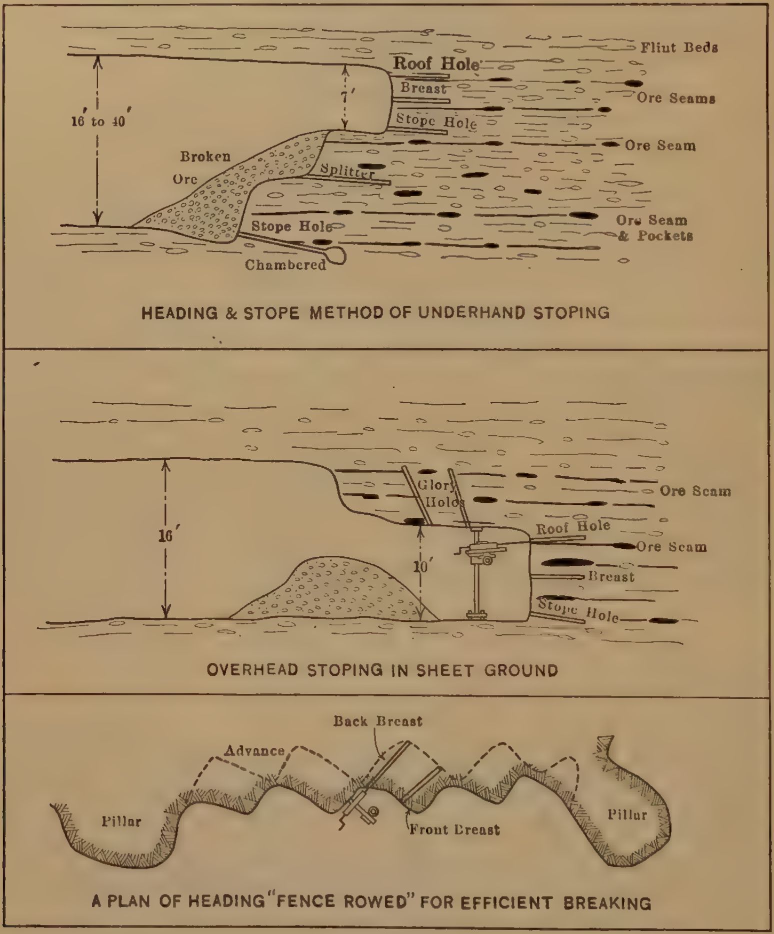

Fig. 2.

Drilling and Breaking

Where the orebody is over 16 ft. in height, it is customary to employ the "heading and stope" method of cutting the ground. The heading is carried on 6- or 7-ft. posts. The face is very irregular, which is necessary

for the best breaking results. From five to seven holes are drilled at each set up, and the heading is usually advanced from 15 to 20 ft. ahead of the stope. If the stope is over 10 ft. in height, a splitter is drilled horizontally so as to relieve the load on the stope hole. In cases of very high stopes, more than one splitter is used. Stope holes are drilled so as to dip below the horizontal in order to maintain a level bottom.

Where the orebodies are less than 16 ft. in thickness, the "heading and overhead stoping" method is profitably employed. A 10-ft. post is used to drive a bottom heading, from five to eight holes being drilled to each round. These holes vary in depth from 10 to 12 ft., and usually from 75 to 150 tons of ore are broken to each round of holes. After the bottom heading is advanced from 10 to 25 ft., holes are drilled in the overhanging ore, and this is blasted down at a very low breaking cost.

A comparison of these two methods is shown in Fig. 2.

Two types of air drills are used in this district, the "piston" and the "hammer" types. In sheet-ground mines, where the laying of dust is an important consideration to the health of the miners, the "hammer" type or water drill has been used with great success, it has also shown an increased drilling efficiency of from 25 to 40 per cent. above the piston machine. Drills are operated by compressed air, at a pressure of from 85 to 100 lb. per square inch.

High-carbon hollow drill steel is used with hammer drills. Holes are started with a 2¾-in. bit and finished with 1¼-in. It is necessary to change the size of gage in each 2 ft. drilled, and it is sometimes necessary to use several steels to drill 2 ft., the ground being exceedingly hard. The drill steel is resharpened at an underground blacksmith shop, or on the surface where oil forges have been found to give the best results. Because of the large amount of steel used, it is absolutely necessary to sharpen the steel by mechanical sharpeners.

Drilling costs vary with the character of ore mined. The tabulation in Table 1 shows costs for 4 months of 1917, in a mine where the faces are from 10 to 16 ft. in height.

Table 1

Tonnage broken 136,272

	Amount	Cost Per Ton Ore
Machine men	$9,630.40	$0.0706
Machine helpers	7,940.45	0.0583
Drill repairs	3,505.14	0.0257
Drill steel	2,627.29	0.0193
Sharpening drill steel	3,194.86	0.0234
Oil for machines	207.78	0.0015
Compressed air	6,168.63	0.0453
Air hose and fittings	703.43	0.0052
Total	$33,977.98	$0.2493

Blasting

After drilling, the holes are chambered and prepared for powder. Both ammonia and gelatin dynamite of a strength of from 30 to 40 per cent. are used, in the form of sticks of from 1 to 1¼ in. diameter. No. 8 caps have been found to give the best results. The powder crew in mines where hoisting is carried on double shifts do their work on the second shift, and fire shots when the shift goes off; this gives the mine several hours to be cleared of powder smoke and gases. It is necessary to have skilled labor for this work, both on account of the high cost of explosives, and for the safety of the men. The holes are loaded with charges varying from 25 to 150 lb., depending upon the formation of the ground, and the burden of the hole.

Breaking, or blasting, costs are from 15 to 30 c. per ton, varying with different heights of working faces.

The costs shown in Table 2 are for a 12-ft. working face.

TABLE 2

Tonnage.................................136,272

	Amount	Cost Per Ton Ore
Labor	$4,006.20	$0.0294
Powder	17,914.00	0.1314
Fuse	782.33	0.0057
Caps	217.62	0.0016
Total	$22,920.15	$0.1681

These costs are on powder purchased on contract, the present open market price being 70 per cent. higher.

Shoveling

After the ore is broken it is sometimes necessary to drill large boulders with a jackhamer, and blast them, to reduce them to shoveling size. The usual practice of the district is to shovel into tubs, which are commonly known as "cans," holding from 800 to 1200 lb. (362.9 to 544 kg.) of ore. These are usually 30 by 30 in., 30 by 32 in., or 32 by 32 in., and are trammed to shaft on small trucks. The shovelers use short-handled No. 2 scoops and an average of 20 tons per shoveler per shift is obtained. All shoveling is done on a contract basis, and shovelers earn from $3 to $7 per 8-hr. shift. Both the Thew steam shovel and the Myers-Whaley shoveling machine have been used for underground shoveling to good advantage, but without a decrease in the shoveling cost. However, if labor continues to become scarcer, it may be necessary to adopt this method to insure a steady production.

A number of the mines have adopted the use of cars to take the place of "cans." These hold from 1500 to 2000 lb. (680 to 907 kg.) of ore.

The cars are easier to shovel into, as they are lower than the "cans." They are also more stable, which permits faster tramming.

Shoveling costs vary from 20 to 25 c. per ton of ore.

Tramming

Several different methods of tramming are employed in this field, namely, hand tramming, mule haulage, motor haulage, and rope haulage, depending on the length of the haul, tonnage required, and the grade encountered.

The tracks are from 15 to 24-in. gage, the steel varying in weight from 12 to 30 lb. (5.5 to 13.6 kg.) per yard.

The trucks are either wooden or steel frame mounted on plain wheel and axle, or roller-bearing. The roller-bearing truck has been found to make a saving in tramming expense.

Where the haul is 200 ft. (61 m.) or less, the shovelers usually run their cans directly to the shaft, but for any distance above this and up to 400 ft., additional tramming labor is required to take the dirt from the shovelers' lay-by to shaft. Where a large tonnage is required, as is always the case in sheet-ground mines, and the distance from the shaft to the working faces is from 400 to 1500 ft., mule haulage is very successful. A mule will haul 100 tons 1500 ft. in an 8-hr. shift, at a cost of from 5 to 6 c. per ton.

In the large sheet-ground mines, where approximately 1000 tons are hoisted in an 8-hr. shift, and all of this ore must be hauled 2000 ft. or more, motor haulage is necessary. The 6-ton gasoline locomotive is the most popular and gives no trouble with regard to ventilation. Roller-bearing cars are required for long hauls.

Rope haulage is used in some mines where the grade is very severe.

The costs in Table 3 show comparison of gasoline locomotive and mule haulage for 4 months of 1917. The distance on locomotive haulage is 1750 ft.; on mule haulage, 700 ft.

Table 3

Gasoline-locomotive Haulage			Mule Haulage		
Tons hauled..............136,272			Tons Hauled............81,145		
	Amount	Cost Per Ton		Amount	Cost Per Ton
Engineer...........	$1,883.62	0.0138	Mule driver........	1,868.70	0.0230
Brakeman,..........	1,659.12	0.0122	Maintenance mules	406.68	0.0051
Oil for locomotive...	239.27	0.0018	Car couplers.......	964.56	0.0119
Gasoline for locomotive...........	1,458.10	0.0107	Car greaser........	470.37	0.0058
Repairs............	2,063.39	0.0151			
Car couplers........	2,241.65	0.0164			
Total............	$9,545.15	$0.0700		$3,710.31	$0.0458

Hoisting

Three systems of hoisting are used in this district; cans, cages, and skips. The can is the best adapted to this depth of hoisting, and the tonnage requirement of the average Joplin mill. Both steam and electric hoists are used. The hoisting cycle is as follows: A trammer or bumper runs a loaded can to the side of the platform, and the hoisting cable is unhooked from the empty can on the platform and attached to the load by the tub-hooker; this operation is done very quickly and without any signals to the hoisterman, who now picks up the loaded can, which swings in over the platform where the tub-hooker steadies it in the center of the shaft. This is raised to the top of the derrick, where the hoist is located. Here the hoisterman hooks a tail rope, which is fastened out in front of the shieve timbers, to a ring in the bottom of the can. The hoisting cable is slacked, and weight is taken by the tail rope, causing can to dump directly into the mill hopper. The hoisting cable is now taken up, and the tail rope unfastened so as to allow the can to descend the shaft. With practice, a hoisterman and tub-hooker will become so expert at their respective operations that as many as 150 cans an hour can be hoisted from a depth of 250 ft. The record of a single compartment shaft for the district is 1071 cans in an 8-hr. shift. Where two compartments are used for hoisting, the hoists are placed side by side in the top of the derrick, and the shaft is divided for its entire depth to prevent cans from bumping.

In order to do all the shoveling on the day shift, and to secure a large tonnage, many of the operators hoist from several shafts, and tram the ore on the surface to the central mill hopper by means of inclined surface trams; some few aerial trams are used, but the level topography and short hauls have made the inclined tram nearly universal in this district.

The cost of hoisting is between 6 and 7 c. per ton, while surface tramming costs 1 c. per ton.

Table 4 shows a distribution of hoisting cost for 4 months of 1917.

TABLE 4

Tons hoisted........................136,272

	Amount	Cost per Ton Ore
Hoisting engineer	$2,316.07	$0.0170
Tub-hooker	2,115.57	0.0155
Cable	720.50	0.0052
Hoister repairs	429.92	0.0033
Hoister derrick repairs	110.85	0.0008
Shaft lacing repairs	99.16	0.0007
Steam for hoister	2,744.92	0.0202
Slickers for hooker	140.30	0.0010
Oil and waste	73.16	0.0005
Total	$8,750.45	$0.0642

Pumping

The serious problem of pumping in the Joplin mines is not the amount of water to be handled, but, in many instances, the acid character of the mine water. It is seldom that over 1000 gal. (3785 l.) per minute has to be pumped at any one mine. In the sheet-ground camp, the mines are so cut together that many of them do no pumping at all; where this is the case, a central drainage company is formed, to do the pumping for all the operators benefited.

Many types of centrifugal pumps are used underground; the Pamoma and Texas are favorite surface installations, the Texas being used very generally over the district for unwatering mines, while the large-geared duplex or triplex pumps are usually found in the large permanent underground pumping stations.

Costs

There is no uniform system of cost accounting among the mines of this district. At present, operating costs, which include mining, milling, pumping, miscellaneous, and administrative, are from $1.20 to $1.75 per ton of ore mined. On account of high prices of supplies and the scarcity and inefficiency of labor, costs are gradually increasing throughout the district.

Table 5 shows a distribution of mining costs over a period of 4 years, 2 before-war years and 2 during the war.

TABLE 5.—*Underground Mining Costs*

	1912		1913		1915		1916	
Tons Hoisted	306,263		303,331		387,436		374,937	
	Amount	Per Ton Ore	Amount	Per Ton Ore	Amount	Per Ton Ore	Amount	Per Ton Ore
Ground foremen......	$3,102.50	$0.0101	$2,942.43	$0.0097	$4,468.99	$0.0115	$4,462.99	$0.0119
Drilling.......	60,886.71	0.1988	53,671.22	0.1769	99,824.46	0.2576	103,800.20	0.2769
Blasting......	50,762.53	0.1657	49,371.41	0.1628	71,469.12	0.1845	67,040.58	0.1788
Roof protection.....	1,857.26	0.0061	1,341.80	0.0044	6,169.85	0.0159	6,817.19	0.0182
Shoveling.....	47,671.59	0.1557	44,658.31	0.1472	81,585.48	0.2106	87,756.52	0.2341
Conveying to shaft........	24,434.08	0.0798	27,880.36	0.0919	42,301.72	0.1092	53,045.54	0.1415
Hoisting......	14,340.92	0.0468	13,118.55	0.0433	15,446.79	0.0399	18,126.29	0.0483
Lighting......	1,767.64	0.0058	2,113.28	0.0070	2,389.49	0.0062	1,929.96	0.0051
Miscellaneous.	3,991.05	0.0130	2,312.82	0.0076	13,143.44	0.0339	21,654.02	0.0577
Betterments...	65.43	0.0002	....	....	2,555.54	0.0066	4,987.64	0.0133
Total.......	$208,879.71	$0.6820	$197,410.18	$0.6508	$339,354.88	$0.8759	$369,620.93	$0.9858

Discussion

F. W. Sperr, Houghton, Mich.—Why does 10 to 20 per cent. of the ore formation have to be lost? It is not clear to me why it should not all be taken out, from the conditions as I saw them underground this afternoon.

H. I. Young.—The grade of our ore is so low that we cannot stand the expense of timbering to support the roof, which, in a sheet-ground mine, would be from 25 to 50 per cent. of our total mining cost at present. After the orebody is mined out, we can often recover 10 to 20 per cent. of the material that is left.

F. W. Sperr.—I think it should be a very simple mining engineering problem to mine all of that mineral without the use of a stick of timber. Similar work has been done in the mines of Grand Rapids Plaster Co. The usual operation in deposits of this kind is to open up and excavate as far as the roof will stand; when it caves, start in and dig somewhere else. But in the case I referred to they learned that by working the ground more systematically and leaving in a sufficient amount of pillar, they could later draw those pillars and get them all, and avoid uncontrolled breakages which would be likely to occur if they tried to hold up the overburden permanently. From the point of view of safety and also from the financial point of view, I contend that it is better to keep the caving under control and make it do what you want it to do, rather than to take out what you can conveniently for the time being, and after a while be compelled to let the overburden do what it pleases.

J. A. Ede, La Salle, Ill.—Do you refer to any particular mine of the district or to the district generally?

F. W. Sperr.—To the district generally.

J. A. Ede.—The mine we have visited cannot be considered as typical of the Joplin district, although a very good example of local practice. The ore of the district is found in various forms and the *modus operandi* is subject to the nature of the occurrence. There are at least three, each of which requires a different method of mining. The disseminated ore of the sheet-ground, the channels or furrows arrangement, and the circular or elliptical forms, commonly designated "basins." The successful exploitation of the deposit will depend largely on the extent to which a clear conception has been acquired as to the class to which the ore in question belongs. At the present time this matter is being considered very carefully; a system of "blocking out" is practised, the outline and value of the deposit being ascertained by drilling before a shaft is sunk and the system of development determined.

I was rather favorably impressed with the condition and management of the Davy mine. The pillars left to hold the ground contain a high

percentage of zinc, but the ore can be all recovered when the limit of the lease or boundary has been reached. It is quite possible that some mines in the district could use timber to advantage, as suggested by Prof. Sperr. Its usefulness, however, I think, would be restricted to a few exceptional cases and to protect a local rather than a general condition. It would be well to keep in mind Prof. Sperr's suggestion for substituting timber for pillars.

I would like to ask Mr. Young again to give the cost of haulage. What difference has he ascertained between the cost of mule and electrical haulage? Between the cost of electricity and gas? Has he had any experience with a battery motor?

H. I. Young.—In 1912 our haulage was changed from mule to gasoline motor. Mule haulage was costing us 7½ to 9 c. per ton of rock; we reduced the cost to 3½ to 4½ c. a ton by substituting gasoline, but at present it would be absolutely impractical to haul our tonnage the required distance by mules. We regulate the spacing and number of pillars according to the formation of our ground. On the east side of the mine the pillars are spaced much closer than on the west side, as on the west side the ground is much harder. The space marked "caved ground" (p. 673) is where we did trim pillars to about 25 or 30 per cent. of their original size, and as a result a number of pillars were crushed to pieces before they had been reached, so that the size of each pillar is regulated according to the formation of the ground being mined.

In regard to trimming and removing pillars in the Joplin district, it has been suggested that we practise a longwall system or a modification of one of the numerous caving systems used on the iron ranges. One great objection to either of these systems is that our roof is of such nature that it will not come evenly or gradually; in many cases it will hang for a long period until sufficiently oxidized and then it will come with a rush over a large area. For this reason it is difficult to maintain the desired control unless artificial means are used, such as breaking the roof with explosives. This would be prohibitive on account of the cost of explosives, and because we have only 16 or 18 ft. depth of ore over which to distribute this first cost, as compared with several hundred feet in the iron mines.

Then there is the additional objection that in many cases our surface is too valaube to cave, and in many instances it has been proved that our ground does not arch well and is very likely to cave to the surface.

Oxide of Zinc

BY GEORGE C. STONE,* PH. B., NEW YORK, N. Y.

(St. Louis Meeting, October, 1917)

The method of making oxide of zinc direct from the ore was invented and developed at the works of The New Jersey Zinc Co. at Newark in the middle of the last century. The process was invented by Burrows, who had not the ability, financial or technical, to work out the details necessary to make it of commercial value. This was done by Col. Wetherill, whose name is commonly attached to this process. The grate bars used are also frequently called Wetherill grates, which is a misnomer because they were in common use for boiler firing at the time, and he never claimed their invention.

The invention of the process was due to the efforts of The New Jersey Zinc Co. to find a profitable means of working the ores from Franklin and Sterling Hill, N. J. These are a mixture of franklinite, willemite, and zincite, containing about 20 per cent. of zinc. The first attempts were to make spelter, which were not successful, owing to the low grade of the ore and the fusibility of the residue. Failing in this, the next attempt was to make oxide in large muffles and reverberatory furnaces. This succeeded, although the cost of operation was high, the recovery low, and the quality of the product uncertain. In 1855, the new process was patented and has been in successful operation ever since.

Essentially, the process is to spread a mixture of coal and ore on a body of burning coal on a perforated grate, and blow an excess of air through the grate. The zinc is reduced in contact with the coal, volatilized and burned by the excess of air in the upper part of the furnace and in the flues. It is then carried to the bag rooms by the excess air and products of combustion which are forced through the flues by fans. In its main features, the process is the same today as at the time of its invention, but the details have been so modified that it would hardly be recognized by its originators.

Ores

The process is applicable to all the oxidized ores of zinc and to roasted sulphides, provided the gangue is not so fusible as to leave a residue that is impervious to the blast. In many cases the ores contain impurities that make it impossible to produce an oxide of a good enough color to be

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available as a pigment. These impurities are any volatile metals that form colored oxides or sulphides. Cadmium is one of the worst, because it is very volatile and its dark brown oxide and bright yellow sulphide both have strong tinctorial powers, so that small fractions of a per cent. seriously injure the color of the oxide. Lead, which is one of the commonest impurities in zinc ores, also injures the color, though to a much less degree, as it usually forms basic sulphates which are nearly white. On other accounts lead is often objectionable, particularly when the oxide is to be used for the manufacture of rubber goods. Sulphur may occur in oxide as sulphides, sulphates, or as sulphurous anhydride, the first and last of which are objectionable; the first on account of probable injury to the color, while the last is believed by many paint manufacturers to have a bad effect on the grinding properties. Opinion on the latter point is by no means unanimous. Sulphates, if soluble, have very objectionable qualities for outside paints as they leach out on the paint coat, leaving discolored spots. Chlorides, in appreciable quantity, are rare, but if present would be open to the same objection.

Fuel

The fuel used must be one that does not give a black smoke, which would ruin the color of the oxide. Anthracite is the most satisfactory fuel for the purpose and the one generally used. In the West, where anthracite is expensive, semi-bituminous coals, low in volatile qualities, are sometimes substituted for it in whole or in part, but they are never as good. Coke alone does not work well, as it causes a very intense local heat that makes the residue impervious to the blast. It is occasionally used as a part of the charge fuel.

Fluxes

With siliceous ores, limestone is sometimes added, but in general it does more harm than good, owing to its tendency to make the residue too fusible.

Preparation of Charge

The ores and coal are usually received at the works in fine enough condition for use without further crushing. Nearly all the ores are either concentrates or roasted products and are inevitably finer than is necessary. As the small sizes of coal are suitable for the purpose, and are cheaper than the large, they are invariably used. If fluxes are used they are crushed to about the size of the ore and coal. There should not be a great difference in the size of the different materials, or it will be impossible to mix them properly.

Palmerton Plants

The Palmerton plants of The New Jersey Zinc Co. (of Pennsylvania), are the largest and best equipped oxide plants in the world.

Fig. 1 shows the general arrangement of the West plant and Fig. 2, that of the East. They are similar, but the East plant has fewer and larger bag rooms and differs somewhat in details. At both plants the raw materials are delivered on the trestle at the south of the plant and are either placed in bins or stocked in piles. The ore and coal are taken from the bins to the mix-house by larries The mixed charges are delivered to the bins above the furnaces and distributed by traveling cranes.

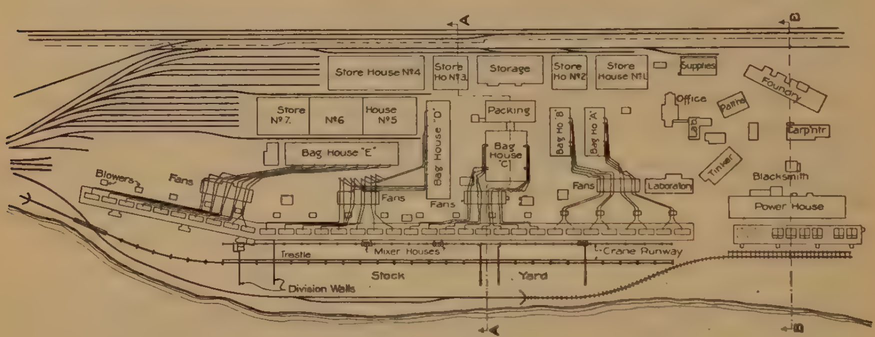

Fig. 1.—Plan of west plant.

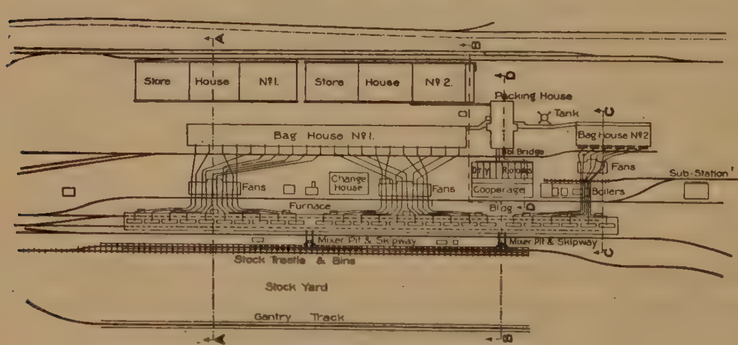

Fig. 2.—Plan of east plant.

Each block of furnaces is a separate unit, having its own blower, exhaust fan and bag room. The oxide is trucked from the bag rooms to the packing room and the final packages skidded or trucked to the storehouses.

Stocking and Mixing—East Plant

The ore and coal are delivered on the double track, steel and concrete trestle, 22 ft. (6.7 m.) high, with Brown tangential bins below the track next the furnace room. Materials for current use are dumped

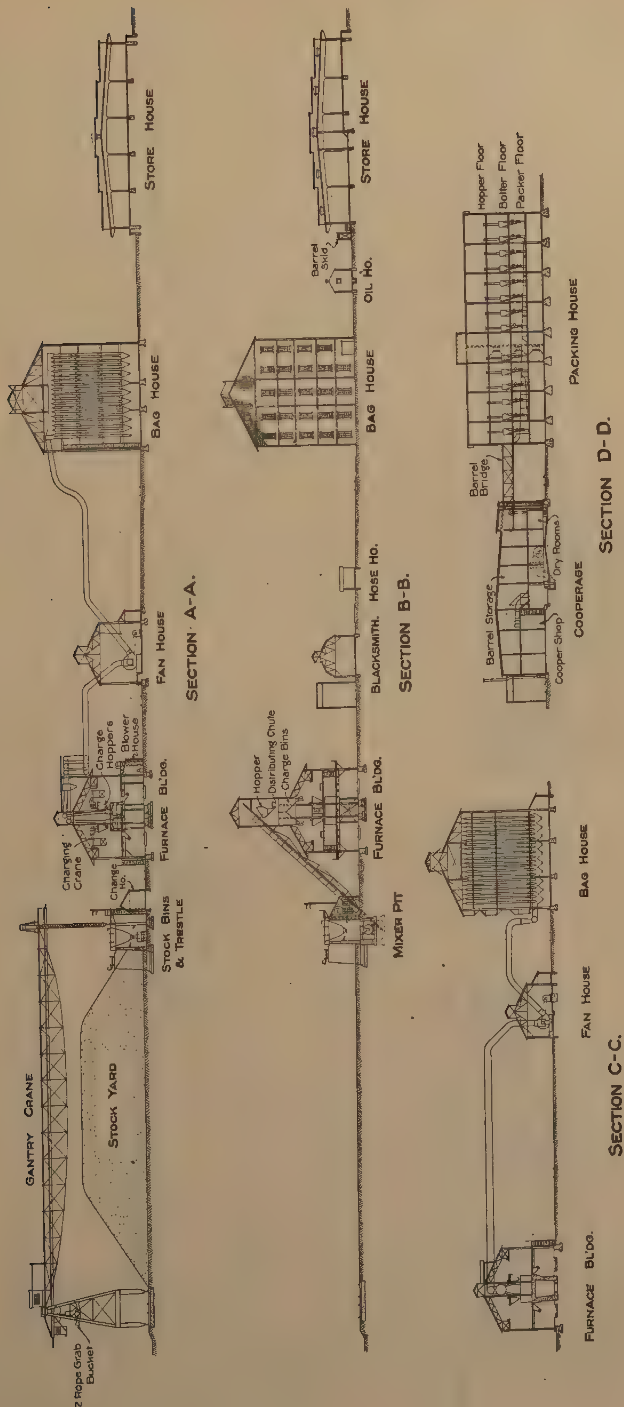

FIG. 3.—SECTIONS OF EAST PLANT.

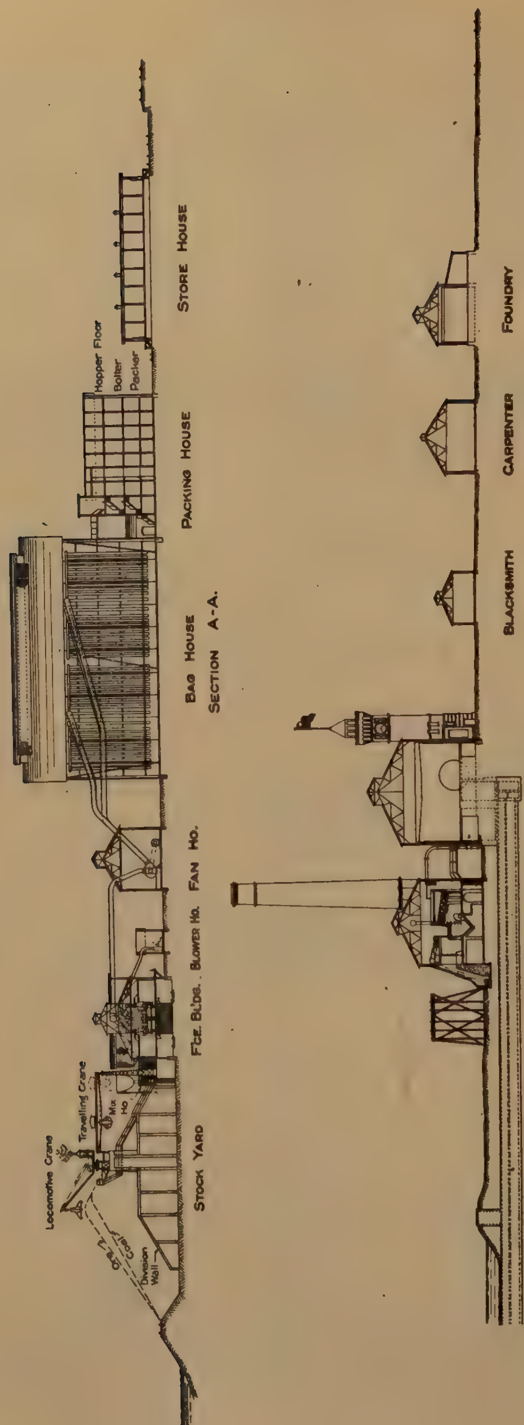

Fig. 4.—Sections of West Plant.

directly into the bins and the surplus is stocked back of the trestle by a bridge crane of 233 ft. 9 in. (71.25 m.) span equipped with a 10-ton Brown grab-bucket. This equipment also returns materials from the stock piles to the bins when necessary. The charges are drawn from the bins to a weighing larry which has two hoppers, each containing a furnace charge. These are carried to Ransome concrete mixers placed at one end of the furnace room. The mixed charges are elevated by a pair of counterbalanced skips to bins above the furnaces. From the bins the mixed charge is taken to the furnaces by cranes, each having three hoppers holding one-third of a furnace charge each. Each hopper is placed on a scale so that the charge is equally divided between the three.

Stocking and Mixing—West Plant

All materials are delivered on a single track steel and concrete trestle 47 ft. (14.4 m.) high, down the center of the storage yard. The materials are taken from this by a bridge crane of 48 ft. 8 ½ in. (14.85 m.) span with a 2-yd. (1.8 m.) bucket and placed in elevated bins next the furnace room. The ore and coal are stocked and reclaimed by the traveling bridge and by locomotive cranes on the high trestle. The total storage capacity is 200,000 tons of ore and 175,000 tons of coal. The fuel and the various ores are taken by bridge cranes to small bins placed over the mixers. The materials are dropped from these bins into weighing hoppers and from these into the Ransome mixers. The mixed charges are elevated to bins over the furnaces and then distributed by cranes much as at the East plant, except that there are no scales on the cranes or larries. This arrangement of doing all of the weighing at the mixer on a fixed hopper is more satisfactory than weighing larries, because the latter are very difficult to keep in adjustment.

Furnace Rooms

The furnace rooms at the two plants are quite similar, the main differences are that at the West plant the furnaces are carried on a concrete foundation, and at the East plant, on the floor. In both cases the furnaces are elevated sufficiently to allow the residue to be dropped into hoppers below the floor and then into cars. The framing of the buildings is quite different, the East plant (the later one) being arranged to give more room for the charge cranes. The West plant furnace room contains 34 blocks of furnaces and the East plant, 26. All the blocks are alike, consisting of four furnaces 19 ft. 6½ in. by 5 ft. 11½ in. (5.96 by 1.82 m.) (Fig. 5). Each furnace has independent blast and flue connections and three charge openings in the roof surmounted by hoppers. Each furnace works three charges in 24 hr. The different furnaces of the

block are charged alternately at 2-hr. intervals in regular order. Charging in this way, there is always one furnace starting, one nearly finishing, and two working strongly, and the volume and temperatures of the gases to be handled are nearly uniform at all times.

Furnace Operation

When the furnace is worked off, the dampers in the flues are closed, cutting it off from the bag room. The blast is shut off and the working doors taken down. Three men work on the furnace at once, each at a separate door. The loose material on top of the charge is raked off and dropped on the floor close to the furnace. The clinker is then broken up by heavy slice bars and raked into hoppers under the floor.

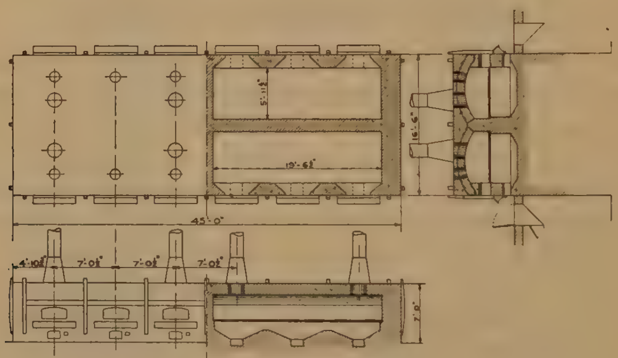

FIG. 5.—FURNACES.

The furnacemen then thoroughly clean from the side and back walls any clinker that has fused to them. When all of the old charge has been removed and the furnace cleaned, the grate is covered as rapidly as possible with a thin layer of coal, the doors closed and a light blast turned on. The coal is lighted by the radiation from the hot arch and it is important that it should light quickly and evenly. When the bed coal is burning brightly, the charge from the hopper is dropped on to it and leveled. The doors are now closed, the blast turned on, and the furnace left to itself with occasional inspection to stop any blow holes that may form. With the large charges that are now worked, it is considerable time before any zinc vapor appears. When it does, the dampers in the flues are reversed, sending the gases and oxide to the bag room.

Fans and Piping

The blowers for the oxide furnaces are direct-connected to motors, the wheels of the blower being placed on extensions of the motor shaft. They are located in small houses outside the furnace room, two in each house.

Each furnace has two outlet pipes, usually 20 to 22 in. (0.5 to 0.56 m.) in diameter, which lead up to the drum pipe overhead. The drum has an outlet in the middle of its length leading to the exhaust fans. Formerly the gases passed through a brick settling tower between the furnaces and exhaust fan, but this is now usually omitted as unnecessary. The exhaust fans are extra heavy plate fans with special Hyatt roller bearings and water-cooled shafts, belted to motors. Aside from the shafts, bearings, and extra thickness of plate, they are of the ordinary types. For convenience in attendance, a number are placed in one building. The West plant has four fan rooms, three containing eight fans each and one containing ten fans. At the East plant there are two with ten fans each and one with six. From the exhaust fans to the bag rooms, in each case, run round pipes 3 ft. (0.92 m.) in diameter, made of No. 16 sheet iron and provided with numerous clean-out openings. In all cases, the pipes are arranged to give as nearly as possible the same length from furnaces to bag rooms. This is done partly to equalize the friction in the different lines, but mainly to secure proper cooling of the gases before reaching the bag room.

Bag Rooms

The method of collecting oxide in bags was first patented by S. T. Jones in 1852 and has been patented by others many times since. The bag rooms at Palmerton vary considerably in size; the first two built holding bags enough for four blocks, while the latest contains the bags for twenty. The larger rooms are more economical in labor and do not cause any inconvenience, provided the ventilation is equally efficient. The bags used are all of heavy, closely woven cotton sheeting 22 in. (0.56 m.) in diameter and about 40 ft. (12 m.) long. In the majority of cases the gases enter a header at the top of the room, and from there pass to a series of parallel pipes, from the bottom of which the bags are connected to hoppers, four bags to each, with a single collecting bag tied to an outlet at the bottom of each hopper. At intervals the gases are shut off from the two lines of pipe connecting with a single line of hoppers, and the bags shaken. This pair of pipes is again connected to the main header and the next two lines shut off and shaken. The frequency with which the bags are shaken depends upon the relative area of grate and bags, and the amount and quality of the oxide being made. The proper ratio of grate surface to bag surface has been a subject of much discussion. It varies in practice between 1 : 80 and 1 : 130. The amount actually necessary depends mainly on the efficiency of the ventilation of the buildings and the frequency of shaking. A well-ventilated room will consume no more muslin with a ratio of 1 : 80 than a poorly ventilated one with a ratio of 1 : 130. Recent experiments indicate the probability that a well

ventilated bag room, with a mechanical shaker operating at short intervals, will allow of a very great reduction in the necessary bag area.

The collecting bags are removed from the hoppers every 24 hr., or oftener if necessary, and taken to the packing room. This is done by a trucking gang that removes the filled bags and replaces them with fresh ones, trucking them by hand to one side of the bag room, where they are coupled together, and the train drawn to the packing room by a motor.

Packing

The packing room is a four-story building of mill construction. The trucks are hauled into the ground floor and then taken one by one on a platform elevator to the upper floor. There are a number of openings in this floor leading to bolters on the floor below. Around each opening is a wide shelf on which the bags are rested while their contents are emptied into the bolters. The third floor contains nothing but the bolters, and a room for sewing and making bags. The bolters used are of the ordinary types used in flour mills. The packing machines are arranged in four lines on the second floor. They are also of a type largely used for flour. There is usually one packer below each bolter and fed directly by it. In some cases, owing to the lack of room, one bolter feeds two packers. The oxide is packed partly in paper bags holding 50 lb. (22.7 kg.) each, partly in barrels with 30-in. (0.76-m.) staves, and 19⅛-in. (0.49-m.) heads, holding 300 lb. (136 kg.) each, while export material is packed in special barrels and weights to suit the different markets.

The barrels are kept in a storage room, where the hoops are driven by a machine and then nailed by a second one; and from here they are passed to the packers. All barrels are weighed empty and filled, the weights adjusted and the barrels headed. They are passed to the ends of the building and rolled on skids to the storehouses. The second floor of the packing room is high enough above the ground level to allow sufficient slope to the skids to carry the barrels to all but the most distant storehouses. The paper bags are loaded on trucks and taken to the storehouses on the same skidways.

Cooperage

The cooperage is at the East plant, next the packing room. It is fully equipped with machinery and has a capacity of about 2000 bbl. a day. The barrels are delivered directly from it to the East plant packing room, and by special railroad cars to the West plant storehouse.

The storehouses are all one-story mill-construction buildings with the floor elevated to the level of a box car. Each has doors and platforms on opposite sides, one for receiving material from the packing room and the other for loading cars.

Power Plant

Power for both plants is supplied from a central station at the West plant, between the oxide and the spiegel furnaces. The boiler room is equipped with Edgemoor boilers of 536 hp. each, which can be fired with coal or with the waste gases of the spiegel furnaces. The engine room parallels the boiler room and is located about 20 ft. (6 m.) north of it. It contains three horizontal cross-compound blowing engines for the spiegel furnaces; four direct-current generators, two of 500 and two of 600 kw., each direct-connected to a horizontal cross-compound Corliss engine; three Allis Chalmers turbo-generators, two of 2000 and one of 4000 kw., 6600 volts, and three phases; two motor-generator sets used as a reserve for both alternating- and direct-current lines, as either part can be used as a motor to generate current of the other kind, if needed. In addition, the engine room contains the exciters, condensers, and pumps. The engine room is entirely above ground, giving a light basement for the pipes and elevating the machinery above danger from floods. This also permits a railroad track to enter at the ground level, so that new machinery or repair parts can be brought into the building, and there lifted from the cars and put in place by a traveling crane that covers the entire space and can lift the heaviest piece of any of the machines. The switchboards are in a tower at the north side of the building where they are visible from all parts of the generator floor, but do not project into the room. The tower gives ample room for leads and anchorages of cables.

Ores

The ores used at Palmerton are almost entirely concentrates from the company's mines at Franklin Furnace and at Sterling Hill, N. J., Three of these products are smelted for oxide, franklinite, half/half, and dust. The composition of these is about as follows:

	Franklinite	Half/half	Dust
Zn	18.2	18.4	17.6
Fe	35.0	15.2	20.1
Mn	12.8	12.2	8.9
SiO_2	3.6	12.6	7.2

The franklinite is worked by itself, and the residue smelted in blast furnaces for the production of spiegel. The half/half and dust are mixed and the residue wasted, as it contains too little Fe and Mn and too much SiO_2 to be a profitable smelting material.

Franklinite is the easiest of these ores to work and gives the best product. The half/half and fines are mixed and worked together. The charges per furnace average:

	Ore	Coal
Franklinite	2.62T	1.62T
Half/half and fines	1.95	1.36

The recovery from franklinite averages about 86 per cent. and from the half/half and fines charge about 78 per cent. Both charges and recoveries vary somewhat with changes in the ore and quality of the oxide desired. The recovery and quality of product are controlled by the selection of the ores, ratio between ore and coal, and of both to the grate surface, and what is of great importance, the balance of blast and exhaust and the proper proportioning of both to the charge being worked. Where leaded ores are worked, the charges and recoveries vary much more and average figures are of little value. In general, with such ores the charges are a little lighter and the recoveries lower than for the New Jersey ores. The recoveries of lead are usually rather better than of zinc, roughly as 9 to 8. For estimating purposes the make of oxide is generally taken as equalling the combined $Zn + Pb$ in the ore, the gain in O and SO_3 about balancing the losses in Zn and Pb.

Product

The product is divided into five grades, two for the use of paint manufacturers, two for rubber, and an off-grade that is reworked either for oxide or spelter. For the paint trade, the grading is entirely by color, samples being daily rubbed down with oil and compared with standards. For the rubber trade, slight variations in color do not make so much difference and the grading in this respect is not so close, but freedom from lead and absence of small hard particles is insisted upon.

The variation in composition of the different grades is very slight, the range of composition of all being nearly the same. The material as shipped, uniformly contains 99 per cent. or over ZnO, the principal impurities being: SO_3, 0.25 to 0.33 per cent.; H_2O, 0.10 to 0.50 per cent.; and PbO, 0.05 to 0.30 per cent.

Uses.—The principal use of zinc oxide is as a paint pigment, and for this purpose its purity of color and freedom from discoloration by gases and atmospheric conditions, fineness and uniformity make it particularly useful. As a pigment it has the disadvantage of slow drying and a tendency to become unduly hard in time. The general concensus of opinion among paint manufacturers now is that a mixture of pigments is better than any single one. In these mixtures, zinc oxide is an almost invariable constituent, because it prevents chalking and gives a surface that retains its color.

Next in importance is its use in rubber goods. In the manufacture of such products, zinc oxide is used for two purposes. It is used as a pigment to produce white goods, and is excellent for this purpose, provided it is substantially free from lead. If this were the only effect it had it might be replaced by cheaper materials, but it also greatly increases the tensile strength of the resulting material. A mixture of pure rubber and sulphur vulcanized has a tensile strength of about 2000 lb. per square inch

and an elongation of 960 per cent. of its original length. The effect of adding zinc oxide is to increase the tensile strength but diminish the elongation, as shown in Table 1.

TABLE 1

Per Cent. ZnO	Tensile Lb. per Sq. In.	Elongation, Per Cent.
25	2,400	720
35	2,400	700
45	2,700	680
55	2,500	620
65	2,000	540
75	1,300	400

It will be seen that up to about 60 per cent. ZnO increases the tensile strength of rubber. No other material tested has this effect.

Leaded Oxide.—As the supply of lead-free ores is limited and the demand for ZnO is increasing, much more leaded oxide is being made. The process is the same but the charges are usually lighter and the recoveries lower. In most cases the furnaces used are double ended with one or two working doors at each of the opposite ends. This makes the furnace easier to clean, as there is no back wall, which is the most difficult part from which to cut accretions. On the other hand, this type of furnace cools off more when charging, and does not, as a rule, light up so quickly. Which type is better is still an open question, the users of each claiming that theirs is superior. In making leaded oxide, it is customary to have a brick combustion chamber over, or close to, the furnace. This insures a sufficient time of contact at a proper temperature to cause the lead to be converted into basic sulphate. This is essential, as all of the oxides of lead are strongly colored and injure the color of the product, while the basic sulphate is very nearly white and does not have this effect. In many cases the bag rooms of the works making leaded oxide have no overhead distributing pipes, the hoppers being connected and used for this purpose.

Leaded oxide is divided into several grades, depending on the amount of lead it contains. This is always stated as the amount of neutral sulphate ($PbSO_4$) equivalent to the total lead present, although it is always present mainly as a basic sulphate. It forms an excellent material for the manufacture of mixed paints. So far this grade has not proved satisfactory to the rubber manufacturers.

Production

It is impossible to give the production of oxide of zinc, as the statistics are not published by the Government. Such figures as have been published are not comparable, because the grouping of the different products is not always the same. The increase has been very rapid, especially within the last 3 or 4 years.

Discussion

L. E. Wemple, St. Louis, Mo. (written discussion*).—Mr. Stone refers to cadmium as one of the worst impurities in ores used for the production of zinc oxide for pigment purposes, because it is very volatile and its dark brown oxide and bright yellow sulphide both have strong coloring powers so that small fractions of 1 per cent. injure the whiteness of the oxide.

Numerous tests have recently been made by the American Zinc, Lead and Smelting Co. to determine what effect the percentage of cadmium, as ordinarily present in zinc ores, has upon the color of oxide made from such ores by the usual grate-furnace process. It has been found that while the cadmium is readily volatile and may be detected in the oxide product by chemical analysis, the oxide product is not discolored, but is of a clear white color.

Analysis of the 12 leading brands of zinc oxide, French process, lead-free, and leaded grades now on the market for pigment purposes, show them to contain from 0.02 to 0.35 per cent. cadmium, figured as metal. If these proportions of cadmium were present as oxide or sulphide, the percentages would run from approximately 0.022 to 0.40 per cent. in the first case and 0.025 to 0.45 per cent in the latter case. All these grades of oxide are widely used for pigment purposes and are of a clear white color, except the leaded grades, the whiteness of which may be diminished by the lead sulphate.

Tests made with admixtures of as low as 0.02 per cent. of cadmium oxide or sulphide to zinc oxide of the greatest hiding power (lead-free oxide) show that such small quantities distinctly affect the color, and this coloring power is found to increase rapidly with increased amounts of these compounds. In leaded oxide the effect of CdO or CdS is much more strongly pronounced.

It is concluded from these tests and assays, that the small quantity of cadmium ordinarily found in zinc ores does not persist in the oxide product from grate-furnace treatment in the form of cadmium sulphide or cadmium oxide, but is present as white cadmium sulphate, either neutral or basic.

The Chairman (G. C. Stone, New York, N. Y.) I think Mr. Wemple is quite right, that in a very large proportion of cases the cadmium is present either as sulphate or carbonate, both of which salts are white, but there is always a great danger of getting cadmium oxide, 0.02 per cent. of which will reduce the value of the oxide by about 50 per cent.; for that reason I said it was a bad thing to use cadmium-bearing ores on account of the danger of occasionally getting one of the colored salts in the oxide. We have had trouble with it at times, but it is not always so, by any manner of means.

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* Received Oct. 11, 1917.

F. D. James, St. Louis, Mo.—You say the ores received are fine enough to use without further crushing. What size is that?

Chairman Stone.—The ore goes through about a 0.1-in. opening, and I have worked it as large as ⅜ in.; it varies somewhat with the character of the ore, but I do not think it makes much difference whether it is ⅜ or ⅝ in.

F. D. James.—Has anyone done any experimental work on the burning of retort residues for zinc? Have experiments been made as to whether it is better to charge residue straight from the retorts, or give it preliminary treatment before charging it into the grates? Has any attempt been made to remove the coke?

Chairman Stone.—I do not know of any work in that line. All that has been done has been to pick out the big pieces of old retorts and throw them aside.

E. G. Spilsbury, New York, N. Y.—This paper of Mr. Stone's, I think, is the first one describing the methods of manufacture of zinc oxide that has ever been published, and I feel that we owe a great debt to Mr. Stone for giving us the amount of detail as well as the history contained in this paper. The manufacture of oxide of zinc has always been kept more or less secret, and that one of the large companies should permit the publication of the details of its work to the extent that they appear in this paper is a matter on which we ought to congratulate the company, and also Mr. Stone, for having been able to overcome the desire for secrecy which has heretofore prevailed.

Zinc Burning as a Metallurgical Process

BY W. R. INGALLS

(St. Louis Meeting, October, 1917)

The manufacture of zinc oxide directly from the ore is one of the most important contributions that America has made to the metallurgy of zinc. Heretofore, this has been done chiefly for the production of zinc and zinc-lead pigments, and the method has been known as the Wetherill process. While that term may properly be applied to the process for pigment manufacture, it must be recognized that the principles of the process are of far wider application; and that there is going to be wider application in the immediate future is more than an expectation. I have in mind the use of the process as a method of igneous concentration.

Igneous concentration is broader in its scope than the Wetherill process, for it may be done in the blast smelting furnace, or in the reverberatory, or by other means. Igneous concentration is a rather clumsy term, and I prefer to introduce the new generic expression of "zinc burning," which is convenient, readily understood, and precise, for in all cases the zinc oxide is reduced to metallic form, is volatilized and is then burned to zinc oxide, the latter happening, however, after the zinc has been separated from the residue of the ore.

As previously indicated, zinc burning may be done either in blast or reverberatory smelting, the gangue of the ore being scorified and drawn off as slag, while other metallic minerals are reduced to matte or metal. Such processes have heretofore been practised on fairly large scales. Thus, F. L. Bartlett smelted zinc ores at Cañon City, Colo., in specially designed blast furnaces, wherein the smelting column was only about 18 in. high. Such a limitation was necessary in order to prevent zinky accretions in the shaft of the furnace, which in course of time would have interfered with its operation. In running the Bartlett furnaces it was easy to bar them off, indeed, to poke right down into the smelting zone, and keep things going freely. Some iron blast furnaces in Virginia and the spiegeleisen furnaces of the New Jersey Zinc Co. afford other examples of zinc burning in blast-furnace smelting, although in their cases the happening is incidental rather than a primary purpose.

Just as zinc may be burned off in a blast furnace, so also may it be done in a reverberatory. Indeed, there is less trouble, in some respects, in a reverberatory, for therewith there is no shaft in which accretions may

form and hang. The first attempt to burn off zinc in a reverberatory furnace on a large scale whereof I know was made with the Fry process, first in a works at Swansea and later in a large plant on the Manchester Ship Canal. Trial of the process was also made by the Anhaltische Blei und Silberwerke in Germany. In this process the roasted ore was first smelted with about 25 per cent. of salt cake in a blast furnace, yielding silver-lead and a zinkiferous slag. The slag mixed with 10 to 20 per cent. of coal was then smelted in a reverberatory furnace, wherein the zinc was reduced and burned off. Although something like 25,000 tons of Broken

Fig. 1.—Interior view of furnace building, east plant, New Jersey Zinc Co., Palmerton, Pa.

Hill ore was treated by this method, the process did not prove a commercial success. About the same time, Ellershausen was working on similar ideas at Angoulême, in France. He began by burning off zinc in a reverberatory, but he abandoned it in favor of a blast furnace.

About the same time, also, three European metallurgists—Pape, Witter and Babe—were working together on the same problem and took out certain patents in common, but subsequently they separated. Pape developed a method of burning zinc out of the Oker slags, developing a special furnace of the blast-furnace order, and made a great commercial success of it, his zinc fume being a large source of zinc supply to the smeltery near Hamburg. Babe did further experimental work with apparatus conforming to the principles of the Bessemer converter. Witter took up the idea of reverberatory smelting and about 1908 carried on some rather

extensive experiments at a works in Germany. However, neither Babe nor Witter developed a commercial method. It remained for Frederick Laist, at Anaconda, Mont., to make zinc burning in the reverberatory furnace a successful large-scale process. This was done early in 1917 for the extraction of the zinc remaining in the residues of the leaching for electrolytic zinc extraction.

The blast furnace and the reverberatory furnace are both smelting furnaces. A smelting furnace may be defined as one in which reduction, fusion, and scorification are performed. The Wetherill-grate furnace is a blast furnace in which reduction is performed but no fusion and no scorification, or but to a slight degree. There is no run of slag, but merely enough formation of it to clinker the residuum. Anything more would clog the grate and the furnace could not be worked. We have here a

Fig. 2.—Flue, cooling pipes and bag-house plant of Anaconda Copper Mining Co. at Great Falls, Mont.

combination of the conditions that exist in the retort for zinc distillation and in the blast smelting furnace. The zinc is distilled and the residue remains in somewhat the same physical condition as in the former. The product cannot be metallic zinc, however, for just as the carbon dioxide and excess of air act oxidizingly in the blast furnace they act more intensively in the Wetherill furnace. That the reduction of zinc oxide to zinc takes place is well known, for the beautiful bluish-green flames of burning zinc may be observed immediately above the charge, but it is quickly converted into zinc oxide by the carbon dioxide and excess of oxygen that are present, and what leaves the furnace is gas laden with zinc oxide (and lead oxide and sulphate if lead be present in the ore).

Heretofore the process of zinc burning on Wetherill grates has been confined to the manufacture of pigment. In that are involved problems in collecting a product of the requisite physical properties, especially purity of color. It is necessary to effect a perfect combustion of coal

dust mechanically carried over, a perfect separation of all mechanical impurities, and the ore that is treated must be free from certain impurities, such as cadmium, which affect adversely the color of the zinc oxide. In accomplishing these things much skill, that can be gained only from practice, is necessary. As a means of concentrating zinc, however, no great skill is required, the color of the product being no consideration, and the process then becoming so simple attention is being increasingly directed to it. There are several directions for its application, the most important of which seem to be as follows:

1. As a simple means of concentrating low-grade ore, especially calamine, to save freight and treatment charges on worthless gangue. Thus, the calamine of Leadville might be taken to Cañon City, Colo., the zinc might there be burned off and the zinc oxide shipped to distillers or to electrolytic refiners. This is, in fact, being done on a small scale.

2. The residues from lixiviation in electrolytic zinc works, containing a large quantity of zinc owing to the formation of insoluble zinc ferrite in roasting, may be burned on Wetherill grates for the extraction of that zinc. This is going to be done. An ore containing 48 per cent. of zinc, and yielding 83 per cent. of it to sulphuric acid, may have a residue assaying 20 per cent. zinc, the proportion of residue to original ore (roasted) being about one-third, and that 20 per cent. of zinc is well worth extraction.

3. It is possible that it may be found most economical in the electrolytic process to roast the ore, burn it on Wetherill grates, and leach the fume.

4. The production of zinc fume in this way affords an excellent supply of zinc for the manufacture of lithopone and other chemical purposes.

All of the above are important opportunities for the process of zinc burning, and within the next few years extensive applications of it are going to be witnessed. As to whether the application will be by means of the Wetherill grate or of the reverberatory smelting furnace (I think the blast furnace may be dismissed from consideration) will depend upon the nature of the gangue, the precious-metal content of the ore, and the commercial and metallurgical conditions that prevail at any given place. Before undertaking to analyze the factors that will govern, let attention be directed to the behavior of silver and lead. Gold may be left out of consideration, being of but rare occurrence in zinc ores.

Lead is even more easily burned out of an ore, either in reverberatory furnace or Wetherill, than is zinc. In my own experience, the extraction of lead has almost always been higher. In treating a lead-zinc ore, therefore, the product will inevitably be a lead-zinc fume. If the ore that is being burned contains sulphides there will be sulphur in the fume, for lead sulphide is volatile as such. The behavior of silver is variable. With some ores the larger proportion will go into the fume. With other

ores it may remain in the residue. It would be preferable if it would all stay in the residue or all go into the fume, *i.e.*, preferable in consideration of subsequent steps for its extraction, but that never is to be expected. As to what determines the behavior of silver, I am not prepared to pronounce. I imagine that if the ore contains a little copper, enough to form a matte as collecting agent, more silver will remain in the residue than otherwise. Anyway, that is the experience in blast-furnace smelting.

As between Wetherill-grate burning and reverberatory-furnace smelting and burning, I think it impossible to lay down any hard and fast rules. The subject has been insufficiently studied and analyzed. Reverberatory smelting follows the same lines as for the treatment of copper ore, but sufficient carbon is mixed with the charge to reduce the zinc oxide. No great excess over the theoretical quantity is needed, the conditions being very different from those of the retort for distillation. Nor does it appear that the quantity of coal that must be burned in the firebox is greatly in excess of what is used in smelting an ordinary copper ore. It might be feared that the loss of zinc by scorification would be high in this process. It is rather hard to keep zinc out of the slag in any kind of smelting. Even in electrothermic smelting it is common to have 5 or 6 per cent. of zinc in the slag, with good work, while the percentage may be very much higher. If, then, in order to make a proper slag for the running of the furnace, it is necessary to flux the ore so that the proportion of slag to ore is high, the zinc loss in the slag may be uncomfortably large although the assay of the slag may be moderately low. However, by maintenance of proper conditions, especially a neutral or reducing atmosphere, in the furnace, the zinc tenor of the slag may be reduced to rather surprisingly low figures and the extraction of zinc as fume may compare very favorably with what would be obtained in burning on Wetherill grates.

Bearing on this subject, the *Metalbank und Metallurgische Gesellschaft* in German patent, No. 290,013, Oct. 8, 1913 (addition to No. 252,195) says that the driving off of the zinc from the mixture of ore and fuel has been found to proceed more satisfactorily the smaller the excess of air in the heating gases entering the reaction chamber. Ordinarily in smelting a charge, a portion of the zinc is held back by the slag which forms, while the reduced zinc vapor is reoxidized by the excess air to ZnO, which is dissolved by the molten slag. In order to overcome this difficulty, the process should be carried out in a reducing atmosphere, avoiding fusion. In order that the highest possible temperatures may be used, the charge is mixed with ores or additions so that only a sintering results, without fusion. The CO_2 present in the heating gases has been found sufficient to oxidize the zinc vapors without excess of air.

W. Troeller, in German patent No. 291, 853 (Apr. 2, 1913) reports that experiments have shown that the expulsion of zinc and other metals

from liquid slags or melts is dependent upon the degree of oxidation of various metals contained in the slags, *e.g.*, iron, and that such expulsion is practically complete only when these metals have reached their lowest degree of oxidation or are maintained in that condition. In view of this fact, a current of reducing gas, *e.g.*, illuminating gas, water gas, or the like, may be conducted through the slag or melt of ores, maintained in a liquid state at the reaction temperature in a directly or indirectly heated furnace, whereby volatile metals such as zinc, bismuth, antimony, etc., are reduced from their compounds and are driven out with the escaping gases.

In other words, once the oxide of zinc is reduced, the zinc vapor must be given the best possible chance to be volatilized and be reoxidized above the slag bath, not in it. If slagging be prevented, the scorification of zinc oxide will be reduced. It should be theoretically possible, for this reason, to burn the zinc more completely out of an ore on a Wetherill grate than in a reverberatory furnace, and I believe the results of practice are in line with this idea, but, as I have previously indicated, good work with the reverberatory furnace may discharge slag with only 3 per cent. Zn, and that is not far behind the zinc assay of the cinder from the best grate burning.

In the use of labor and fuel, the large reverberatory, of the Anaconda type is far ahead of the best form of the Wetherill furnace. With a small reverberatory the difference will be less, and indeed may disappear. Local conditions will, no doubt, determine the choice of method in all cases. For a small plant to treat ore giving an essentially basic or essentially acid residuum of no value for silver or copper content, the Wetherill furnace would be on the most advantageous terms. If, however, the residuum approximated self-fluxing character and contained silver and copper contents worth recovering by smelting, it would be likely to be most advantageous to smelt at once and be done with it, recovering the zinc incidentally. The latter process will of course suggest itself in connection with the metallurgical treatment of deposits of zinkiferous pyrites containing sufficient copper for matte formation.

Discussion

The Chairman (G. C. Stone, New York, N. Y.).—It is a matter of history that the smelting of zinc ore in a reverberatory furnace was the first process used in this country for making oxide of zinc, and it was abandoned for three reasons: it was very expensive, the recovery was very poor, and the color of the oxide was worse. It is probable, however, that the Anaconda company is working in a totally different way. While Mr. Ingalls does not say so, my own suspicion is that they are using coal-dust firing, very much as they do in the copper furnaces, and by that method there is much more chance of getting good recovery.

Some Economic Factors in the Production of Electrolytic Zinc

BY R. G. HALL,* B. S., ST. LOUIS, MO.

(St. Louis Meeting, October, 1917)

An article on the subject of electrolytic zinc no longer needs to be preceded by an apology. The production of zinc by electrolysis is past the laboratory stage and has become an economic factor of considerable importance in the total production of zinc in the United States.

The various methods of procedure to be followed for the greatest economy of manufacture are fairly well known, consequently a detailed description of any method is no longer necessary or desirable. In this paper I shall endeavor to lay down some of the governing conditions in this industry, the relation of methods to ore deposits, freight rates, and sources of power supply.

Power Points

Assuming for a moment that, for the present at least, the electrolytic production of zinc will be confined to those localities having developed water power, a glance at these powers so developed in the United States, with their relation to known zinc orebodies of economic size, will not be without interest.

Northwest

In the Northwest we have water powers, developed and undeveloped, in Washington and Montana. These must, of course, bear a close relation to the rapidly growing zinc production of the Coeur d'Alenes and of the Butte districts. In general, the power of the territory in question is relatively low-priced, at least as compared with the cost of developing power from coal, and the near neighborhood of large bodies of more or less complex zinc ores makes these water powers of more than usual interest.

The same remarks may be applied equally to the developed water powers of the Canadian side of the line in the Northwest, where we have again commercially available complex ore deposits such as are being exploited today at and near Trail.

California

One of the large complex ore deposits of the far West is to be found in Shasta County in the immediate neighborhood of the territory of the

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Northern California Power Co., and developments in this connection are said to be now under way. Again, we have the highly desirable combination of a relatively low-priced power with a good-sized available ore deposit.

Utah

In the Utah territory we have certain developed water powers in connection with such ore deposits as the Tintic and the Park City districts. Evidently the relation between the power and the value of the ore deposits is sufficiently close to justify further development, as we see such already undertaken in that territory. Further to the east we have power development at various points in Colorado in the neighborhood of the large and valuable complex ores of the Lake, Summit, and Eagle County districts, as well as those of the San Juan district.

The writer is not sufficiently familiar with the cost of large quantities of electric power in the Colorado district to judge whether future development of electrolytic zinc in Colorado will be permitted in connection with these water powers.

Southwest

In the Southwestern country, namely, Arizona and New Mexico, there is not, so far as I know, sufficient development of low-cost power to make attractive any particular effort to develop electrolytic zinc in that territory.

Mississippi Valley

Passing east from the Rocky Mountain territory, we have no further development of power on a large scale until we reach the Mississippi Valley. Here we have the power plant at Keokuk, Iowa, producing power, which, while perhaps not as low-priced as some of the Northwestern powers or as the older Niagara Falls power, is yet, when considered in connection with its economic position in reference to freight rates, to be considered as a point of considerable advantage.

Further north there was formerly some power available at Sault Ste. Marie, Mich. This, however, is so remote except for Canadian or Northwestern ores that it need hardly be considered in connection with the present inquiry.

The development of power in the southern Mississippi Valley has not yet become of sufficient industrial importance to attract any attention for electrometallurgical work.

Further east we have the power developments at Niagara Falls, but from all accounts these are already taken up to the fullest extent and no power is available, at least none in large quantities for electrometallurgical work.

It is not to be considered that the above survey is in any sense complete, but only as more or less indicative of those powers which may be considered as of importance in connection with developments of electrolytic zinc in the future.

Coal and Fuel Supply

The electrolytic refining of copper requires not alone power for the electrolytic tanks, but also requires large quantities of coal for heating the solutions with steam heat or otherwise, as well as coal more or less of special quality for cathode furnaces.

In the electrolysis of zinc solutions we do not have this requirement because, in general, the solutions do not require heating, but on the contrary require occasional cooling, and the temperature for the melting down of zinc is so low as not to require a large amount or any special quality of coal for the cathode furnaces. Consequently, for an electrolytic plant alone, depending on central-station power, coal is relatively of very small importance. This does not mean that coal is not to be used at any point in the process, but only that it is not of any importance in the electrolytic process itself.

Acids and Chemicals

The consumption of acids and chemicals in electrolytic-zinc production is relatively quite small. No matter what kind of ore is used as a supply of zinc, unless it should be an oxidized ore with considerable quantities of soluble carbonates other than zinc, the amount of sulphuric acid lost in a cycle is a very small factor, and in a modern electrolytic plant the use of other chemicals than those for laboratory purposes may be considered as nil.

Freight Rates

The relation of freight charges to grade of ore is an important one and requires to be considered more in connection with the general ore supply than by itself. It is to be noted, however, that the basing point for lead and zinc, the principal metals with which we are concerned in this inquiry, is Mississippi Valley, consequently the value of the refined product must be considered only in connection with some delivery point having a freight rate basis that can be compared to the rate from East St. Louis.

At the present time, of course, with much of our metals going for export, this factor is considerably complicated, but for the purpose of this paper we shall consider the cost of the metals f.o.b. Mississippi Valley points, or equivalent to East St. Louis basis.

Ore Supplies

As indicated above, the ore supplies most available for the production of electrolytic zinc are those of the Rocky Mountain and far western territory. We shall consider these further in detail with especial reference to their availability for the process in question, and the various factors in their treatment to bring them into the most desirable shape before the real electrolytic treatment begins.

Joplin Territory

To those acquainted with the ores and concentrates produced in the Missouri Kansas-Oklahoma field, it need not be said that at the present time the electrolytic process for zinc recovery has nothing to offer to this territory. The ores are simple in their character; that is, the number of metals concerned is small. They are easily concentrated as compared with most of our zinc ores of the Rocky Mountain territory, and, best of all, they are easily concentrated to a very high tenor of zinc. The concentrate consisting of 60 per cent. zinc (in the form of sulphide) has only about 10 per cent. of total gangue left in the ore. I do not make any prediction as to the future development of the electrolytic process with its relation to the ores from the Joplin territory, but I can say that today the older process in the field has nothing to fear from the newer.

Northwestern Field

The developed orebodies in Montana are somewhat in a class by themselves. While they are complex in the variety of metals present, they do not present the complex concentration problems that are found in some other territories of the Rocky Mountain field. For example, the largest producer in the territory seems to find no difficulty in making a concentrate containing 50 per cent. or more zinc and a recovery which, viewed in the light of our experience in concentrating zinc ores of a few years ago, may be considered very remarkable indeed. The concentrate so produced is high-grade and of a character desirable for the retort process of smelting, and this, in general, may be said of all the concentrates being produced, not alone in the Butte district but in the Coeur d'Alenes as well.

The concentrates of the Butte district, however, for the most part contain, besides zinc, a very valuable constituent in their silver, while the elements of less value are lead and copper. Whether the electrolytic process has sufficient attractions to offer to the producers of these ores is a question that must be answered for each individual case, and I think has been answered successfully and conclusively by one large company for its own work.

The principal points that I wish to emphasize in regard to the use of Butte concentrates are their high tenor in zinc and their considerable values in silver, with copper and lead following after in various proportions. Any method of treatment, therefore, must take account of these lesser values as well as of the major value of the zinc itself.

The whole value of the electrolytic process for the treatment of ores of this character must then be considered, not alone the actual cost of reduction of the zinc itself, but also the relative values of the subsidiary metals recovered by the electrolytic process as compared with the direct process of retort smelting.

The recovery of the subsidiary metals, especially the precious metals, which may be obtained by electrolysis, may well be so much greater than the recoveries by the somewhat wasteful retort method as to show a substantial advantage over the latter in ultimate profits. This remark will apply equally to all ores of the Rocky Mountain country where these ores contain values other than zinc.

In the case of the ores of the Coeur d'Alenes district, so far as my limited acquaintance with that territory goes, we have in general a somewhat different condition. They resemble the Butte ores in that a concentrate of a relatively high grade in zinc can be made; but the silver is usually of a lower tenor and the lead of a somewhat higher. In both cases, however, it is to be noted that the element which presents the most difficulty in the treatment of zinc ores; namely, iron, is to a large degree absent. Consequently, the ores lend themselves readily to concentration, either by the older method of gravity concentration, or by the newer flotation, or a combination of both, and the resultant concentrate is such as to be of value either for the retort process of smelting or for the electrolytic process of zinc extraction.

California

The deposits of complex ores in Shasta County, tributary to the distribution lines of the Northern California Power Co., are of course of very great importance and the developments now taking place promise to form a large factor in the production of electrolytic zinc of the United States. These ores, so far as my acquaintance with them goes, are generally of a somewhat complex character and do not lend themselves readily to a gravity separation of the various minerals. Consequently, they have always presented a more than usually attractive field for the electrometallurgist, and about the first really intelligent effort to produce electrolytic zinc, at least the first with which the writer is acquainted in the United States, was made in connection with the large ore deposits in that territory.

Utah

The zinc ores of Utah are of great importance, and it is confidently believed that the application of the electrolytic process for the treatment of these ores will bring Utah up to a point where that State will be a considerable factor in the production of zinc. The ores of the Park City district are somewhat simple in character, in that they are readily concentrated to a fairly high tenor in zinc but are also apt to carry very considerable values in silver. In the Tintic district the ores are of an even more complex nature, so far as the writer is acquainted with them, and they have proved in general somewhat difficult to concentrate, classing in this way more with the Colorado materials to be mentioned later.

Nevada

In southwest Nevada there are very considerable bodies of the mixed carbonates of lead and zinc. So far these have been handled by wet concentration with considerable success, and on account of their great value as "mixing" ores in the retort process, it is somewhat questionable whether the electrolytic process would have an application here.

Colorado

The deposits in practically all of the districts of Colorado are of an entirely different character from those of the Northwest and, to my mind, must be approached in a different manner for intelligent exploitation. Taking the ores of Lake County or the Leadville district as an example, we have a variety of minerals and a complexity of mineral structure rarely occurring in the Northwestern territory.

The primary zinc mineral, "black jack," or marmatite, is in itself a complex of zinc and iron sulphides, which for the most part contains, when pure, only from 40 to 45 per cent. metallic zinc. And, in addition, we have galena, generally in commercial quantities, often crystallized so closely with the zinc-bearing material as to be impossible of separation without extremely fine crushing. Pyrrhotite and pyrite also are constituents which must be reckoned with in the concentration, as these generally carry some precious-metal values and are often crystallized with the other minerals so closely as to be difficult of mechanical separation.

Any method of concentration, therefore, which will extract all of the values from these ores without attempting to separate one mineral from another will necessarily result in a complex concentrate containing zinc, iron, lead (as sulphides) together with more or less silver and gold, with occasionally some copper.

The present method of concentration of these ores is one that has been

worked out in a most ingenious way by the pioneers of the business and for the most part makes three products:

First in value generally, we have zinc concentrate. Averages are somewhat dangerous, but on the whole I believe I am not far from the truth in saying that the concentrates produced from Colorado ores by these methods will not average much over 40 per cent. and will rarely go above 45 per cent. zinc. In addition, they usually carry more or less lead, the balance for the most part being iron in the form of sulphide, with the rock gangue very low. The silver contents will vary all the way from 1 to 2 oz. up to such a point as to make it a considerable factor in the value of the concentrate. It is quite evident that such a complex as this presents equally complex problems in the subsequent metallurgical treatment.

Next in value in general, we have lead concentrate. This concentrate will necessarily show a considerable tenor in zinc, and as the silver shows a preference for the lead, the silver contents of the lead concentrate is usually considerably higher than that of the zinc concentrate.

Finally, there is a product usually produced from magnetic machines, which is not in all cases of economic value and cannot always be considered as a concentrate, although the writer has sometimes thrown it out as tailings and afterwards sold it as a concentrate, so it depends somewhat on conditions which it is. This is usually a very high iron, low lead, relatively low zinc, and low silver material. For the most part, in the district under discussion, the silver and lead values are very low and, as indicated above, it is often a question whether it is a concentrate or a tailing.

Lastly, there is a rock or gangue matter which is generally eliminated with difficulty and usually carries with it a considerable percentage of the zinc.

Taking the process as a whole, then, it is seen that there are three products of more or less economic value and one of waste, and each one of these carries off with it something which does not belong there, and occasionally carries it off to such an extent that it also carries off the profit of the mine. It is to the handling of these ores that the writer has devoted most of his attention.

The ores of the San Juan district in Colorado differ from the ores of the Lake County and Summit County districts only very slightly in their complex characteristics, and remarks made concerning the one will apply with very slight modifications to the others. I do not mean to intimate that there are no simple zinc ores in Colorado. I have seen such ores there, but the production has not yet become an appreciable factor in the world's production of zinc.

Carbonate Ores.—The remarks made above concerning concentration of complex ores in Colorado do not apply to the carbonate

ores of the Leadville district and some other territories in Colorado. These ores are in a class by themselves and have hitherto been handled only for the recovery of their zinc directly by retort smelting, without preliminary concentration.

A certain quantity is now being converted into a zinc concentrate or oxide, for use in the manufacture of pigment, but so far as I know no real attempt at concentration of these ores has been made for the purpose of beneficiating them for manufacturing spelter; at least none has been undertaken on a commercial scale.

Arizona

Complex ores having their principal values in zinc and minor values in copper, lead, and silver, have been developed at various points in Arizona, notably in the Chloride and neighboring districts and certain other points in the south central part of the State.

For the most part, these ores present problems similar to those of Colorado in their mineral complexes, having minerals not easily separated from one another, and, so far as the writer is acquainted with the territory in question, remarks on the subject of the Colorado ores will apply, with but slight modifications, to those found in Arizona.

New Mexico

The various fields in New Mexico may be considered in somewhat the same category, some of the ores being even more difficult of concentration than those of Colorado so far as making a high-grade concentrate of zinc is concerned.

Of this Southwestern territory in general, then, it may be said that the ores are found to contain zinc, copper, silver, and lead, mentioned in general in the order of their importance, with iron as an invariable accompaniment, and rock minerals which have sometimes proved very difficult to eliminate by methods of gravity concentration.

Requirement for Electrolysis of Zinc

The requirement, and I use it in the singular, for the electrolysis of zinc is zinc sulphate and zinc sulphate only. Most other elements found in the solution are harmful to the electrolysis of zinc sulphate. I have never found any that I could confidently say were beneficial. This requirement seems relatively simple, but when it is understood that this process is to be applied to such a complex of ore minerals as is indicated above in the review of ore deposits, it will be understood that the production of zinc sulphate, pure, of the standard strength, is not the easiest problem in the world. And yet it is only this problem that has stood in

the way, for all time past, of the manufacture of zinc electrolytically. All of the factors necessary to the production of a solid coherent plate of electrolytic zinc were known long ago, but it is only recently that we have been able to produce this pure zinc sulphate on a commercial basis and in large quantities, and have been able to obtain the electric current at such a cost as to make the production of electrolytic zinc a commercial possibility.

Roasting of Ores

Inasmuch as sulphide of zinc as it occurs in nature is not in general soluble in dilute sulphuric acid for the direct production of zinc sulphate, it is necessary to roast sulphide ores in order to eliminate the sulphur and convert the zinc to an oxide or sulphate.

The roasting of zinc ore for sulphuric acid leaching presents a very different problem from the roasting of the same ore for retort smelting. In the case of retort smelting it is necessary to have the sulphur eliminated to the greatest possible extent. Sulphur remaining behind either as sulphate or sulphide, beyond a certain amount, is harmful.

For the solution of zinc in sulphuric acid, however, only the sulphur remaining behind as sulphide, and not always all of that, is harmful or causes a loss of zinc. Zinc that has been roasted to sulphate is, of course, not harmful and is even desirable, as the sulphuric acid combined therewith helps to make up for the mechanical and other losses in the subsequent process.

Another difference in the two methods of roasting arises from the fact that in the roasting of ferruginous zinc ore for retort smelting, little attention is paid to that complex compound of zinc and iron which is generally formed in a roaster from such ores, and is usually referred to as a zinc ferrite.

In roasting Colorado ores for the production of spelter, the writer has found as much as one-third of the total zinc in such ore to be insoluble in a relatively strong solution of sulphuric or hydrochloric acid. It is evident that a material such as this would be fatal if formed in quantity when the ore is to be subjected to leaching with dilute sulphuric acid containing only from 6 to 8 per cent. H_2SO_4.

Fortunately, however, the conditions which produce this so-called ferrite are not such as need necessarily prevail in the roasting of ores for leaching with sulphuric acid. The production of this material in the roaster seems to be a function of temperature, fineness of ore, and time.

In roasting for the production of spelter the temperature is usually raised at the end of the roast for the purpose of decomposing sulphates and the elimination of the last possible amount of sulphur. As pointed out above, in a roast for leaching with sulphuric acid, the presence of

sulphates is rather an advantage than an objection, and consequently the same high temperature is not necessary.

The other factors mentioned do not necessarily vary much in either case. It is evident, however, that in roasting a complex concentrate such as that from Colorado and the Southwest, containing only about 35 to 40 per cent. zinc, most of the balance of the metal contents being iron, there is great danger that an undue proportion of the zinc will be held back, insoluble; at least, such has been the writer's experience.

In roasting zinc ores for the complete elimination of the sulphur, the loss in roasting of fine material, carried out mechanically, and of zinc, lead, and silver actually volatilized, is quite appreciable. The writer had the somewhat doubtful pleasure of smelting several thousand tons of some of the earliest concentrate produced by the flotation method in the Montana territory. These concentrates were roasted in the standard type of Hegeler kiln, and by the time they had dropped seven times from hearth to hearth, and leaked out of the doors and other parts of the kiln not made for ores to leak out of, by no means all of the original concentrates found their way into the retort. Subsequent practice has much improved this, but the roasting of a flotation zinc concentrate in a multiple-hearth kiln is not one conducive to sweetness of disposition in the metallurgist.

Extraction of Zinc with Sulphuric Acid

It is evident that in the leaching with sulphuric acid, of a calcine, produced as above mentioned, other metals than zinc may go into the solution. Contrary to the general belief, the solution of iron is not a difficult one to deal with, as this element is rather easily removed.

In the case of a concentrate containing copper, a considerable amount of the copper will be dissolved with the zinc. This removes the copper from the place where it ought to stay—namely, with the lead and silver—and takes it where it should not be—with the zinc. Consequently it has to be removed by separate treatment. Other soluble metals and metalloids are dealt with in the familiar standard manner which needs no description here.

The handling of the residue from the sulphuric acid leach is, however, one that is of considerably more interest. The metals economically valuable in it are copper, lead and silver, with probabilities of some gold, and the method of smelting must be adopted to the metal of principal value, whether copper or lead.

If the lead is high, as is frequently the case, the material will go to a lead smelting plant where the copper is of secondary consideration. In many cases, notably in the Southwest, the copper is the metal of principal value, and under these conditions it may pay to send the material to a copper smelter and sacrifice the lead.

The losses in leaching are apt to be slight in the case of lead, silver and gold, but may be considerable in the case of copper, depending upon the subsequent treatment of the zinc sulphate, and the method adopted will be such as to fit the individual case.

General Conclusions on Roasting and Leaching Processes

Considering the ore as produced from the mine and carried straight through to the production of the finished metals, along the above indicated lines, we have:

1. *Concentration.*—It has been shown that the concentration of a zinc ore may produce concentrate of a very low or very high tenor in zinc, depending on the other minerals present and the easy separation of these minerals.

As pointed out above, when the ore is concentrated for the purpose of roasting and leaching with sulphuric acid, it would be desirable that a concentrate should be made as high-grade in zinc as possible and as low-grade in iron as possible. The loss in leaching will be directly proportional to the volume of tailings remaining and also somewhat proportional to the amount of iron in the calcines.

2. *Roasting.*—The remarks made above on the roasting of zinc ores for subsequent production of electrolytic spelter sufficiently cover this side of the problem. In general, for an ore high in zinc and low in iron, there will not be much difficulty in roasting to a point that will make possible an economical extraction of the zinc.

3. *Leaching.*—It will be seen that a calcine containing 68 to 70 per cent. of zinc oxide, equivalent to about 55 per cent. metallic zinc, would have a residue of about 35 per cent. when this residue contains 10 per cent zinc. Whereas, a calcine containing 40 per cent. metallic zinc would have a residue of very nearly 60 per cent. when the residue contains 10 per cent. zinc.

In the first case, we would have a loss of only about 6 per cent. of our original zinc, while in the second case we have a loss of approximately 15 per cent. of the original zinc, and with such a low-grade concentrate as this the loss is likely to be even higher.

It is evident, therefore, that as the grade of concentrate so roasted and leached is lowered the losses in zinc may rise very rapidly, not alone relatively but absolutely, and this loss will become the greater as the percentage of iron increases in the residue.

4. *Power.*—Again it will be seen that only in those cases where the source of current is relatively near by, or where the concentrate can be made high-grade, will it be permitted to follow this method of handling.

Comparison of the total amount of freight paid on the contents of a 50 per cent. zinc concentrate from Montana, landing 85 per cent. of the

product in New York, is of interest as showing the approximate distribution of the freights paid and the relative amount of freight paid in each case.

2000 lb. ore to Mississippi River	$7.00
5 per cent. moisture	0.35
850 lb. metal to New York	1.44
	$8.79
850 lb. metal, Montana to New York	4.31
Difference in freight paid on 1 ton 50 per cent. ore	$4.48

From the difference of approximately $4.48 per ton of ore shown above, in favor of electrolysis at or near the mine in Montana, must be deducted such local freights as may be paid in case the power is not obtainable at the mine itself. This may decrease the above difference by as much as $1 per ton.

All of these conditions have to a very desirable extent been fulfilled in the Montana and Idaho field, in that there can be made a high-grade concentrate with a high efficiency of concentration, and that there is power close to the ore supply, so that in no case does transportation become a serious matter. I think the balancing of these factors is of the greatest importance for success in the production of electrolytic zinc.

Concentrating Smelting

As pointed out above in the discussion of ore supplies, ores of the Colorado territory are of a very complex character which do not admit of the high ratio of concentration or of the good recovery in the concentration itself. The cost of milling is relatively high and the recovery of all the metals contained is low as compared with the modern practice in simple ores.

The writer approached this problem, therefore, with the idea of eliminating to the greatest possible extent this preliminary concentration, as a great many of the ores of Colorado are already what one would consider, in other industries than zinc, relatively high-grade ores. Where the ores as mined contain too much waste, or are too low tenor in valuable metals, these can readily be concentrated to a product containing all the metals in a form which, though valueless to the retort process, are easily treated by the method proposed. That is, the ores may contain from 15 to 20 per cent. zinc with varying quantities of lead, silver, and gold. It was necessary, therefore, to find a method for handling these ores which would:

1. Produce a high-grade simple concentrate, enabling us to carry such concentrate to a central refining point without incurring high freight charges.

2. Put this concentrate into such shape that it would be relatively easily separated into its constituent metals. Concentration smelting, with the volatilization of the zinc and the lead, was consequently decided upon.

Most of the ore supplies of central Colorado today are of sufficient grade to permit direct smelting in the blast furnace, whereby the principal part of the precious metal values become collected in a copper matte and the zinc-lead values are driven off as a fume and collected in the bag house. This substantially is the process as carried out at Florence, Colo., today.

It is evident that the physical character of the ores must be such as to permit charging into the blast furnace, or in the case of concentrate, that it must be put into this physical character. Again, that the ore charges must be balanced in such way as to give a slag that will carry the minimum of zinc to waste. Fortunately the Colorado ores are of such varied character as to permit of the fulfilling of both of these conditions quite readily.

The ores are charged without preliminary roasting; and with low copper contents on the charge a first matte is generally made carrying sufficient values to bear shipment directly to the refinery. Part of the silver in the charge is carried into the matte and the balance goes over with the volatilized metals into the bag house. The volatilization of the zinc and the cleanness of the slag still leave much to be desired and a great deal of interesting work in this direction remains to be done.

Concentrates produced in this way will contain only those metals and metalloids that are volatile under the conditions of the blast furnace, consequently it is a relatively simple mixture of oxide of zinc and sulphate of lead, with some sulphate of zinc and a small amount of silver.

In this connection there is necessary a word of acknowledgment of the pioneer work carried on on these lines by F. L. Bartlett in his smelter at Cañon City, Colo., for a number of years.[1] A somewhat intimate acquaintance with Bartlett's work, which the writer had the opportunity of obtaining some years ago, really forms the ground work for our practice in Florence, Colo., today.

It is evident that in such concentrate as this the zinc is readily soluble in dilute sulphuric acid, and the lead and silver quite insoluble, consequently the purification problem becomes a very simple one. The physical character of the material is such as to make it easily handled for shipment and easily handled in the leaching vats. The metal contents are high, and shipment is in the direction of consumption of the metals, and this point must be borne in mind in connection with our refining at Keokuk.

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[1] *Trans.* (1893), **22**, 661.

Residue

The residue of lead and silver is relatively small when figured back on the tonnage of the original ore handled, and being a very finely divided, rather heavy material, it is with considerable ease washed reasonably free from zinc sulphate. The treatment of this for its lead presents no great difficulty, as the only metals of economic value are lead and silver.

Influence of Freight Rates

It will be seen from the above that, considering the character of concentrate possible with the ores with which we have been dealing, and the location of the power that was available for the electrolytic process, only some such method as the above could be employed.

It is true that the loss of zinc in the blast-furnace slag is high, extremely high when compared with metallurgical work on other metals, but we endeavor as far as possible to make the milling and smelting operation one, so that one loss covers everything, and I am inclined to think that, under the conditions, to have a big hole in one pocket is perhaps no worse than to have little holes in three or four.

For work of this character it would, of course, be necessary that the smelting plant be located close to the ore supply. Such condition is easily attained in the Central Colorado territory, as the freight rates to the valley are relatively low, or a smelting plant could be located near the mines should it be necessary or desirable.

In the case of the ore supply of the San Juan territory, conditions are somewhat different, but it is rather fortunate that most of the ores as mined in the San Juan district are of a somewhat higher grade and would lend themselves readily to a rough concentration before shipment.

The freight on the zinc concentrates from the smelting point to the refinery becomes relatively small when figured back on the original ore from which it is produced, and in any event the concentrate is moving toward the market for final metal production. Should such concentrate be moved in any other direction, the freight charge would become a direct charge on the cost of production which would not be redeemed in any way.

Cost of Current

The cost of electric current in the production of electrolytic zinc is fortunately directly proportional to the metal recovered, rather than to the tonnage of ore treated. For this reason it becomes profitable to use it on a 20 per cent. ore or concentrate from Colorado, where it would not be profitable to use it under present conditions on a 60 per cent. concentrate from the Joplin district.

And, further, when the actual cost of current per ton of ore is figured back on a 20 per cent. ore it becomes, relative to the other charges incurred, only a small factor. In such case it would probably not exceed 25 per cent. of the whole cost of producing the metal and shipping to point of consumption, when figured on the present power cost in most of the eastern territory.

The actual power consumed in the production of metallic zinc will be for most cases about 1½ kw.-hr. per pound of zinc in the electrolytic tanks. Additions to this, of course, must be figured in the auxiliary power for the plant, transformer losses, rectifier losses, conductor losses, etc., all of which will be perfectly familiar to any electrochemist, so that altogether, probably, when working on central-station current 75 per cent. of the current purchased will go into the electrolytic zinc on the basis above mentioned.

Costs on this basis need not be figured, as each district will be a law unto itself, and power contracts will be based on load factor as long as central-station power is used. Central-station power is usually received as high-tension alternating current, and for the electrolytic tanks has to be stepped down and rectified. For the auxiliary power it has to be stepped down and may or may not be rectified, as the purchaser pleases.

Load factors in electrolytic work are usually very high, approaching 100 per cent., consequently the horsepower per year cost can be figured almost directly into a kilowatt-hour cost; that is 0.1 c. per kilowatt-hour is approximately equal to $6.50 per horsepower year at 100 per cent. load factor.

General Conclusions on Concentrating Smelting Processes

From the above rather crude analysis of the various factors entering into the production of electrolytic zinc, it will be seen that practically every district becomes a subject of study by itself. There are certain fields more than usually favorable for concentration, roasting, and direct solution of the zinc.

This involves an ore easily concentrated, a concentrate relatively simple, having its zinc value high and preferably low in copper and low in iron. The more nearly these conditions are attained the more probable will be ultimate success in the work.

This requires, also, either close proximity of power to the ore supply or such a high-grade concentrate as to bear carriage to the power supply. We have seen that the ores of the Northwest particularly lend themselves to these conditions.

On the other hand, low-grade ores, ores difficult to concentrate, giving a complex concentrate high in iron or copper, or both, and power at a long distance from the orebodies, would suggest concentrating smelting.

Such conditions we have in the Colorado territory when considered in connection with power developments in the Mississippi valley.

The remarks concerning ore supply include those concerning power. Either the power must be convenient to the ore supply or as the ore moves toward the power it must also move toward the metal market. Again, we have the case of Northwestern power favorably located with reference to its orebodies, and Mississippi Valley power favorably located with reference to metal market. In neither case will transportation charges accrue to a too great or unredeemed extent.

Discussion

Lawrence Addicks, New York, N. Y.—We have really two competitive processes for the production of electrolytic zinc. In the process used at Great Falls, the ore is roasted and the calcines leached, leaving a residue which has to be smelted; in the other method, the ore is burned, the zinc and lead fumed off, and the condensed fumes are leached. I do not think that we can yet come to any decision as to which is the better way to attack the problem. Of course if we roast first and then leach, various substances remain in the calcines which may consume sulphuric acid; whereas if we do the burning first, we obtain simply the zinc and lead oxides, with considerable sulphate, in the fume, and there is no shortage of acid. It also depends partly on the complexity of the ore. The one that Mr. Hall is treating and the ore they are treating at Great Falls are not quite alike, but I think they are sufficiently similar to make a comparison of the two methods, working in competition, on different parts of the country, very interesting.

C. E. Schwartz, Miami, Okla.—Mr. Hall states that the Joplin conditions are not suitable for the manufacture of electrolytic zinc. When we speak of Joplin ore in general, we consider, of course, a 60-per cent. concentrate with comparatively low iron, but if a low-grade concentrate could be made from tailings and so-called chats, then the problem would be quite different. A Joplin ore is comparatively simple to treat electrolytically, and I believe the electrolytic process would open a field for the treatment of such ores as would not be ordinarily smeltable; that is, we could make a low-grade concentrate of 25 or 30 per cent. zinc from these chatty products, which would give us a very good material for electrolytic treatment.

Jos. W. Richards, So. Bethlehem, Pa.—Under the heading "The Requirement for Electrolysis of Zinc," Mr. Hall says, "I have never found any element that I could confidently say was beneficial," meaning, in the solution during the electrolysis. I believe it is true, however, that the presence of manganese in the solution is of advantage during electroly-

sis; in fact, in some of the plants a manganese salt is purposely added to the solution because of its beneficial action during the electrolysis. The manganese separates out on the anodes as manganese dioxide, helping to prolong their life and also furnishing indirectly a method of extracting the manganese from the raw materials used in the manufacture of the soluble manganese salt. It is quite true, however, that small amounts of certain impurities render difficult the deposition of the zinc in coherent form, and great care must be taken to maintain the purity of the electrolyte. I take exception to the statement that all the factors necessary to the production of electrolytic zinc were known long ago. One of the most important factors recently determined is the extent to which the zinc can be extracted from the solution, leaving the solution acid. Many thousands of dollars have been spent in experiments during the last few years to determine just how far the electrolysis can proceed, and the acidity of the solution be increased, without decreasing too much the deposition of the zinc.

I think that not enough emphasis has been placed in the paper on the possibility of the electrolytic methods replacing the retort methods for zinc. I base my prediction, or my stand, on the fact that the retort methods have been used for over a hundred years and at the present time are being only very slowly improved, whereas the electrolytic methods have been in use not more than five years and are susceptible of very much greater improvement. I think that where you may expect a possible improvement of 10 per cent. in the retort processes in, let us say, the next five years, there is possible, in my opinion, an improvement of perhaps 50 per cent. in the electrolytic processes. We must not overlook the possibility of a relatively much greater improvement in the new electrolytic method, and I think that the Joplin territory will very soon see an invasion of electrolytic zinc, as has been pointed out, for their special concentrates or special kinds of ore.

Lawrence Addicks.—I would like to say just a word in answer to Prof. Richards about manganese. I do not think that manganese is used in any of the plants. There was a plan to use manganese, but not in connection with electrolysis; it was because, by being oxidized at the anode, that oxidizing agent put into the ferric state the iron that was leached out of the ore. But since then it has been found that iron does not do any particular damage in the solution and that manganese is objectionable, because manganese dioxide is formed on the anode and makes a crust which is not desirable; for that reason it is not put into any of the electrolytes, so far as I know. Small quantities are present in the ores treated at some of the plants. The whole problem is to preserve the passivity of the zinc in free sulphuric acid, and the merest traces of almost any element will destroy that passivity more or less; hence, the

real problem is to keep an absolutely pure zinc electrolyte, and on the degree to which you maintain that condition depends your current efficiency.

S. J. Jennings, New York, N. Y.—I agree with Mr. Addicks as to the prime requisite in the electrolysis of zinc. We are operating an electrolytic plant in Shasta County, California, and the main expense and the great trouble is to obtain an absolutely pure zinc sulphate electrolyte. Two Englishmen, Mr. Tainton and Prof. Pring, have patented a process by which you can deposit zinc from an impure electrolyte and use your electrolyte to leach out your zinc with certain other elements, in the same way that you use cyanide to leach out gold in a cyanide plant. This is the most promising improvement that I have heard of, and if it can be successfully applied, it will bring about the improvements that have been suggested by Prof. Richards.

The Chairman (Geo. C. Stone, New York, N. Y.).—I have recently received a letter from Australia in relation to the Tasmanian Zinc Co., which is putting up an electrolytic plant in Tasmania. At present they have only a small unit making a few hundred pounds of zinc a day. They find that not only must the solutions be very carefully purified but the electrodes and everything about the plant must be examined and watched; the whole success depends on the purity of the electrolyte.

Jos. W. Richards.—One of the bases of my remark about manganese was the statement of Mr. French, of Nelson, B. C., in charge of the electrolytic plant there, who says that manganese in the solution assists in the purification of the electrolyte; and that he sees no reason why, from his standpoint, if there is not enough manganese in the zinc ore, low-grade manganese ore should not be treated and brought into soluble condition and the salt added to the electrolyte, for that purpose.

Zinc Mining at Franklin, N. J.*

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† Assistant Mine Captain, New Jersey Zinc Co.
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GENERAL REMARKS

The mines of the New Jersey Zinc Co. in New Jersey, are situated in the northern part of Sussex County, at Franklin, formerly Franklin Furnace, and also at Ogdensburg. The mine at Ogdensburg is still in the development stage, but the mine at Franklin, with which this paper deals chiefly, has been a steady producer of high-grade zinc ores for many years.

The town is about 50 miles from New York City, among the foothills of the Kittatinny Mountains. It is equipped with water supply (filtered and chemically treated), good roads, electric lights, efficient police protection, and excellent schools, including a new vocational high school. For a mining community, the conditions and location are far above the average.

Franklin Furnace Folio, No. 161 of the U. S. Geological Survey, describes the geology and mineralogy of the district in detail, so that merely an outline of the main characteristics of the orebody need be given. The

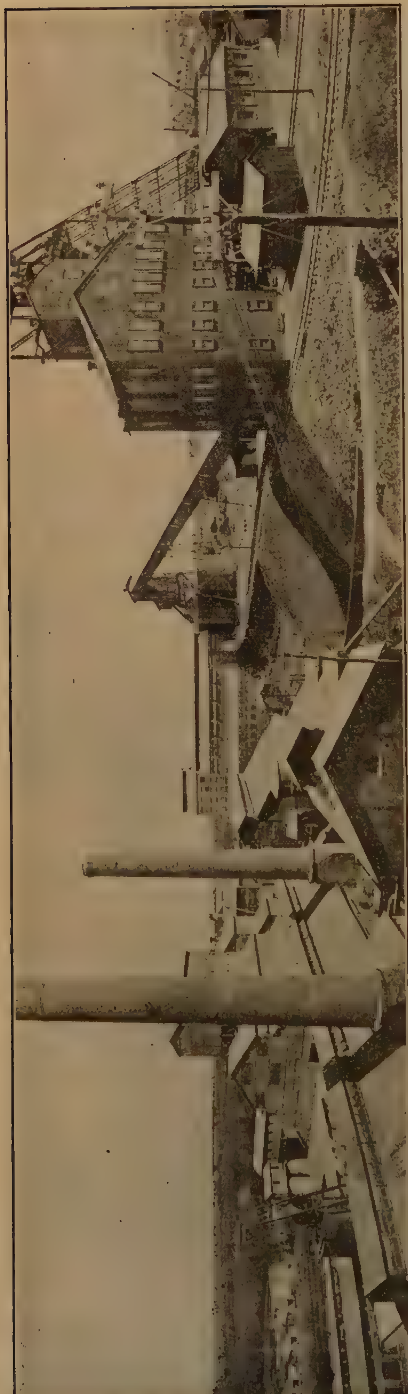


Fig. 1.

Fig. 2.

Figs. 1 and 2.—Views at Franklin.

ore is composed chiefly of the combined oxides of iron, manganese, and zinc (franklinite), zinc oxide (zincite), and the silicate of zinc (willemite). The waste rock is principally a highly crystalline limestone, though masses of feldspar are encountered at times, and numerous trap dikes cut through and across the orebody. A great variety of minerals has been found at Franklin, both in the ore and the country rock.

The shape of the orebody is best shown by the stereogram from the *Folio* in Fig. 3.

The orebody as a whole is a pitching trough, pitching to the north at a gradually varying angle of from 26° to 12° below the horizontal. The western side of the trough, or west vein or leg, as it is called, outcrops at the surface for a considerable distance, but the east vein does not. This is capped by country rock and the extent of the vein above the basin of

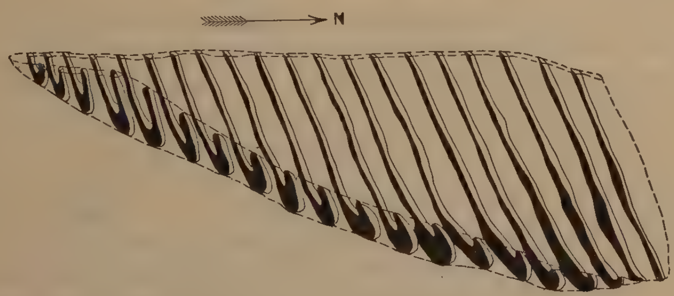


Fig. 3.—Stereogram of Franklin Orebody.

the trough lessens greatly to the north, as is clearly shown by the stereogram.

Historical Sketch

The zinc deposits at Mine Hill (Franklin) and Sterling Hill (Ogdensburg) have been known since very early times. The Mine Hill property came from the East Jersey Proprietory Co., to the heirs of Anthony Sharp, in 1750, and passed down through various owners until it was acquired by Dr. Samuel Fowler about 1815. Dr. Fowler appreciated its mineral worth, but was not able to reduce the various ores successfully. After his death his son, Col. Samuel Fowler, acquired the property having mineral value, and through his efforts, the early mining companies were organized and mining operations started. It was not until about 1865 that the ores of Mine Hill were very extensively mined. The preceding 15 years covered a period of extensive experimental operations for the metallurgical treatment of the zinc ores. The mining at this period was done at the southerly and southwesterly part of the present known deposit.

About 1888–92, diamond-drill prospecting proved the extension of this orebody northward, and a vertical shaft, known as the Parker shaft, was sunk to develop this part of the orebody. The shaft was in operation from 1894 to 1910. In 1910, Palmer shaft was put in operation and is now the main operating shaft on Mine Hill.

The history of Sterling Hill is somewhat coincident with that of Mine Hill, although the early records show that zinc ore was discovered at Sterling Hill prior to the discovery on Mine Hill, since the description of this property, known as the "Sterling Tract," when returned to the heirs of Anthony Rutgers on May 4, 1730, was called the "Copper Mine" tract. The Sterling Hill mines came to Dr. Samuel Fowler and later to his son Col. Samuel Fowler, as did the Mine Hill property, and in a general way has the same history of the early unsuccessful attempts to work the ores. The deed and court records indicate that many of the early mining companies that were formed to operate at Mine Hill and Sterling Hill were unsuccessful in their efforts. The New Jersey Zinc Co. has been operating at Mine Hill since 1897, and at Sterling Hill since 1912.

MINING METHOD

When the present management took charge of the property, a method of mining had to be adopted for the whole orebody, which would allow an increased production of ore to be maintained, while the changes necessary for a comprehensive and economical method could be put in operation. The various separate mining companies that had been working portions of the orebody had not coöperated at all; this resulted in a chaotic condition of shafts, levels, stopes, and so forth, of which only a part were useful. The equipment on the property had to be utilized for a time at least and a coördination of effort gradually reached.

An outline of the method adopted for mining follows:

1. An inclined shaft of large capacity, both in size and equipment, was sunk in the foot-wall rock, near the central point of the orebody. This shaft, sloping 47° 30′ below the horizontal, approximately parallels the dip of the orebody, so that it remains in the foot-wall rock.

2. Levels were established at approximately 50-ft. intervals; where former openings could be included, they were used.

3. Stopes 17 ft. (5.18 m.) wide with center lines at approximately right angles to the strike of the orebody (called transverse stopes), of the shrinkage type, or back stopes, were carried up from level to level, beginning at the bottom. These stopes were placed at approximately 50 to 55 ft. (15 to 16 m.) centers. A stope was carried from one level to the next above, for its whole length (this length was determined by the distance from foot wall to hanging wall), drawn out, and filled. When the filling was completed, the stope was carried up to the next level and

the cycle of operations continued until the surface or top of the orebody was reached.

4. The pillars between the stopes were left standing between the fill until the final completion of the stopes. A part are being mined from the top down by a caving method modeled after one of the top-slicing systems.

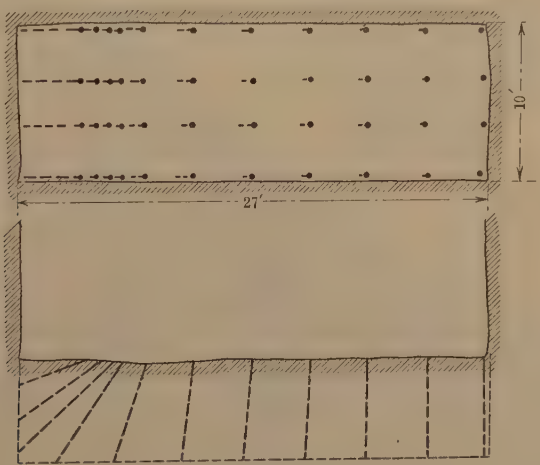
5. Transportation is by hand-tramming, in the main, for short distances to ore passes from which on certain levels the ore is collected by electric haulage and conveyed to the chutes leading to the loading pockets. The filling material is introduced into the mine through an old shaft and transferred from this shaft by electric haulage to distributing raises.

The details of the methods involved in the above mining system and particular exceptions made to suit special conditions are treated in the following text.

MINING OPERATIONS

Shaft

The main shaft, called Palmer shaft, in honor of S. S. Palmer, the former president of the New Jersey Zinc Co., is an incline, sloping 47° 30′ from the horizontal, 27 by 10 ft. (8.23 by 3 m.) outside of timbers. The



The Next Round will be Reversed

Fig. 4.—Shaft sinking round.

shaft is equipped with four hoisting compartments, two of which are used for hoisting ore and two for handling men and supplies. All four of the compartments, however, are equipped for hoisting ore.

The shaft was started Nov. 10, 1906, and the complete equipment ready for hoisting ore from the 800 pocket on Apr. 27, 1910, and from the 1150 pocket on Oct. 15, 1910. Sinking was done with 3-in. piston

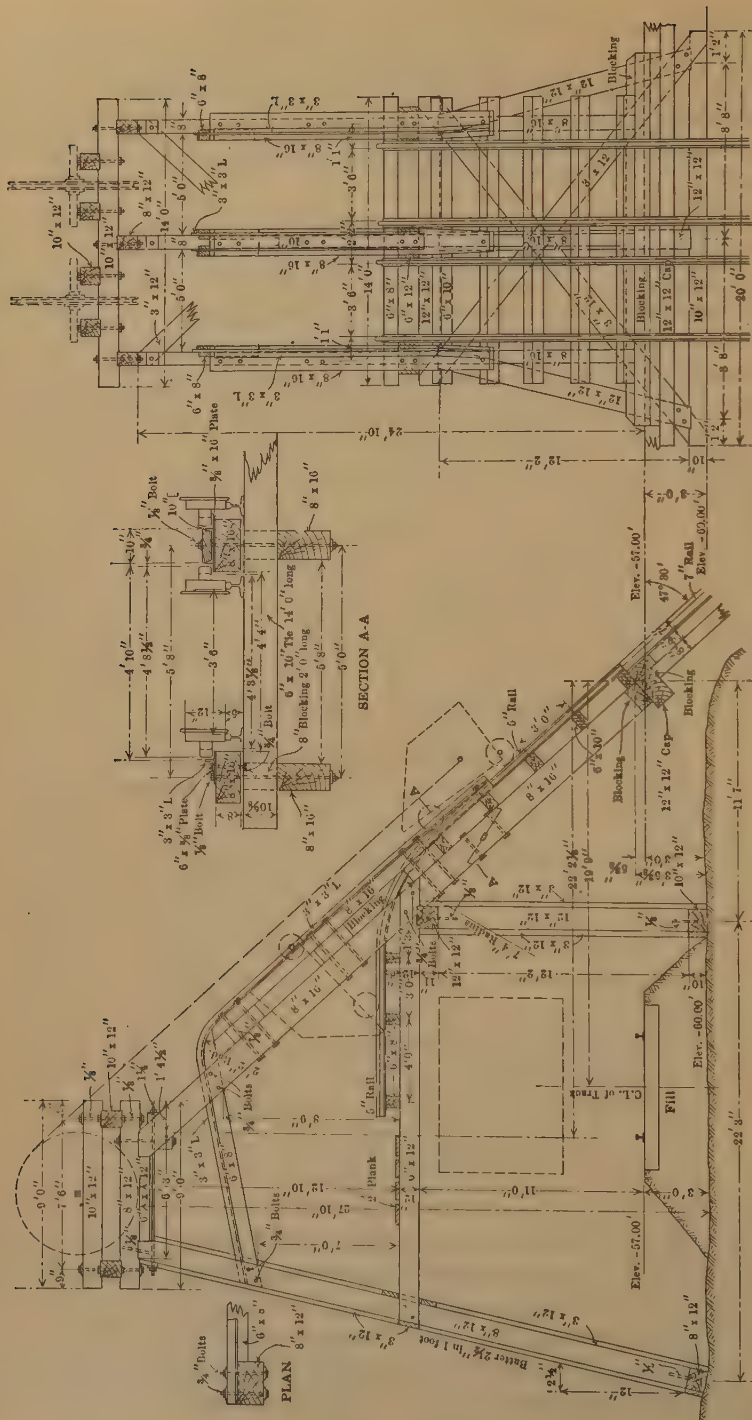


FIG. 5.—TEMPORARY HEADFRAME, PALMER SHAFT.

drills, generally mounted on bars, but in several places where the ground broke away too far, tripods were used. At the start of the shaft sinking, a V-cut was used, but later on it was found better to substitute a method of breaking to one end of the shaft beginning with short flat lifting holes, and gradually lengthening and steepening the holes, ending with holes practically parallel to the center line of the shaft (Fig. 4).

The work was done with three shifts. The day shift did the drilling, which was finished by the afternoon shift, if necessary. The firing was

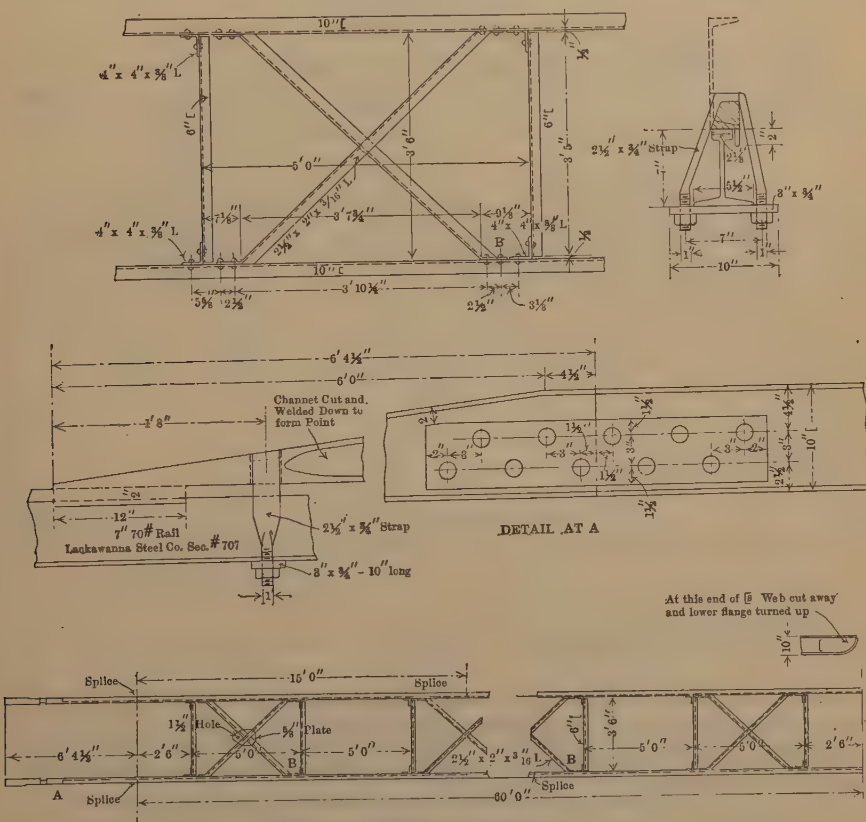


FIG. 6.—EXTENSION TRACK.

done by the afternoon shift, which then mucked until relieved by the night shift. The night shift completed the mucking and then lowered the equipment and prepared for the next day's drilling. The holes were all fired with fuse. The timbering was done on the day shift. A temporary headframe of timber was used during the sinking operations (Fig. 5). Over this the permanent steel headframe and rough crusher was constructed without interfering with the sinking operations.

As soon as possible, a series of raises was run up in the line of the shaft

from crosscuts from the lower levels of the mine, which allowed the muck to be thrown down the raises from which it was taken and disposed of as fill in the stopes of the mine. The raises also did away with the need of sinking pumps, and helped the ventilation of both the mine and shaft. Before the raises were holed into the shaft, the muck was hoisted to surface in small skips; a false set of track of a usual type that could be hoisted above the danger zone was used at the bottom of the shaft (Fig. 6).

Timbering

The sketches, Figs. 7, 8, and 9, show the general arrangement of the shaft timbers and track. Figs. 7 and 8 apply above the pump station, 1050 level, but below that a fifth compartment is set off to provide a

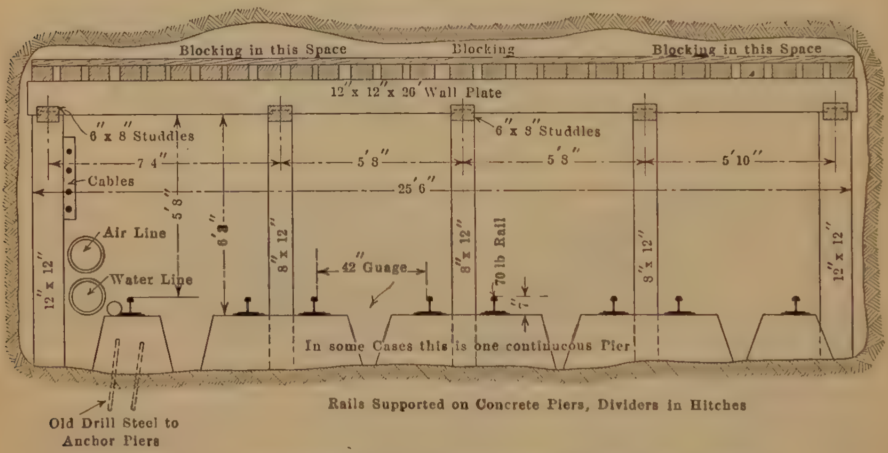


FIG. 7.—CROSS-SECTION OF PALMER SHAFT ABOVE 1050 LEVEL.

stairway for the pumpman to reach the bottom pumps. Fig. 9 shows this. The timbers are long-leaf yellow pine. A stull (12 by 12 in.—304.8 mm.) on each side of the shaft, supports each set as a bearing piece; the side dividers and wall plate are also 12 by 12 in. but the interior dividers are only 8 by 12 in. (203.2 by 304.8 mm.), the studdles are 6 by 8 in. (152.4 by 203.2 mm.). Lagging of 6 by 8-in. square timber spaced 16 (0.4 m.) or more inches apart, extend from wall plate to wall plate, and are held from slipping by ¾-in. (19-mm.) pins which catch on the wall plate. This lagging supports a layer of 2-in. (15-mm.) plank, above which short blocking and pieces of split timber are packed to fill the space to the hanging wall.

The designs of the shaft stations are shown on Figs. 10, 11, and 12, which show those of steel, and those of timber. In the design of the shaft, provision is made for loading and unloading men at a station imme-

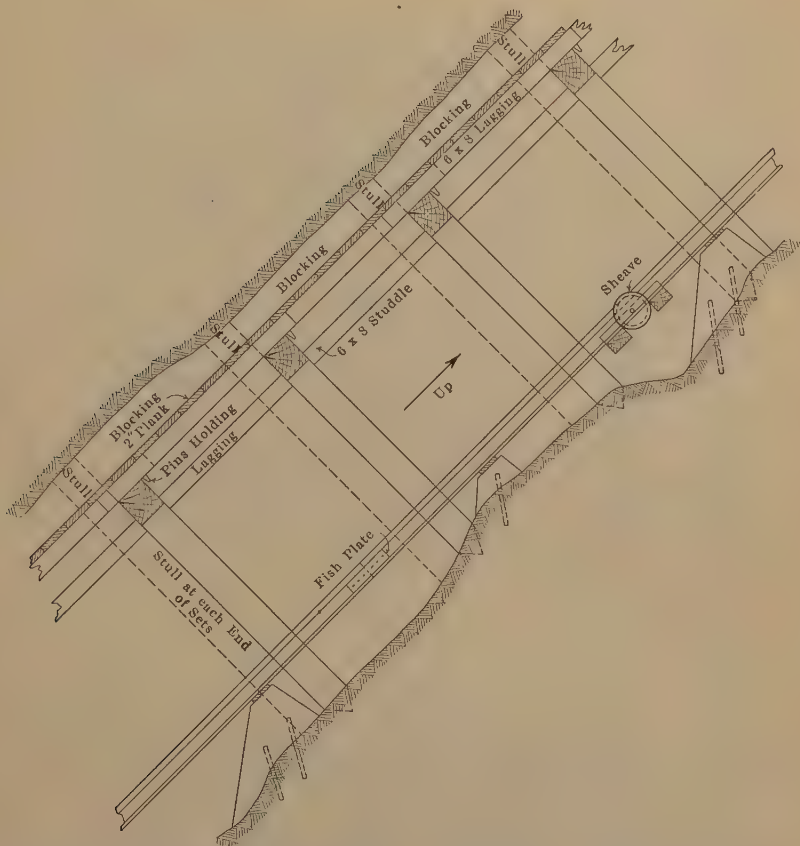


Fig. 8.—Longitudinal section of Palmer shaft above 1050 level.

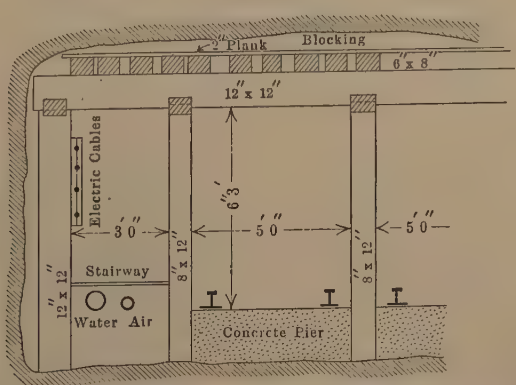


Fig. 9.— Cross-section of Palmer shaft below 1050 level.

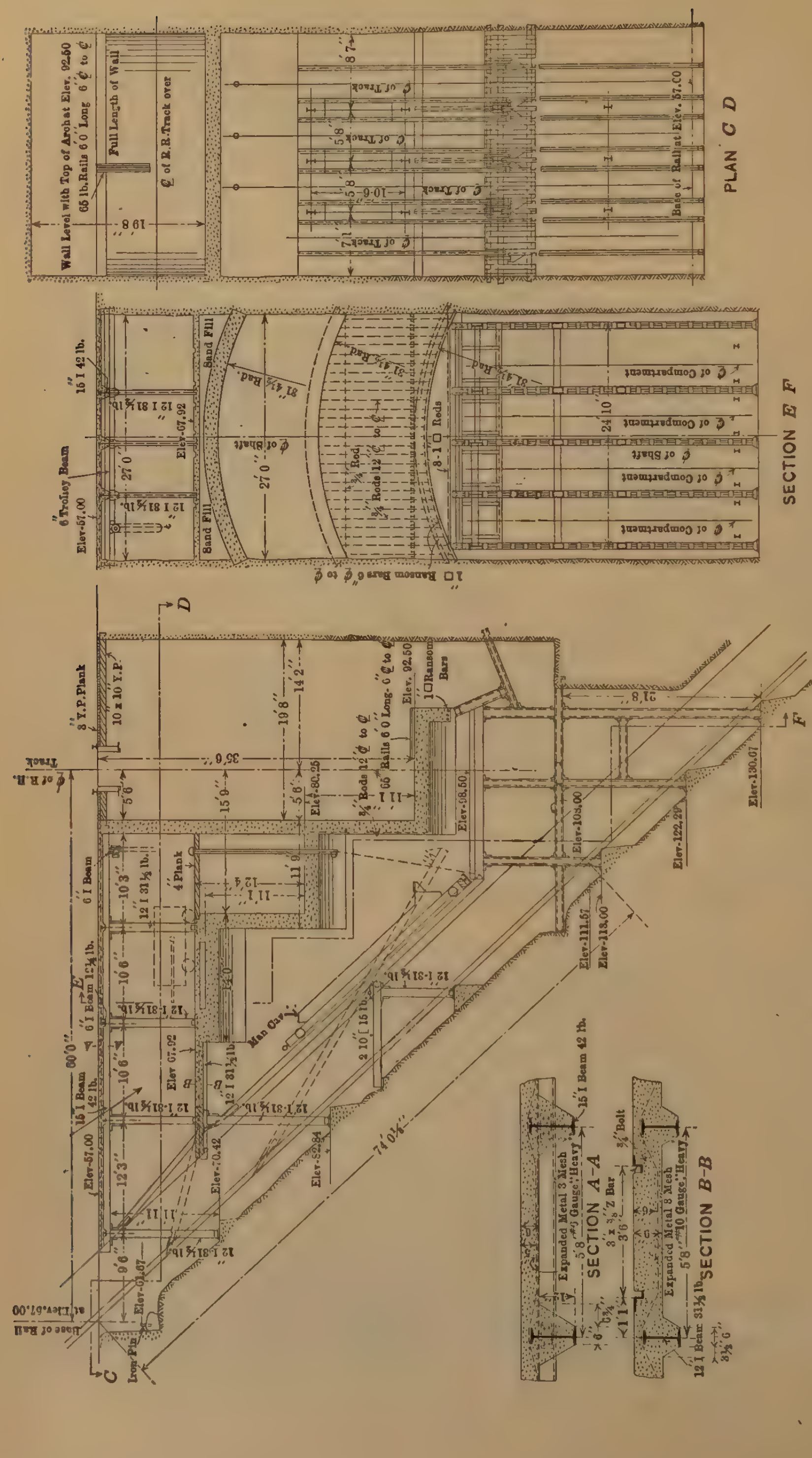


FIG. 10.—TOP LANDING, PALMER SHAFT, AND LOADING POCKET.

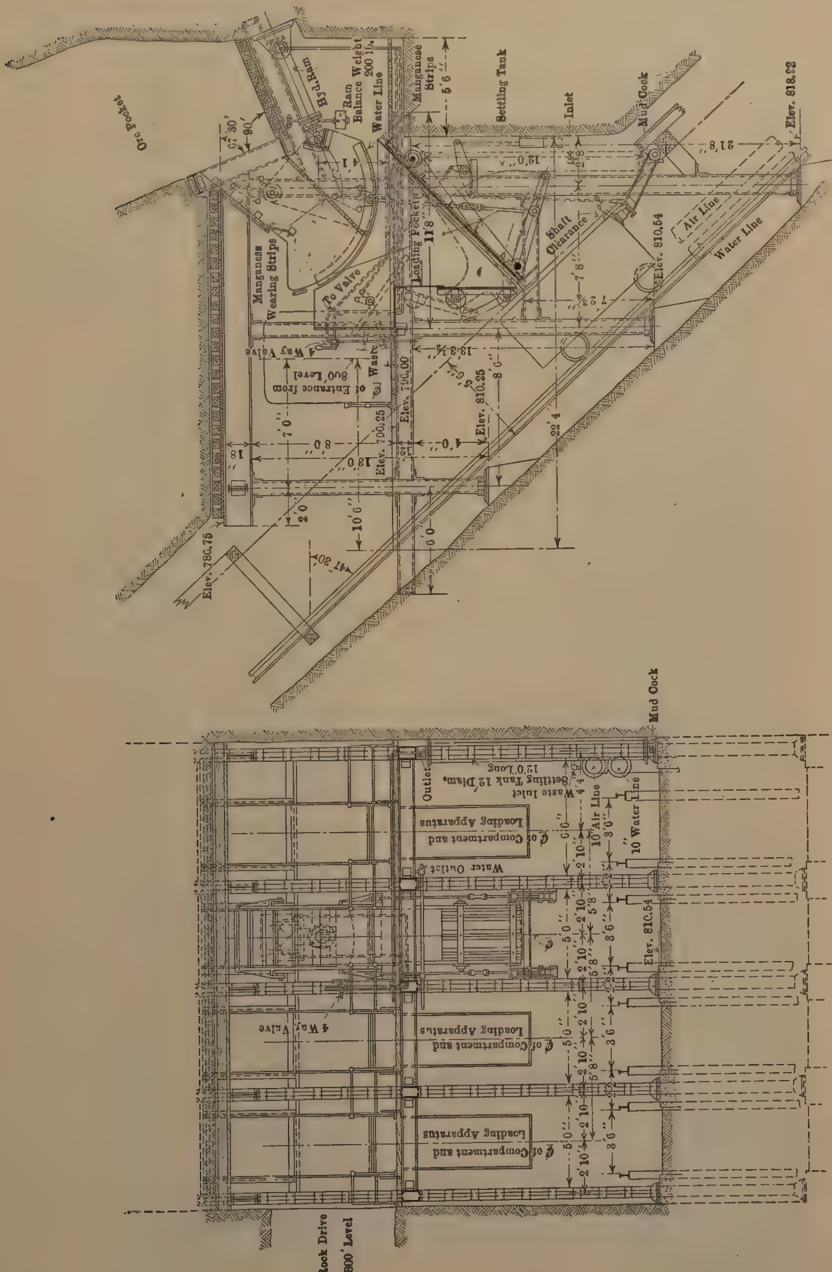


FIG. 11.—PALMER SHAFT, 800 STATION, AND LOADING POCKET.

diately below the collar of the shaft. A covered incline connects this station with the change house so that the men do not have to go into the open between the shaft and the change house, yet no mine vapors reach the change house.

The arrangements for interchanging skips and man cars are shown in Fig. 10 and are self-explanatory. The complete change can be made in 12 min.

During the sinking of the shaft, a derrick was installed near the collar for handling supplies, timber, etc. This has a boom 54 ft. (16.5 m.) long.

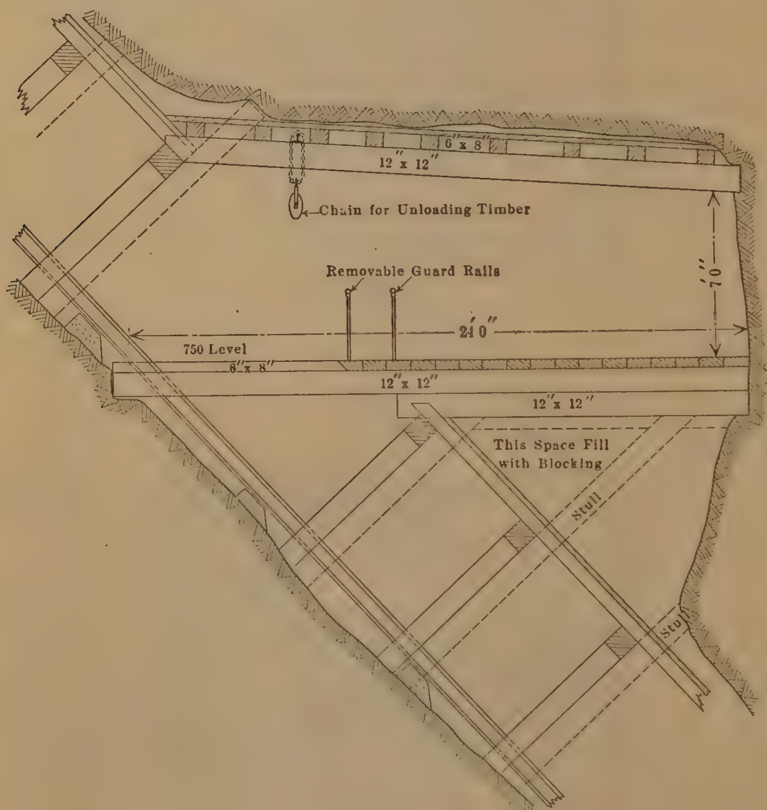


FIG. 12.—TIMBER-SHAFT STATION.

This derrick, shown at *E*, Fig. 13, was found to be so convenient that it has been used ever since for loading timber and other supplies, and also for changing skips and man cars; the other arrangements for this purpose have therefore seldom been used, though of course they are ready should an accident put the derrick out of service.

Stations are not placed at every level, but only at the surface, 300, 750, 800, 950, 1050, and 1150 levels. Access to the other levels is ob-

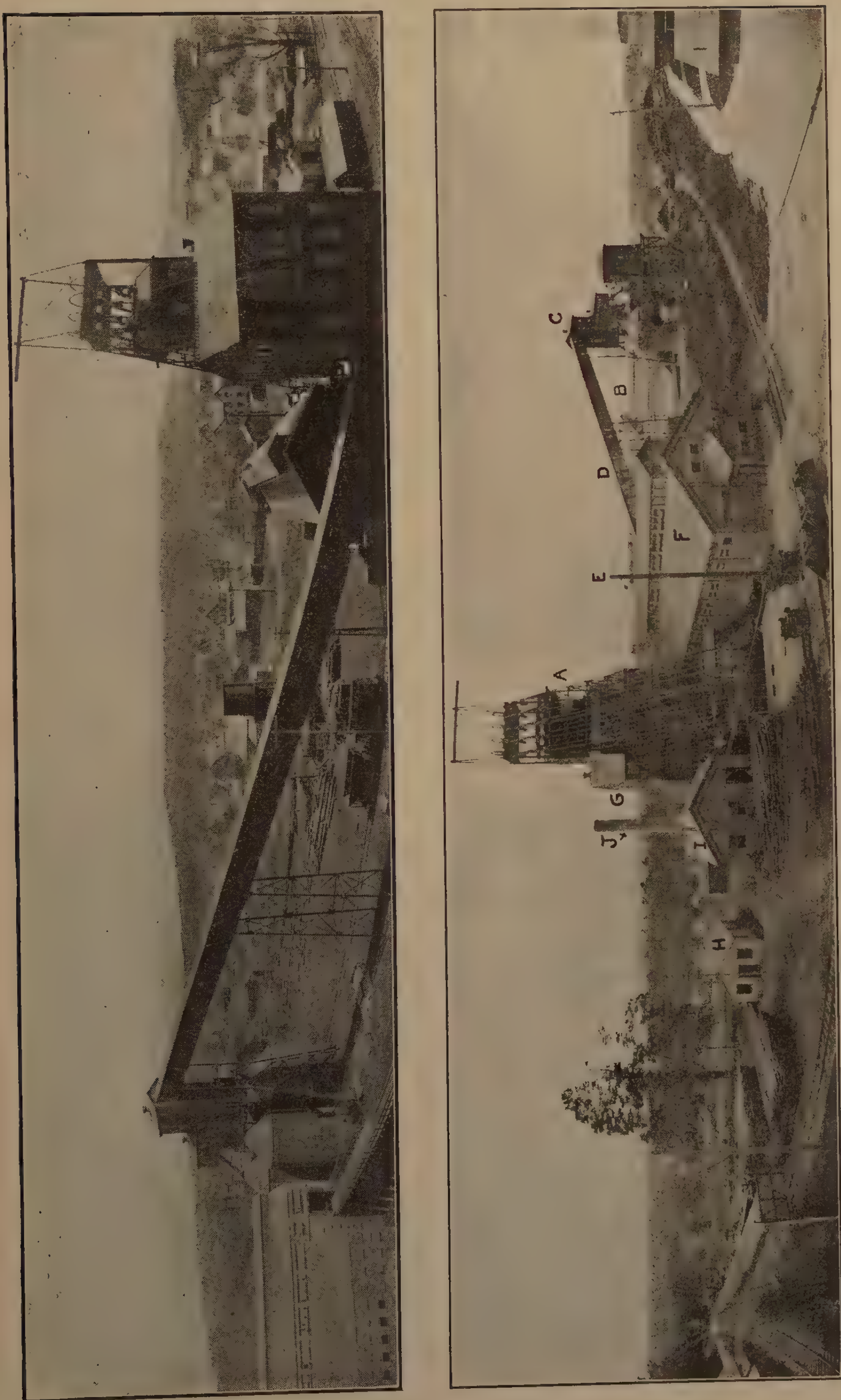


FIG. 13.—PANORAMAS OF SHAFT AND CHANGE HOUSE.

A. Head frame.
B. Mills.
C. Storage tanks.
D. Ore conveyor.
E. Derrick.
F. Change house.
G. Coarse crusher.
H. Laundry and mine supply.
I. Blacksmith shop supply.
J. Power house, below hill.

tained by ladderways or stairways in raises in the orebody. Timber slides are also constructed through the orebody for the transfer of timber to the intermediate levels.

Track

The method of making and supporting the track in the shaft differs greatly from the usual one of supporting railroad rails on continuous longitudinal stringers of either timber or concrete, or both. It is designed to avoid the steady pound and shock to the equipment that the usual methods incur. Rails of section shown in Fig. 14, weighing 70 lb. per yard, 30 ft. (9.1 m.) long, are used. These are supported on concrete piers built to grade, spaced at 10-ft. (3-m.) centers. The rails are bolted to a steel plate ½ in. (12.7 mm.) thick and 8 in. (203.2 mm.) wide, which in turn is bolted to the concrete piers, shown in Figs. 7 and 15, by

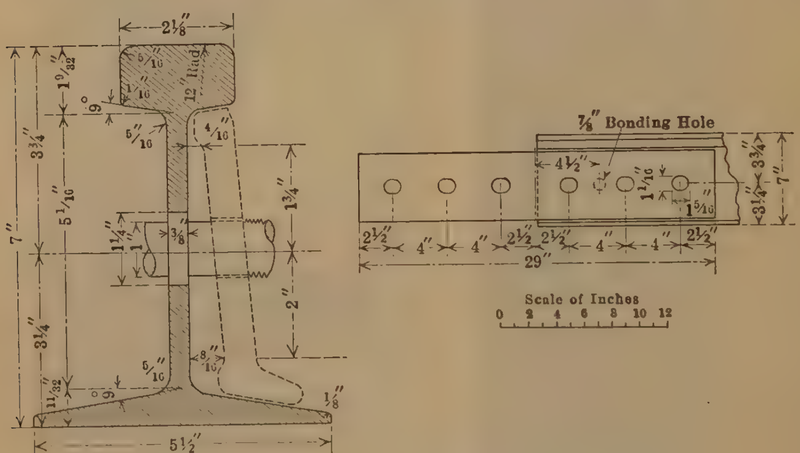


FIG. 14.—SECTION OF 70-LB. RAIL FOR SHAFT.

¾-in. (19-mm.) anchor bolts. The rails join at a point between piers. Splice bars of a type suited to the rail section are used to join the rails. This construction has had a low maintenance cost, but after about 7 years, service is beginning to show need of repairs. New rails will soon have to be put in for the ore tracks, and 80-lb. rails of the same depth have been chosen for this service.

Hoisting

The hoisting skips have a full 6-ton capacity, weigh about 4500 lb. and are built as shown by Fig. 16; they are usually operated "in balance." Two 22 by 48-in. (0.56 by 1.22-m.) duplex, direct-acting double-drum, Allis Chalmers hoists with Corliss valve gear and drums 5 ft. 6 in. (1.67 m.) wide and 10 ft. (3 m.) diameter are used. One hoist operates the ore skips and the other the man cars. On each engine one drum is keyed to

the shaft while the other is held by a Lane-type friction clutch, which allows the drums to be operated independently. The skips or man cars can therefore be "spotted" for any station, and hoisted "in balance" without one skip having to go to the bottom of the shaft every time the other goes into the dump. The engines are set to cut off at about 9/10 stroke and use steam at 145 lb. pressure at a rate of flow reaching a maximum of 45,000 lb. per hour. At the present time the exhaust goes to the atmosphere, but a Rateau regenerator is being installed to utilize the steam at 0 to 4 lb.- gage pressure in the low-pressure stages of a high-pressure turbine. The hoists are provided with an automatic device to prevent overwinding.

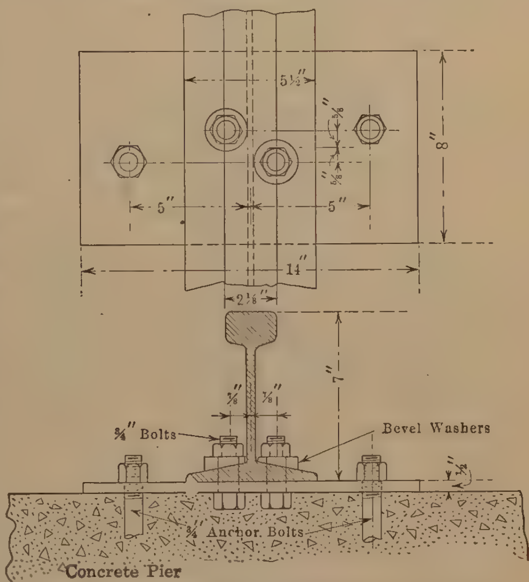


FIG. 15.—METHOD OF ATTACHING SHAFT RAILS TO PIERS.

The hoisting ropes are 1⅛ in. (28.6 mm.) diameter, plow steel, 6 by 19 regular lay and 6 by 7 Lang lay. They are examined each day before regular hoisting starts, and at this time the rope is run very slowly past the inspectors, who look for broken wires or other signs of weakness. A lubricant is used on the rope, but it is such that it does not coat the rope so that the wires cannot be seen. The rope is attached to the bale of the skip with a thimble and clevis, not with a socket. Seven clamps are used to fasten the rope and these are all so placed on the rope that the *U*-bolt part of the clamp bears on the turned up end of the rope and the broader piece bears on the main rope (Figs. 17 and 18).

Tests have indicated that this method of attachment permits the support of a load equal to the ultimate strength of the rope, but when the rope clamps are alternated the rope may fail at 80 per cent. or less of its normal strength because of the concentration of stresses where the *U*-bolt constricts the standing portion of the rope. Experience with various types of ropes has given results favoring a plow-steel Lang-lay rope of 6 strands and 7 wires per strand for use where the sheave wheels and hoist drums are of sufficient diameter to avoid excessive bending (in this case 12 ft. (3.66 m.) and 10 ft. (3.04 m.) respectively), since a greater reduction of diameter by wear may be suffered without fracture of

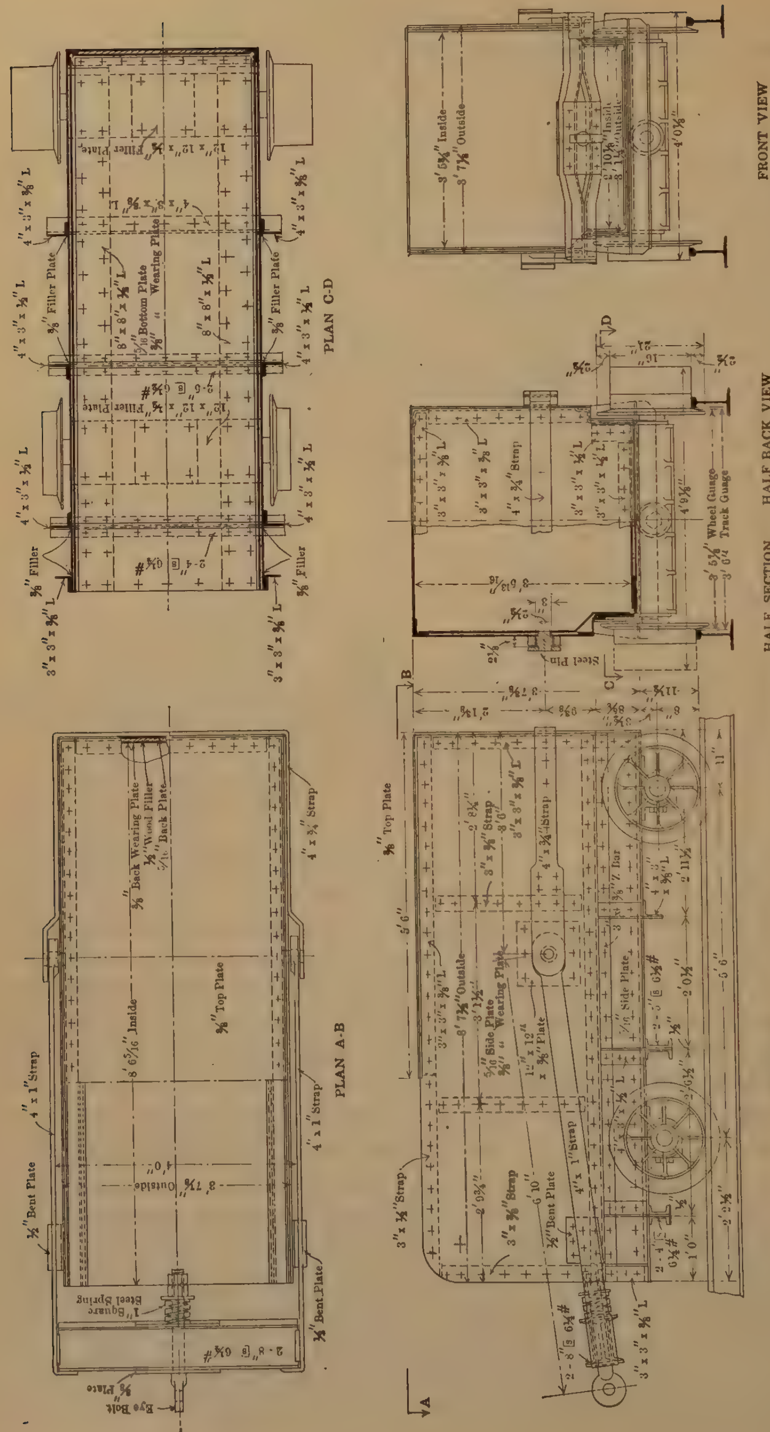


FIG. 16.—PALMER SHAFT SKIP.

the wires. This wearing service is, of course, more important in a slope shaft than a vertical one. With the regular lay ropes first installed, the type selected had a special arrangement of large wires at the outside of each strand (known as Seale patent lay), and this type of rope gave good service. Ropes with Seale's patent lay of wires have also been tried in the Lang-lay type but they increased in diameter at various

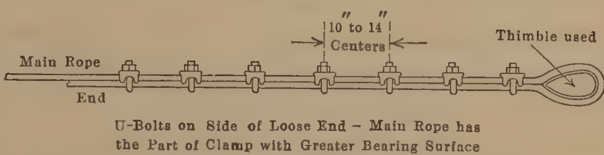


FIG. 17.—A METHOD OF ATTACHING ROPE CLAMPS.

points with a loosening of the outer wires (or "bird-caging"), so the stress was carried largely by the few inner wires. This condition was undoubtedly aggravated by the fact that the hoist drums will not hold all the rope in one layer, so there is a beating or slapping effect as the second layer is guided over the first layer. "Bird-caging" is also more severe when a swivel attachment of a 6×19 Lang-lay rope is made to the bale of the skip, because the twists of both the wires in the strands and the strands of the rope are in the same direction of rotation. Therefore, as the rope is stretched the pitch of the strands becomes greater, and the wires in the strands are untwisted a certain amount. The outer wires of the strands are lengthened in this manner disproportionately with the inner wires and "bird-caging" is the result. If there are only seven wires in the strand there is only one layer about the core wire, so the above effect is not produced. With a regular-lay rope a swivel is an advantage as it permits the rope to rotate as its length changes, and, since the wires in the strand are twisted in the "opposite hand" to the twist of the strands of the rope, they tend to counteract the untwisting of the rope and

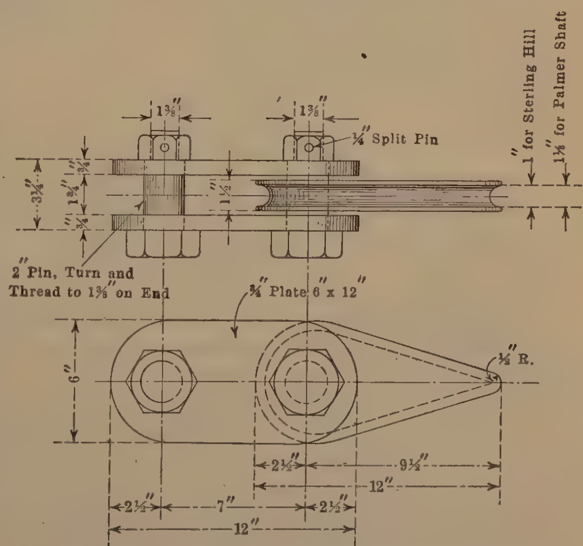


FIG. 18.—CLEVIS AND THIMBLE FOR SHAFT SKIPS.

their outer wires are made tighter. As a result of the excellent manner in which it is standing up under severe service, the 6 × 7 Lang-lay ropes are preferred in slope shafts provided with well proportioned head-sheaves and hoist drums.

Table 1 gives a record of service of various types of rope with the corresponding foot-tons of work performed. These ropes were taken out of service when the appearance of a number of broken wires in proximity made this safety precaution appear advisable. Subsequently breaking

TABLE 1.—*Record of Hoisting Ropes in Palmer Shaft*

Type of Rope	Period in Service	Tons Ore Hoisted	Foot-tons of Work Performed	Remarks
6 by 19 Seale patent, ordinary-lay				
South rope........	Apr. 27, 1910 to Feb. 8, 1912 or 650 days.	315,759	471,758,554	*A*
North rope (above plus)	Nov. 4, 1913 to Sept. 20, 1914 or 320 days.	200,864	326,441,347	*B*
Total North rope	970 days	516,623	798,199,901	
6 by 19 Seale patent, Lang-lay South rope	Feb. 8, 1912 to Nov. 4, 1913 or 634 days.	408,867	651,957,881	*C*
North rope.......	Feb. 8, 1912 to Sept. 20, 1914 or 954 days.	609,732	978,399,228	*D*
6 by 7 Lang-lay each rope North and South the same.	Sept. 20, 1914 to June 1, 1917 or 983 days, and still in service June 28, 1917.	955,470	1,468,287,514	*E*

REMARKS.—Foot-tons of work does not include weight of skip empty and rope when lowering. The above figures should be increased approximately 38 per cent. to cover this item.

A. North and South ropes taken out of service because of the number of places broken wires showed; the maximum number of broken wires within one foot was 12.

B. North rope returned to service as an emergency after Lang-lay 6 × 19 North rope became badly "bird-caged" and had a number of broken wires in one strand ravelling, since breaking tests of sample gave confidence in the rope.

C. Lang-lay South rope removed for reason given in remark "B." It was turned end for end on April 4, 1913.

D. Lang-lay 6 × 19 North rope "bird-caged" considerably after it was in service a short time but had few breaks of wires until Sept. 20, 1914, when there were possibly 16 breaks in three feet.

E. Lang-lay 6 × 7 North and South ropes were turned end for end on July 6, 1916, as a matter of policy but show not more than four broken wires each at present, and these are now wrapped on the drum. The North rope has suffered severe stresses twice because of skip derailments during hoisting. They appear in excellent condition.

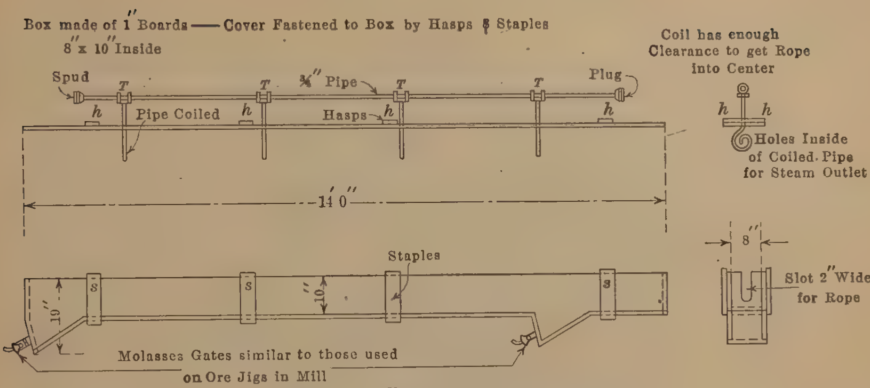


FIG. 19.—ROPE-STEAMING BOX.

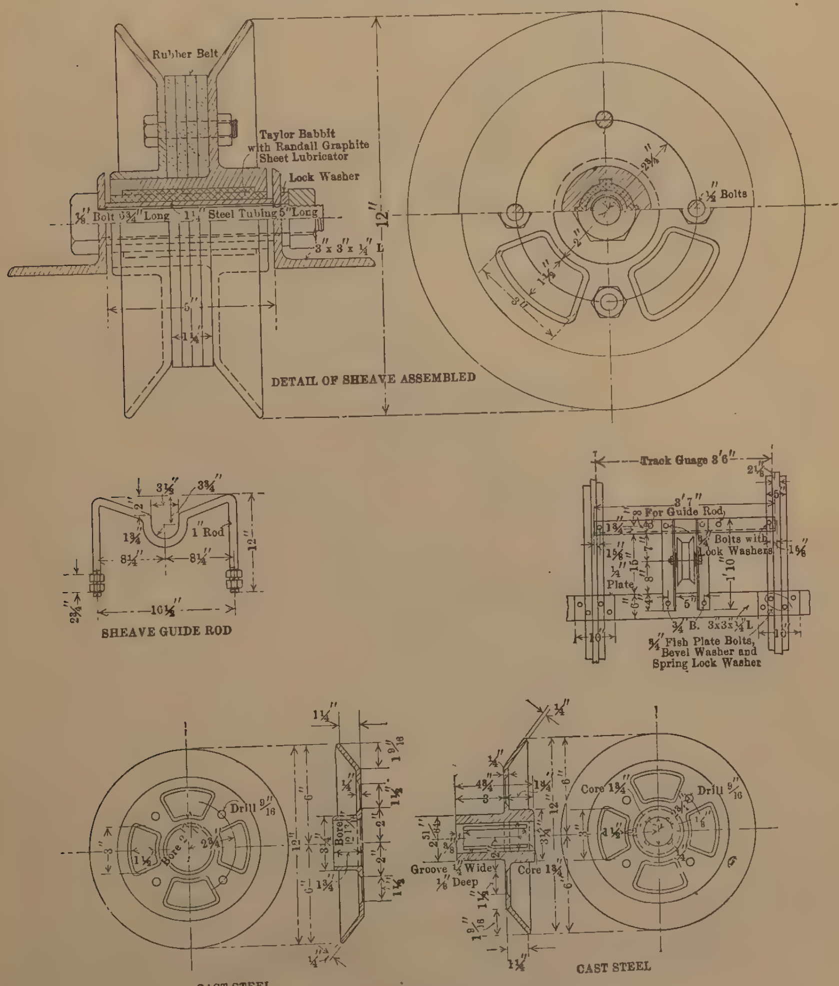


FIG. 20.—IDLER SHEAVES.

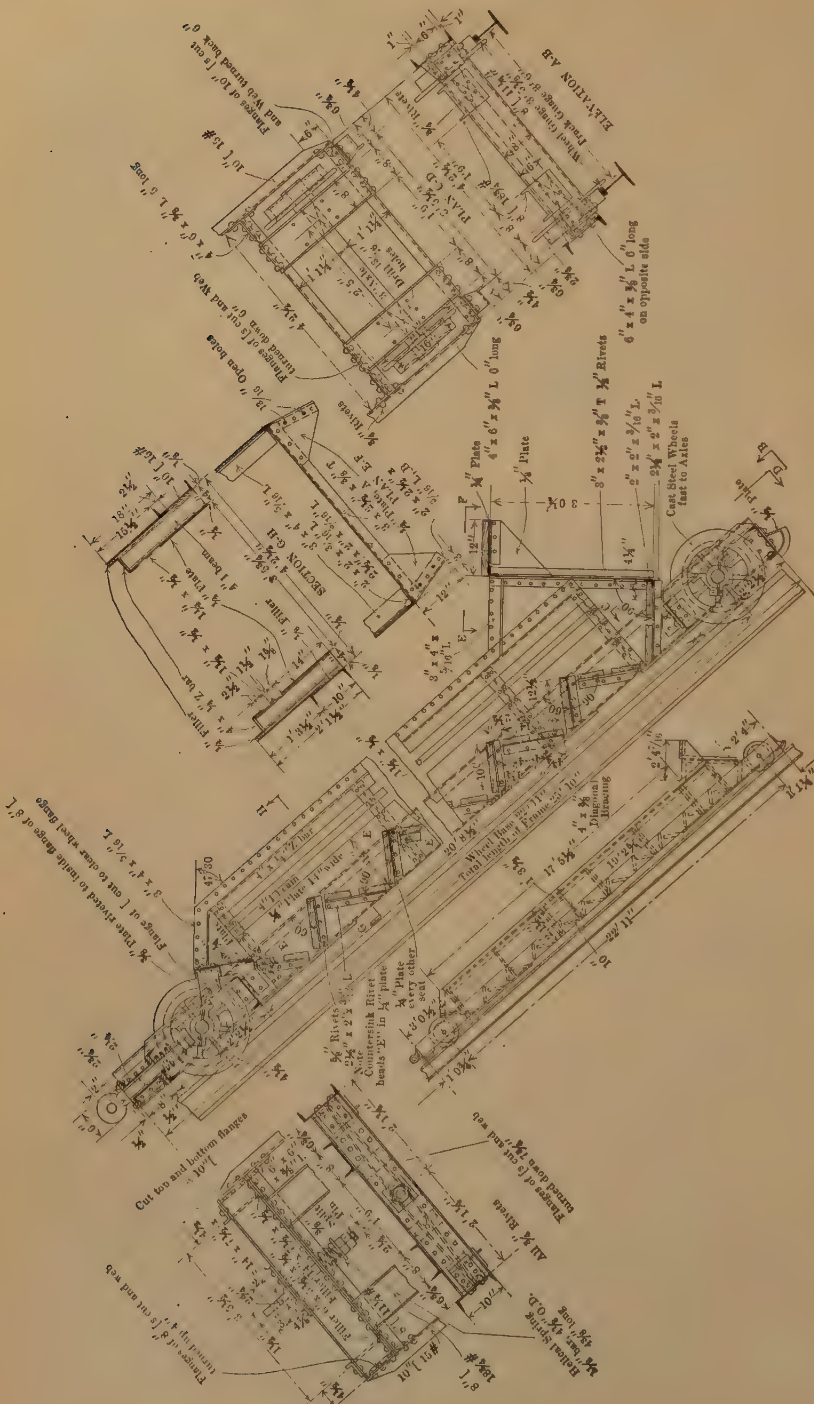


FIG. 21.—PALMER SHAFT MAN CAR.

tests of the same ropes in one instance showed that 20 wires could be severed in the various strands within a length of 1 ft. and the rope would retain 64 to 81 per cent. of its initial strength, and if 12 wires were broken within 1 ft. the strength remained 90 to 93 per cent.

All of the ropes are 1⅛ in. (28.6 mm.) diameter with plow-steel wires, and support a skip weighing empty about 2¾ tons, with an average ore load of about 6 tons extra, and an average weight of "rope out" of about 1000 ft. (304.8 m.) or 1 ton. They are wound over 10-ft. (3.04-m.) diameter drums and 12-ft. (3.66-m.) diameter head sheaves and are supported in the shaft by 9-in. (228.6-mm.) diameter idler pulleys spaced 30 ft. (9 m.) apart on the shaft slope of 47° 30′. Each rope is 2000 ft. (609.6 m.) long.

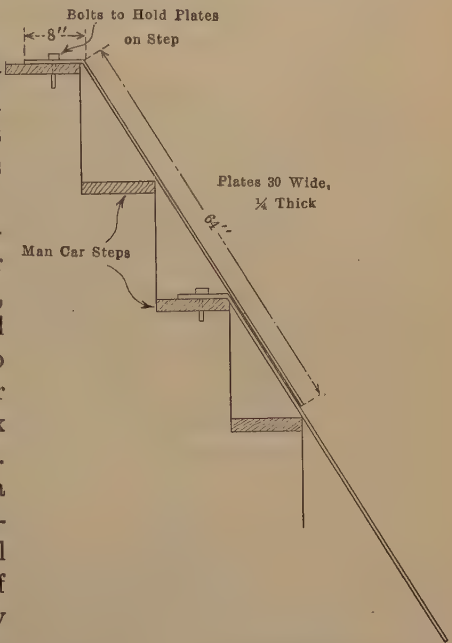


FIG. 22.—PROTECTION PLATES FOR MAN CAR.

In order to permit proper inspection and prepare the rope for impregnation with fresh lubricant, to avoid the considerable internal friction and wear which is likely to take place between the interior wires, the type of cleaning box illustrated in Fig. 19 is employed. A perforated pipe is made in a spiral and is supplied with high-pressure steam emitted in a conical shape (formed by a number of small jets). The rope passes slowly through this cone of steam which softens and scours off the rope dressing. Further cleansing may be done with rags and kerosene or gasoline.

Rope Idlers.—In order to preserve the hoisting ropes from serious abrasion, idler sheaves of 9-in. (228.6-mm.) tread diameter are mounted at 30-ft. (9-m.) intervals in each shaft compartment in conjunction with specially shaped guide frames to deflect the whipping rope onto the idlers. As these reach a speed of 1200 r.p.m., it is almost as important to avoid their turning too freely as it is necessary to have them start easily. A babbitted bearing with graphite inserts has proved most satisfactory, as it does not revolve quite so freely as ball or roller bearings previously tried and yet is cheap to maintain. The original idlers were lined with hard-wood segmental blocks so placed that the rope was in contact with the end grain of the wood, but these blocks wore out rapidly and could not compete in wearing properties with fillers of old rubber belting. Fig. 20 illustrates the details of construction of these idlers and guides.

Man Cars.—The man cars or cages used for lowering men and supplies into the mine are shown in Fig. 21. They are built of steel with the steps of 2-in. (50.8-mm.) plank, protected by 2 by 2-in. angle iron on the edge. Three men stand on a step so that 30 men can be loaded at once. When timbers or supplies are sent into the mine, steel plates, which overlap each other, are placed over the wooden steps for protection, these plates being held in place by pins which go through holes in the plates and in the steps. In addition to protection of the steps on the man car, the plates facilitate the unloading of timber (see Fig. 22).

Loading Pockets

Hoisting is done from three places, from a pocket of 1500 tons capacity just below the collar of the shaft and 140 ft. in elevation below the tipple (into which ore sent in by railroad is dumped), from a pocket of 1340 tons capacity at the 800 level and 828 ft. below the tipple, and from a pocket of 170 tons capacity located below the 1150 level at the bottom of the mine and 1187 ft. in elevation below the tipple. At the two latter places, steel loading pockets are placed in each shaft compartment and for the top pocket only two compartments are so equipped. These consist of two parts. The lower hopper, which holds just a skip load, can be filled while the skips are moving, so that it is ready for discharge as soon as the skip is placed. Reference to Fig. 11 will make the operation clear. The upper chute, which feeds the lower one, is left with its gate open until it is time to load the skip. The gate to this pocket is closed by the same operation that opens the gate of the skip hopper. A hydraulic ram, deriving its power from the water column in the shaft, operates the gate levers. The direction of motion of the ram is controlled by a four-way valve on the loading platform. The ore cannot overflow from the feeding chute into the pocket, as the arc gate raises it to its angle of repose. The pocket cannot jam because the feeding chute gate is swung up through the pile of ore, and the lower gate cannot because it comes back on an empty hopper; also, the system of levers which move the two gates so operate that the upper arc cuts off the feed before the lower door begins to open and the lower door closes before the upper arc

TABLE 2

Level	Actual Hoisting Time, Hours	Skips Hoisted	Tons Hoisted	Skips Per Hour	Tons Per Hour	Tons, Skip	Average Hoisting Distance
800	14.50	780	4,256.57	53.8	293.5	5.46	1,122
1150	11.00	468	3,026.50	42.6	263.8	6.19	1,610
800 } 1150 }	10.00	500	{ 2,542.00 / 537.00 }	50.0	307.9	6.16	1,202
			3,079.00				

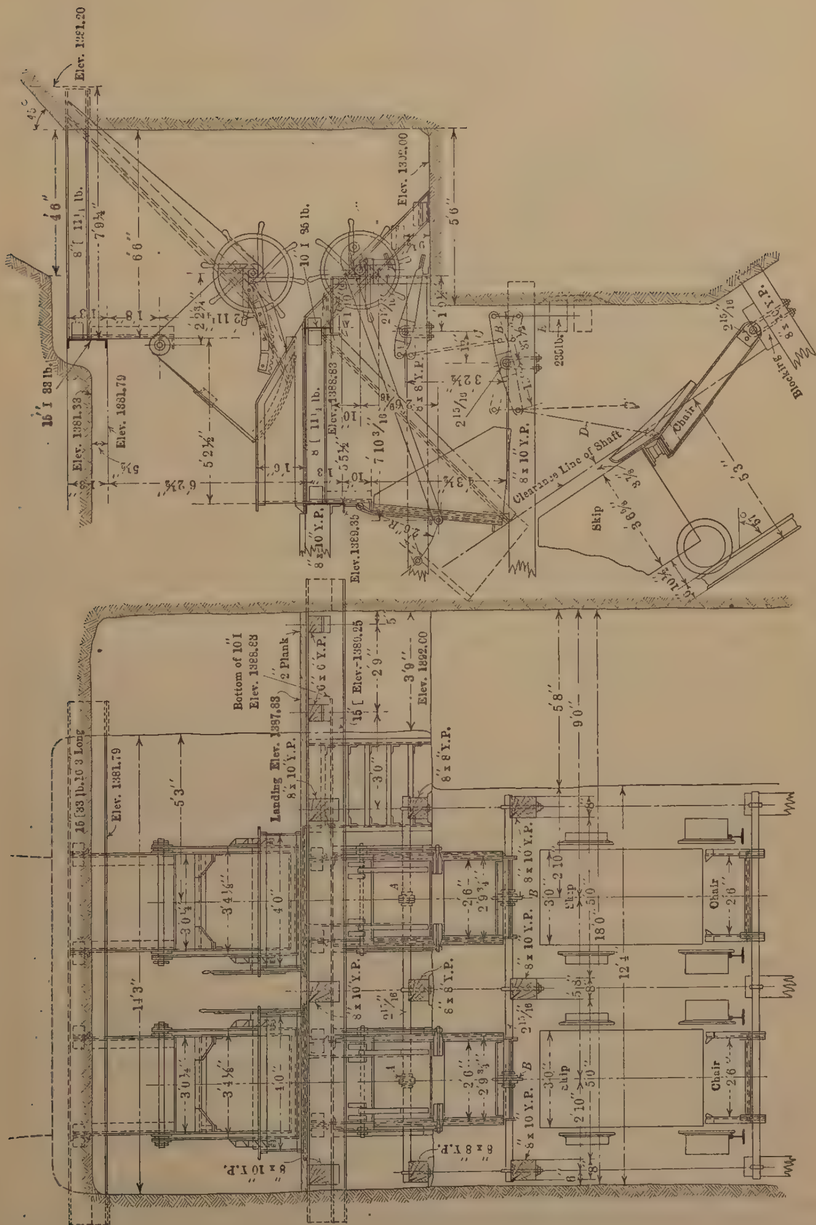


FIG. 23.—LOADING POCKETS FOR OGDENSBURG SHAFT.

has swung below the angle of repose of the ore in the chute. Only one man is actually needed to operate a pair of pockets, as the valves work easily, but a second man is stationed in the shaft below the loading platform to trim the skip. The second man also helps in case the chunks block in the throat of the feeding chute. By means of these loading devices, the idle periods of the hoist are reduced to 10 or 15 sec. to cover simultaneous loading and dumping. Several of the best records made with this equipment with two men on the pockets are shown in Table 2. The best monthly hoist occurred in May, 1915, when 76,017.78 tons were delivered to the mill.

The hydraulic loading pockets described above are shown in Fig. 11, while the equivalent manually operated loading pockets, used at Sterling Hill mine, are shown by Fig. 23.

Headframe

The headframe shown in Figs. 13 and 24 is of steel and includes the rough crusher house. The skips dump near the top; the ore slides over grizzly bars into a trommel with 2-in. holes in which it is washed for picking. The fines drop through a chute to the conveyor belt carrying it to the fine crusher-feed storage tanks, but the oversized ore from the trommel drops onto a revolving picking table. This table is annular in shape, of 30 ft. (9 m.) outside diameter, and 4 ft. (1.2 m.) wide; it revolves at the rate of about 35 ft. (10.6 m.) per minute. Men are placed at this table to rake out all the rock, timber, etc., into chutes leading to the rock bin, leaving the ore to move on to the plow which scrapes the ore into a No. 8 K Gates gyratory crusher, in which it is reduced to 2 in. or smaller. From the crusher, the ore goes to a 30-in. (0.76-m.) conveyor (about 300 ft. (91 m.) centers of head and tail pulleys) which transports it to the mill tank. This is the same conveyor to which the fines come. All the ore passing over this conveyor is automatically weighed by a Merrick conveyor weightometer (see Fig. 24).

The hoisting engine house is 130 ft. (39 m.) in the rear of the headframe, and, as the engines are wider apart than the distance between the centers of the track system, a special arrangement of the equipment had to be made. The engines, while placed side by side, are not parallel, but their drums are placed at right angles to the lines leading to the centers of the tracks they control at the top of the headframe, the sheaves being placed in their proper position to give the desired spacing.

Signals

The signal system is a direct-current electric one. On each station there is a signal box which is rung by pulling a handle. When rung, the bells ring in every signal box in the shaft serving one hoisting signal, but no bell rings to the engineer. A cageman is always on duty with the

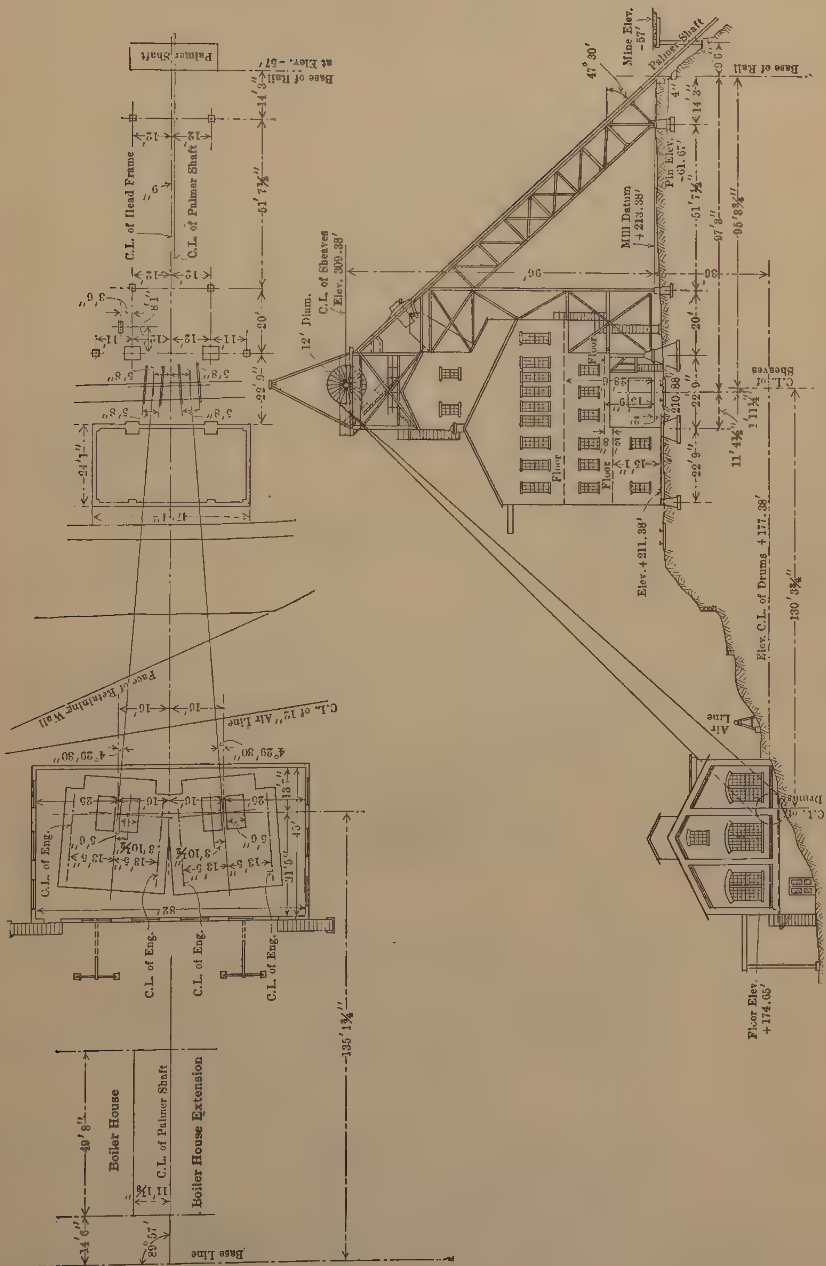


FIG. 24.—PALMER SHAFT; LOCATION OF HEADFRAME AND ENGINE HOUSE.

man cars and is the only one who can ring to the engineer. To do this he puts a key in the signal box, and by turning the key throws a lever which makes a connection for the engineer's bell. When the cageman is ringing to the engineer the bells ring on every station, and the engineer's

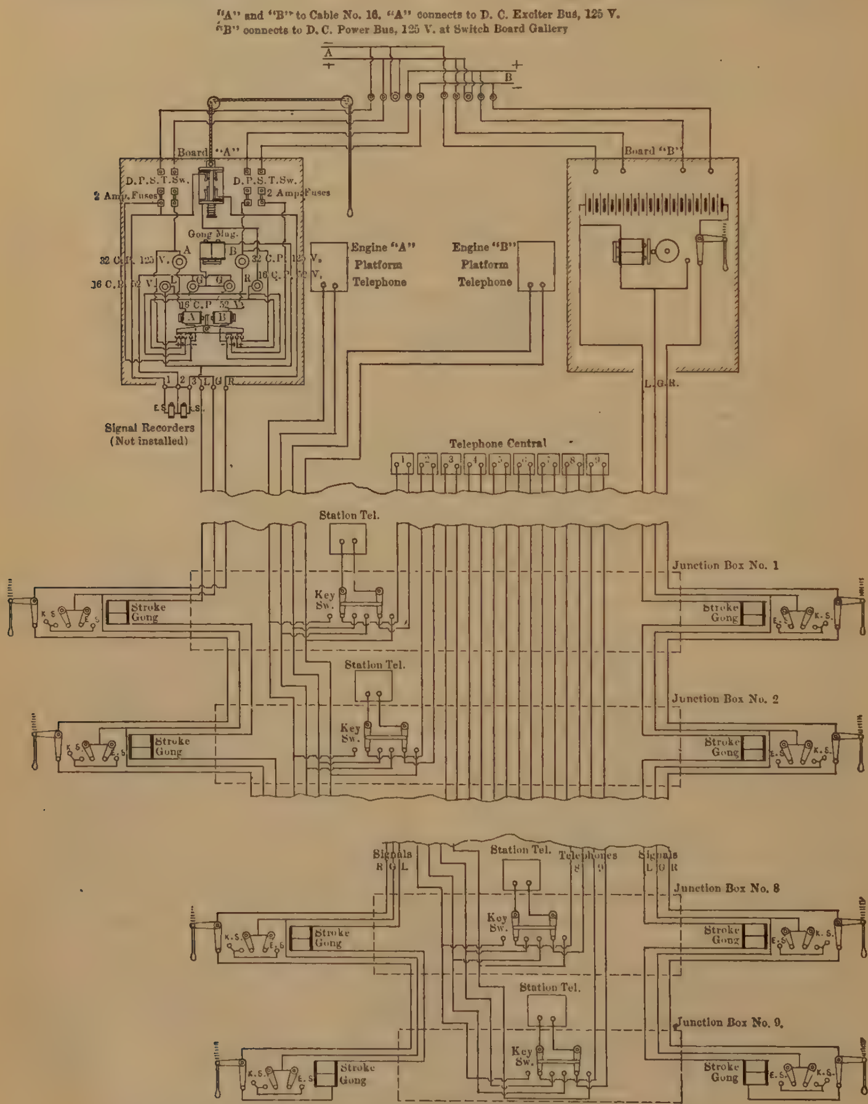


FIG. 25.—WIRING DIAGRAM OF TELEPHONES AND SIGNALS.

answering signal (the engineer answers with the same signal he gets) is rung on every station. If the cageman has his key in the box and turned and some one rings at another station, the bell does not get to the engineer, though it rings in the signal box at which the cageman is standing. This

SPECIFICATION

Bolts and Nuts to be of Brass, and to have U.S. Standard Threads,
Washers to be of Copper.
All Cast and Wrot Iron, both Inside and Outside of Box to have a thorough
Coat of Acid Fume and Moisture Proof Paint (Black)

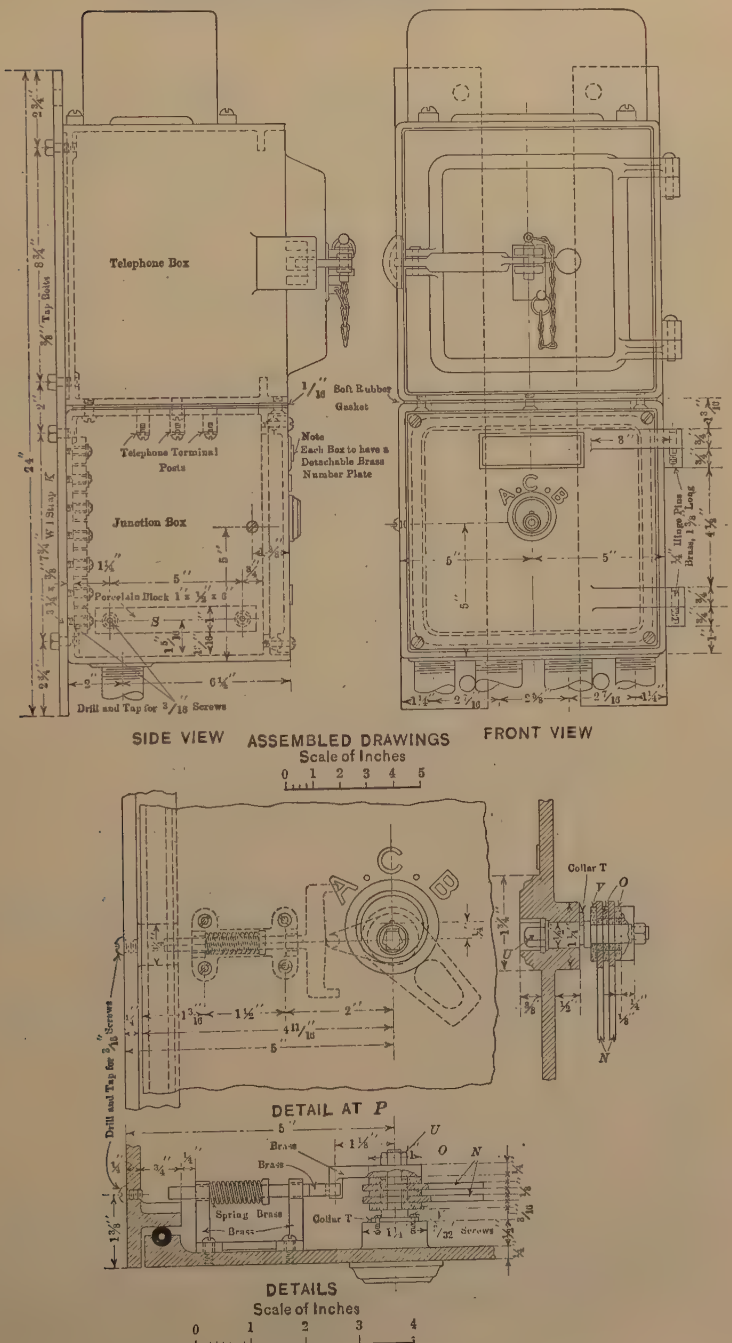


FIG. 26.—TELEPHONE AND JUNCTION BOXES.

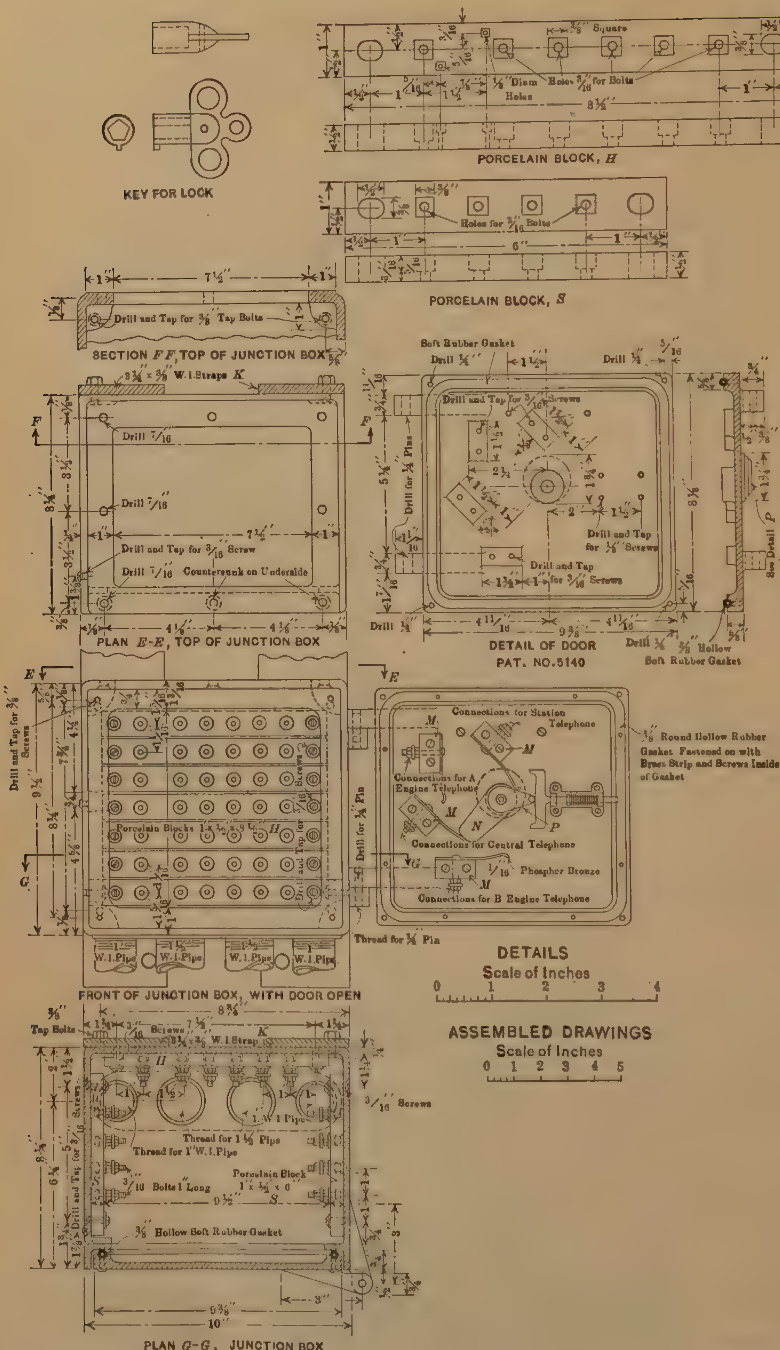


FIG. 26.—CONTINUED.

arrangement is also on the ore side, the ore loader having a key with which to ring to his engineer. The ore loader's key will not fit in the man-side signal boxes, so he cannot move the man cars.

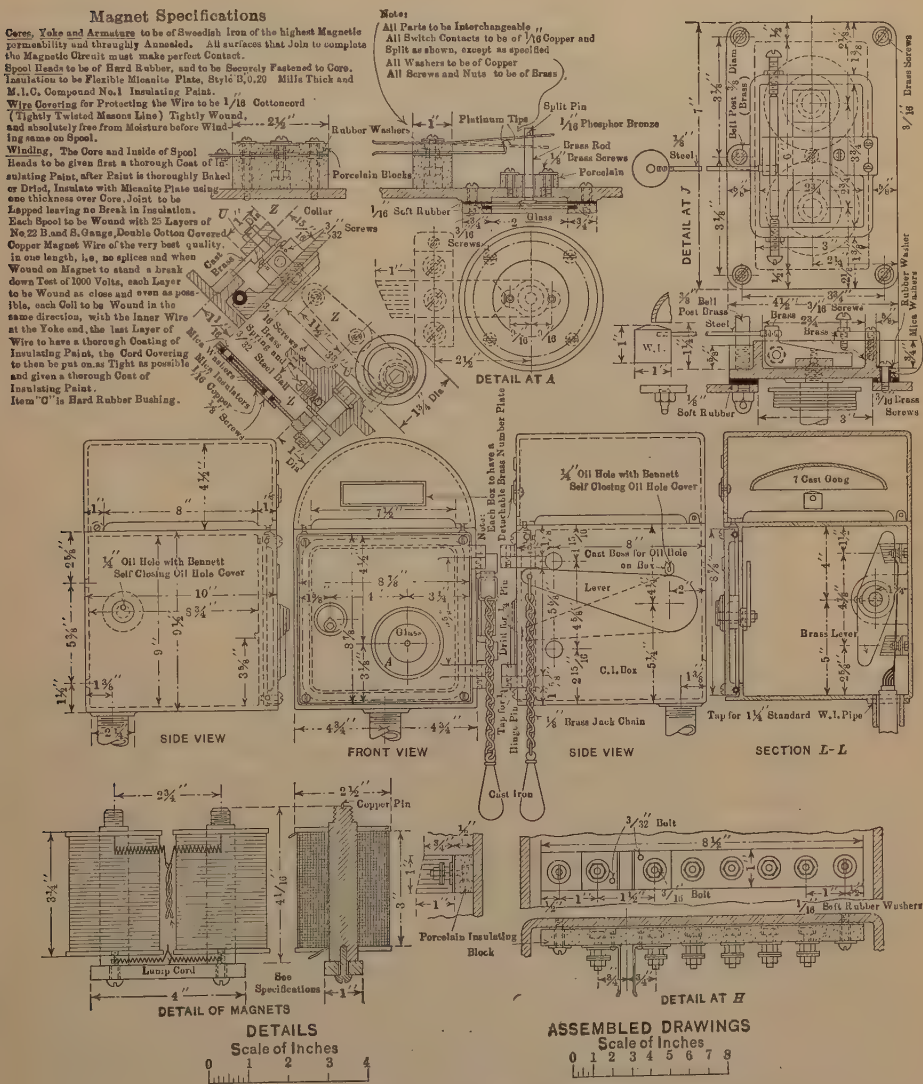


FIG. 27.—SIGNAL BOXES.

Telephones are on each station in the shaft, and also at the engineer's platform. Ordinarily these phones ring to the central office, but by throwing an internal switch with a key, the cageman and the ore loader can connect directly with either engineer according as the key is turned

to the left or right, without the coöperation of the telephone central. Figs. 25, 26 and 27 indicate the wiring diagram and instruments which accomplish these results.

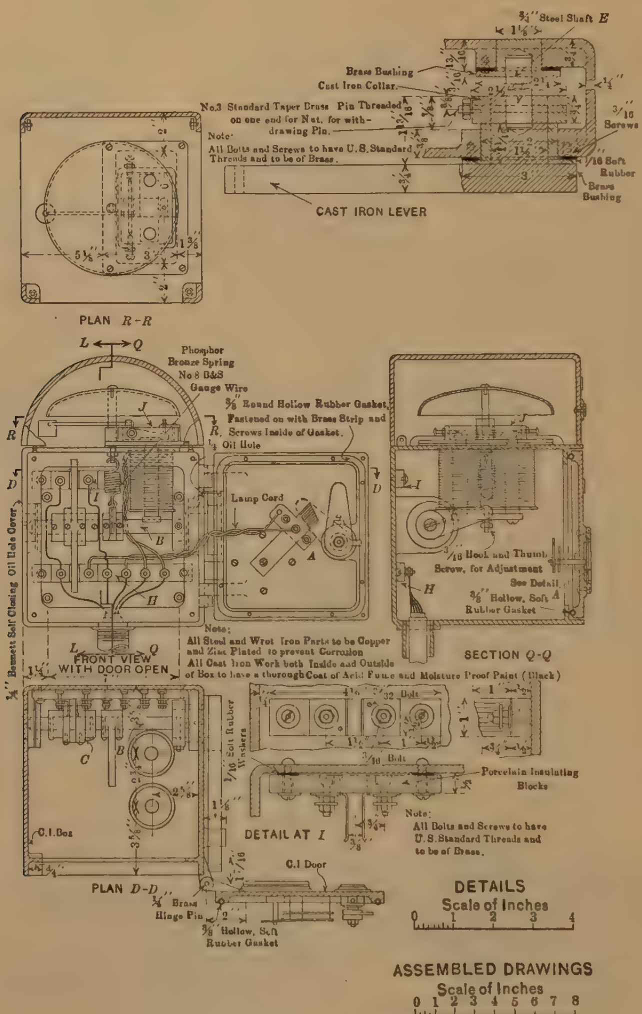


FIG. 27.—CONTINUED.

Pumps

The pumping system for the shaft consists of two units, which handle all the water from the mine, except that which comes from the large open cut at the south end. This open-cut water is handled by two 4-in. two-stage turbine pumps each direct-connected with a 35-hp., 1800-r.p.m., three-phase, 440-volt motor, and each will deliver 300 gal. (1135 l.)

per minute against a total head of 225 ft. (68.5 m.) at 51 per cent. efficiency.

The main pumps are on the 1050 level, near the shaft. They are three sets of 5-in. Worthington eight-stage turbine pumps, with thrust balanced by having four stages on each side of the direct-connected Allis Chalmers three-phase, 60-cycle, 440-volt, Bullock-type motors driven at a speed of 1800 r.p.m. These motors are rated at 250 hp. or 285 amp. per terminal. Each eight-stage pump can be run separately, or all three can be run at once; one alone can pump 450 gal. (1703 l.) per minute, at a 57 per cent. efficiency of pump and column pipe and lifts a total head of 1000 ft. The discharged water goes through a 10-in. pipe to a tank at the surface which provides storage for the operation of the hydraulic rams of the loading pockets.

At the bottom of the shaft are two smaller electrically driven 3-in. two-stage turbine pumps direct-connected to 25-hp., 1800-r.p.m., 440-volt, three-phase motors which are water-jacketed and sealed against flooding, and with the starting apparatus located in the 1050 pump station so that the pumps could be run under water if necessary. They will lift 200 gal. (757 l.) per minute against a total head of 150 ft. (45.7 m.) at a 50 per cent. efficiency. Compressed air is admitted to the inside of the motor housing and prevents leakage of water into the winding should a gasket prove defective, when the motor is under water.

Ore Channels or Passes

Series of chutes lead to the storage pockets above the loading stations. These are fed by some of the small cars, and by the electric haulage systems. Chute series are also established at intervals from the shaft into which the hand-trammed cars are dumped so that the hand-tramming distance will not be too great. The ore from these chutes is drawn out by the electric haulage service, and transferred to the series which leads to one of the loading pockets.

Surface

Besides the rough crusher at the top of the shaft, there is a large two-wing change house for the workers, each wing approximately 48 by 123 ft. (14.6 by 37.5 m.), a 32 by 54-ft. (9.7 by 16.5-m.) shop for steel sharpening, blacksmith work, and rock-drill repairs and a 20 by 41-ft. (6 by 12.5-m.) building used partly for storage of material and partly for a laundry. The mine office formerly was in the change house, but is now in a separate 24 by 44-ft. (7.3 by 13.5-m.) brick building. The old office has been changed into a change room for the shift bosses.

The original change house[1] had 530 steel lockers, with numerous

[1] Described in the *Engineering & Mining Journal* (Aug. 24, 1912), **94**, 358.

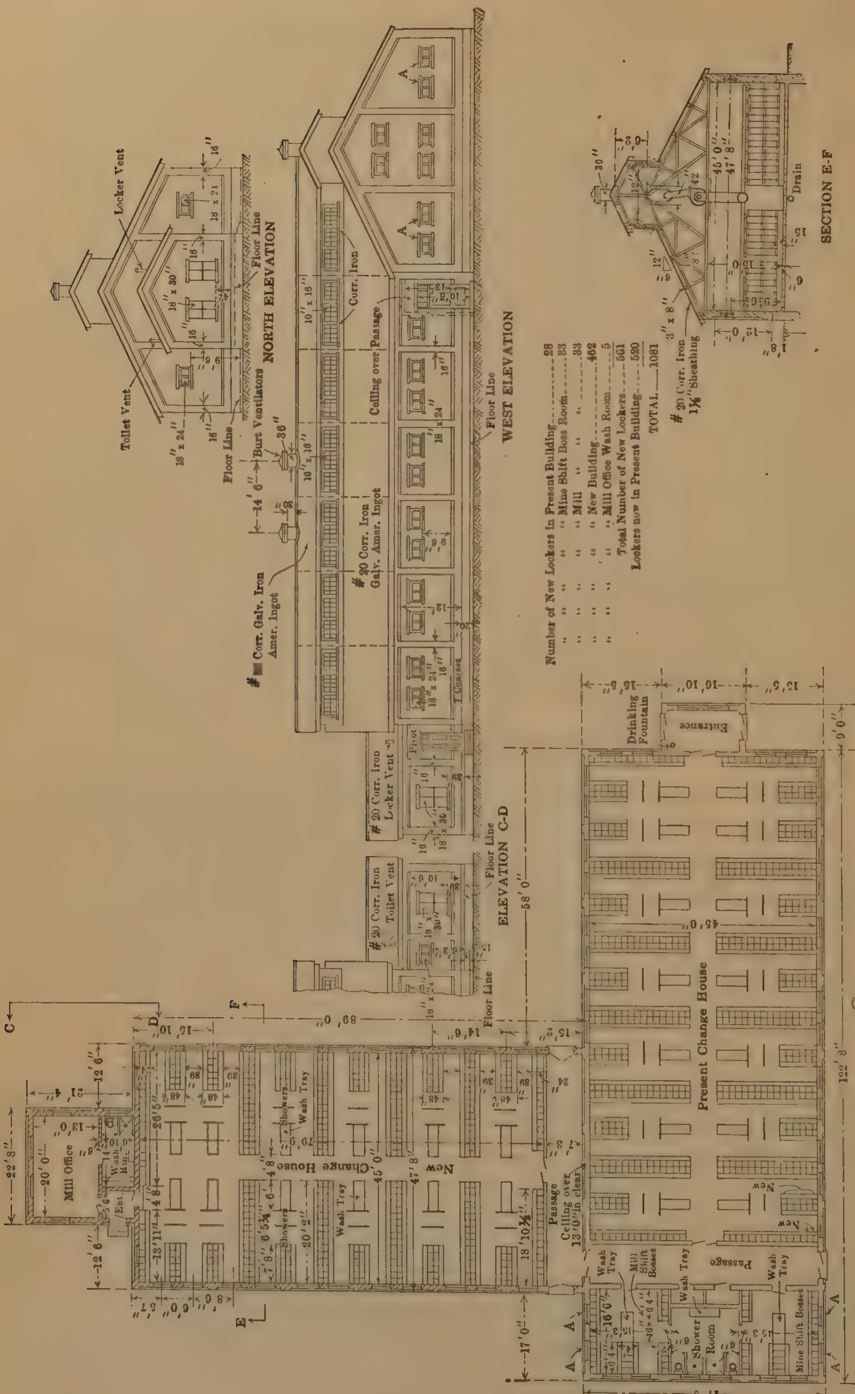


FIG. 28.—FRANKLIN CHANGE HOUSE.

showers, and wash stands, each locker heated with steam, and ventilated by suction pipes from the top. Consequently there is rarely any odor in the building from sweaty clothes, etc. Lockers, towels, and soap are provided free of charge to the men. Recently a duplicate of the original change house has been built so there are now accommodations for 1081 workmen (see Figs. 28 and 29). The toilets for the men are provided in independent and detached buildings. The allowance of 6⅓ lockers to each water outlet or faucet, and 20 lockers to each shower bath, has worked out satisfactorily in this installation.

The blacksmith shop contains one Leyner No. 5 drill sharpener and one Sullivan sharpener, an oil forge for heating the steels sharpened by both machines, also two coal forges, anvils, and tools for the general blacksmith work. A drill press and an emery wheel are also part of the equipment. A 10 by 12-ft. (3 by 3.6-m.) room is partitioned off in one corner for the drill repairmen.

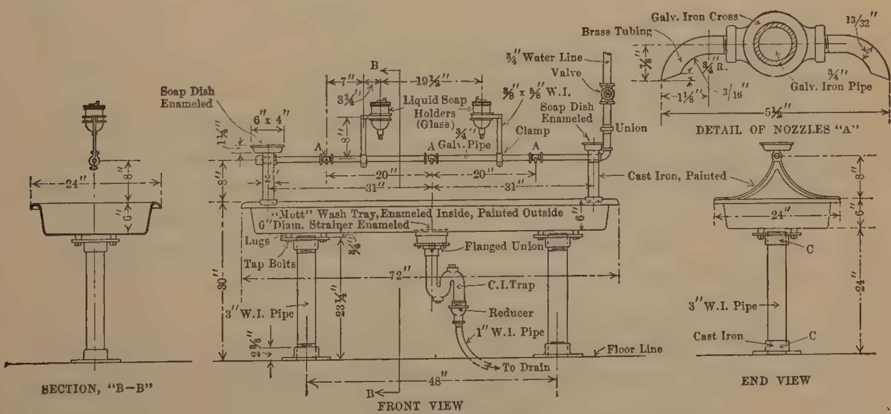


FIG. 29.—DETAILS OF WASH TRAYS IN CHANGE HOUSE.

Safety

The rope inspection each day of both man and ore hoist ropes is one of the safety precautions taken. All stations in the shaft are protected by guard rails or chains, and are electrically lighted as well. Every night the shaft is inspected by a crew of men who examine the tracks, tighten up the bolts, etc., grease the sheaves, and replace those with worn linings, clean up the muck that has accumulated, and in general keep the shaft in the best condition possible. The rolling equipment is greased and looked over each day.

In addition to the overwinding devices on the hoisting engines, the construction of the headframe above the tipple still adds to the factor of safety of operation. If by failure of the other various devices the skip should be pulled above the tipple, it would ultimately strike *I*-beams

which should keep it from being pulled over the top of the headframe and then if the rope should break, the rear wheels of skip would drop into a special rail pocket, so the skip would not coast uncontrolled back down the shaft.

Special Headframes for Prospect Shafts

Under conditions where operations must be conducted on a small scale it becomes advisable to sink either incline or vertical shafts by buckets. The schemes shown in Figs. 30 and 31 have proved very satisfactory both in the reduction of labor for operation (as only one man, the hoisting engineer, is needed on the surface to attend to the hoisting, loading of car, and disposal of material), and in the safety features protecting the workman at the bottom of the shaft from spill when the buckets are dumped at the shaft collar.

Development

For transportation, etc., all the old drifts and other openings were used if they coördinated with the mining plans, but many others had to be made, and at the present time a number of drifts are being advanced. For ordinary tramming, with hand cars, drifts 7 ft. (2.14 m.) high and 6 ft. (1.83 m.) wide are used, but when electric haulage is to be used, the drifts are larger, 8 by 10 ft. (2.44 by 3 m.) to 10 by 10 ft. (3 by 3 m.) in section. This provides ample clearance for cars, and height for trolley wire. In the past these larger drifts have been driven with two machines working in a heading, but at the present time a small drift 6 by 7 is driven first, and subsequently stripped to the desired size.

For hard ground, hammer drills of the water Leyner type such as the Sullivan D.R. 6, the Ingersoll-Rand 18A, and the Waugh Dreadnaught, are used. These are mounted on shells. A special type of column clamp is in use, by means of which an arm made of 2½-in. (63.5-mm.) extra heavy pipe is used, with a column made of 4-in. (101.6-mm.) extra heavy pipe. In addition, the length of this arm on either side of the column can be changed at will, for the clamp is so constructed that the pipe arm is set at a tangent to the column, instead of on a radial line. This pipe fits in the clamp, the same way the column does, and can be moved in the clamp, just as the clamp can be moved up or down on the column. Fig. 32 shows the construction of this clamp, and the photograph, Fig. 33 shows the mounting. The same kind of clamp is used with the lighter machines, but both the column and arm are the same size, 2½-in. extra heavy pipe of 2⅞ in. outside diameter. However, the 4-in. column is preferable when double jack screws are used, as a man may readily distort the lighter columns by failing to equalize the force exerted by each of the jack screws.

For the softer or easier drilling ground, and for opening out the first

cut in the top slices, a block-hole machine, Sullivan D. P. 33 type, is used mounted on the pneumatic feed, type D. 62.

With both types of drifting machines, water is used, through hollow drill steel, to clean out the holes. The larger drills use 1¼-in. (31.75-mm.) round hollow steel, with cross bits, while for the smaller machines ⅞-in.

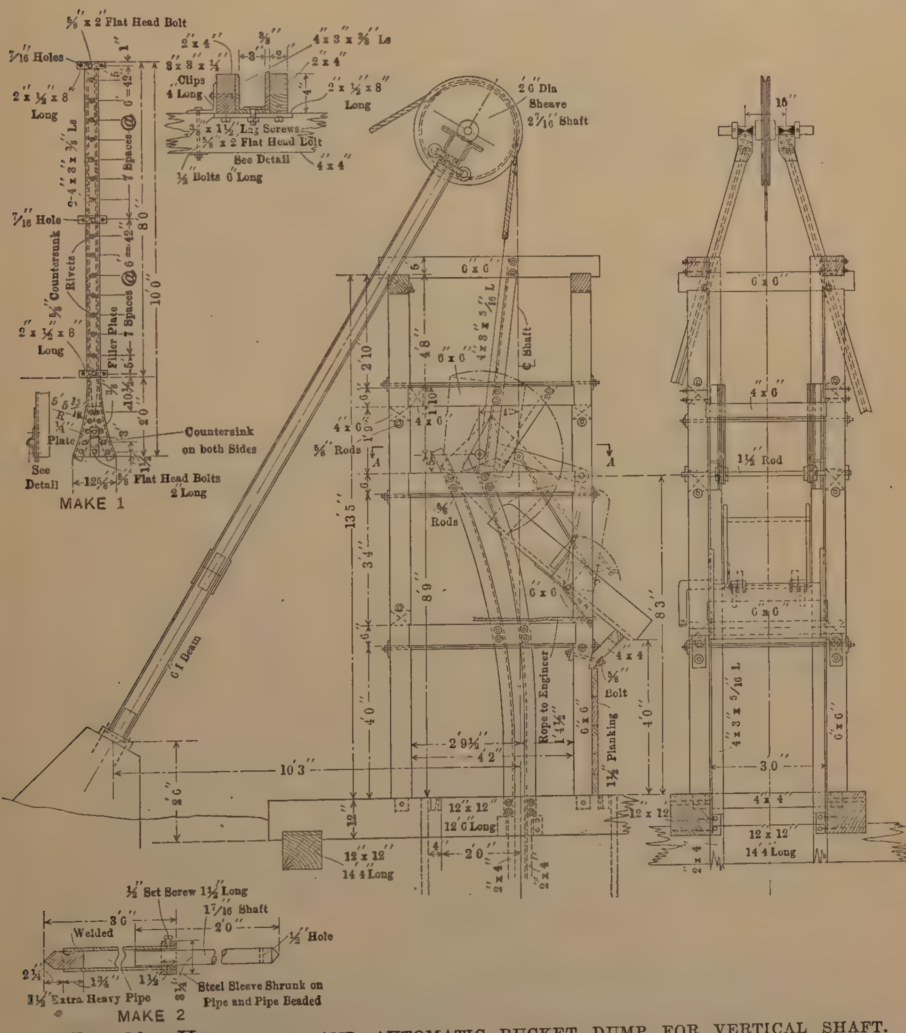


FIG. 30.—HEADFRAME AND AUTOMATIC BUCKET DUMP FOR VERTICAL SHAFT.

(22.225-mm.) hollow drill steel, either quarter octagon or hexagonal in section, are used, with rose bits. Types of bits are shown in Fig. 34.

The ground at Franklin, both rock and ore, is peculiar in that while it drills well it is hard to break. Many known types of rounds have been tried, but the most successful, and therefore the one used for drifting,

consists of a burned cut, of horizontal holes with a large number of parallel holes breaking to the cut; the number of holes for a 6 by 7 drift varies from 19 to 25; the crosscuts going from foot to hanging wall can generally be broken with fewer holes than the drifts paralleling the foot or hanging walls. Arrangements of holes for a round of 20 holes and for a round of 25 holes are shown in Figs. 35 and 36.

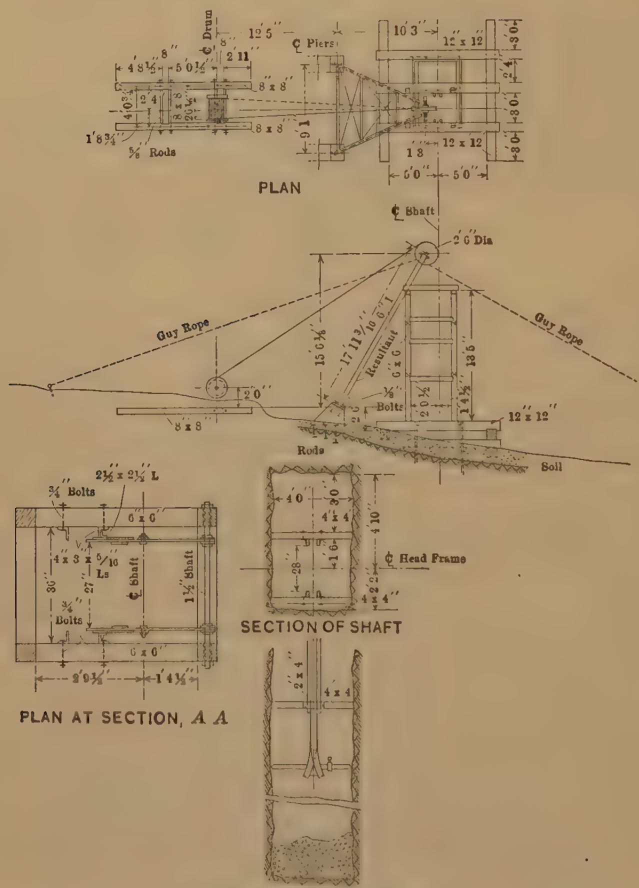


FIG. 30.—CONTINUED.

Eleven hours of work is considered the required time for drilling and firing a round, so the men are paid for that amount of time if the work is done in 1 day and an advance of 3 ft. or more is made. Many times the drill runners complete the round in less time and therefore make a bonus by the difference.

It is necessary to fire a drift heading in stages; first, the "cut hole" either hole No. 1 or holes No. 2 and No. 3 (see Figs. 35 and 36) are loaded heavily with explosive and fired. If the cut does not break to depth, it

has to be fired again, which can generally be accomplished by firing holes No. 4 and No. 5, but in case these are badly shattered, the broken rock has to be dug out, so that the bottoms of the holes may be fired. Holes 6, 7, 8 and 9, composing what is locally termed the "cut side round," are loaded and fired, and make an opening large enough so that the remaining holes will break well. The other holes are fired in either one or

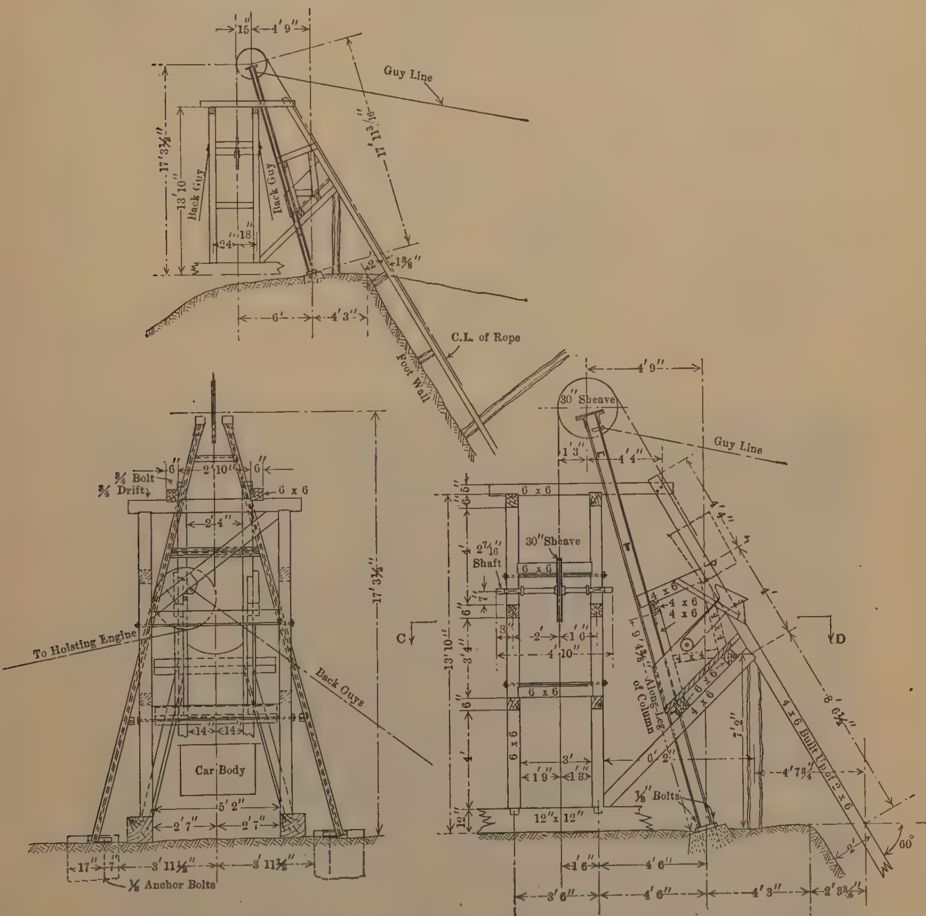


FIG. 31.—HEADFRAME AND AUTOMATIC BUCKET DUMP FOR SLOPE SHAFT.

two rounds, generally one. Gelatine powder of 50 per cent. strength (1 in. diameter by 8 in. long) is used for the "cut" and the "cut side round," the rest of the holes are fired with a 30 per cent. ammonia powder; 150 to 200 sticks in all are necessary to break a heading.

With a few exceptions, mainly tramming, all the mine work at Franklin is done on the day shift. This means either that a special crew has to be sent out at night to clean out the heading, or that the mucking has

to be done while the machine is working. Sometimes two headings are driven with the one machine, one heading on one day and the other on the next, then both drill runners and muckers have their headings to

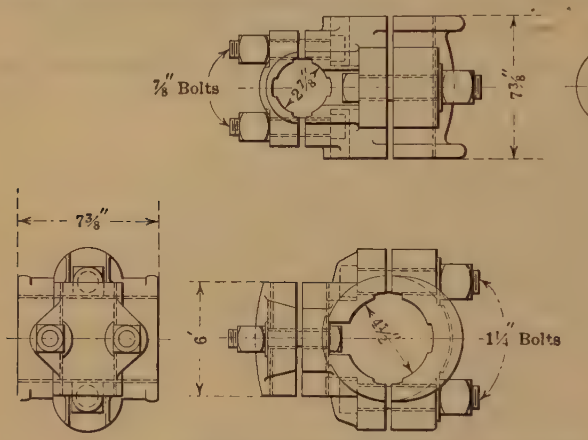
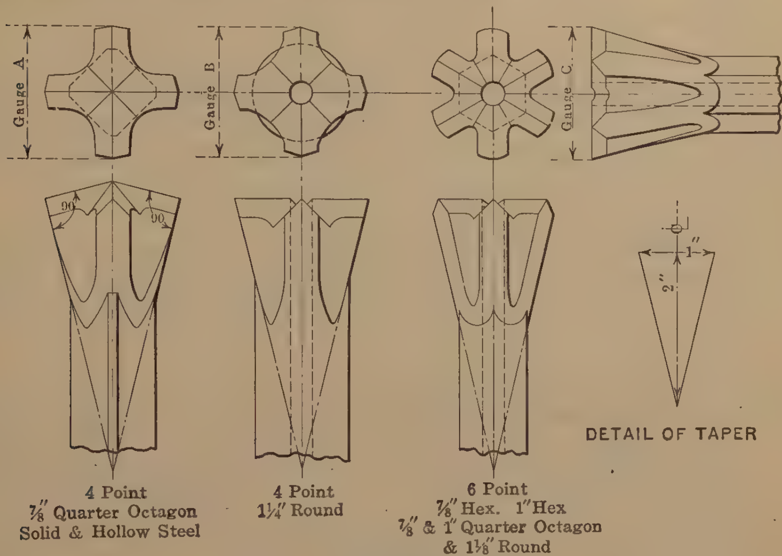


FIG. 32.—SPECIAL COLUMN ARM CLAMP.

FIG. 33.—COLUMN MOUNTING.

themselves. In general, however, the column is rigged across the drift as a bar, and while the runners are drilling the holes in the upper part of the face, the muckers are cleaning up the broken muck. Three muckers

are generally needed to do this, as one has to get in front of the bar to throw the muck back. In certain cases, when the tram is long, two cars



Gauge	A	2" to 1⅜" by differences of ⅛"
Gauge	B	2¼" to 1⅝" " " " "
Gauge	C	2⅜" to 1¼" " " " "

FIG. 34.—TYPES OF DRILL BITS.

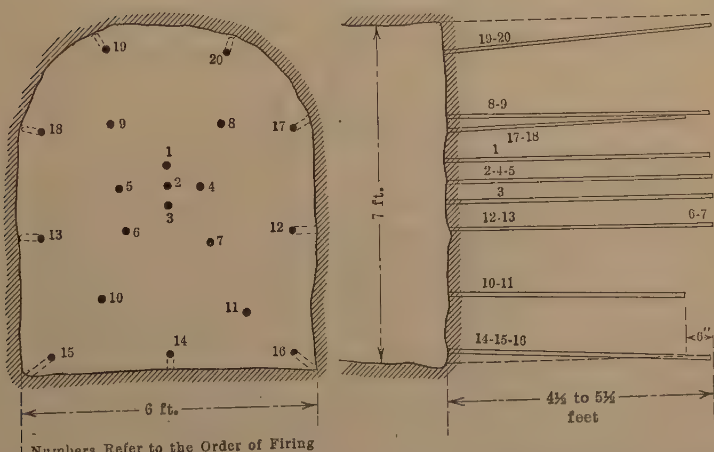
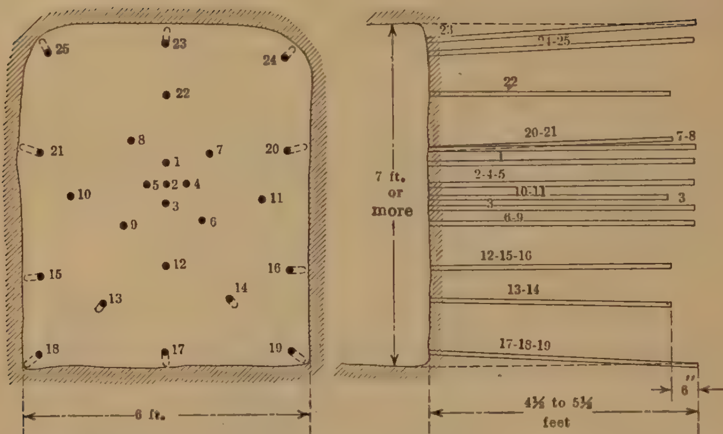


FIG. 35.—ARRANGEMENT OF HOLES FOR DRIFTING—ROUND OF 20 HOLES.

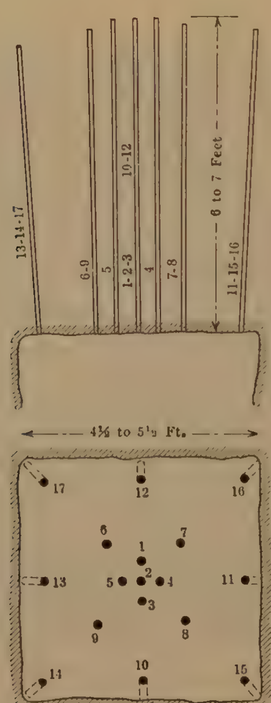
and four men are needed to get the muck out so that the lower holes may be started between the fifth and sixth hour of the shift.

The small machines are considered "one-man" machines, hence, after the set up is made, when a helper is used, the drill runner works



Numbers Refer to the Order of Firing

FIG. 36.—ARRANGEMENT OF HOLES FOR DRIFTING—ROUND OF 25 HOLES.



Numbers Refer to Order of Firing

FIG. 37.—ARRANGEMENT OF HOLES FOR RAISING—ROUND OF 17 HOLES.

alone. The men mucking in the heading or in a nearby heading take care of the water supply, etc., and give a hand occasionally, and a nipper supplies sharp drills. The large machines are too much for one man to run steadily, day after day, especially if 25 or 26 holes are required, and a helper is generally employed. Two men always do the firing, and if many approaches have to be guarded more men are used. The runners get 43½ c. per hour at present, trammers 39 c. per hour and helpers 37½ c. per hour. The extra mucker gets the same rate as the helper.

The men are expected to advance at least 3 ft. per round for the 11 hr. time. In general this is exceeded. The amounts of labor and supplies are reported daily by the shift boss, and recorded at the mine office. Table 3 shows the best drifting record, and also the average figures for all drifting done during 1916.

TABLE 3

Drill Shifts per Foot of Advance	Shifts of Breaking Labor per Foot of Advance	Tramming and Mucking Shifts per Foot of Advance	Explosive Cost per Foot of Advance	Average Size	Feet Advanced	Number of Rounds	Advance per Round	Number of Holes Drilled per Round	Feet Drilled per Round	Tons Broken	
0.16	0.21	....	1.90	5½ by 6½	122	25	4.9	18.4	92.5	570	Best
0.26	0.78	1.09	1.94	6 by 6½	6,011	1,537	3.9	22.0	104.0	34,590	Average, 1916

These figures do not include any service charges—such as repairs to machine, steel and sharpening charges, and miscellaneous supplies—but they give a good indication of the work accomplished in the direct work of drifting.

Raises

A large number of raises is necessary at Franklin, for the system of mining requires raises for each stope, for ventilation and for access, as well as raises for ore or rock passes, timber slides, and ladder roads. A standard size 4½ by 4½ (1.37 by 1.37 m.) to 5 by 5 ft. (1.53 by 1.53 m.) is used for all raises; should a larger opening be required, the desired size is obtained by stripping.

For the raises, a round is used which is somewhat similar to the drift rounds but consists of 17 holes (see Fig. 37). The holes are drilled deeper than the drift holes, for at least 5 ft. (1.53 m.) per round is required for a bonus payment. The raise is fired in three stages; first the "cut," then the "cut side round," and finally the other holes. The center ones on each side go before the corner ones.

Stoping machines, weighing 85 to 90 lb., are employed for drilling, using ⅞-in. (22.225-mm.) quarter octagon solid drill steel with raised center cross bits. The starting gage is 1¾ in. (44.45 mm.) and this gage

drops ⅛ in. (3.175 mm.) with each steel. The machines are equipped with the direct type of air feed and an adjustable extension leg. The use of this extension leg enables odd lengths of drill steels to be used and permits a greater leeway in the placing of staging poles without using insecure blocking beneath the drill-feed pointer.

Two poles and a couple of short pieces of plank are used to make a working stage as shown by Fig. 38. The poles are wedged across the raise the required distance below the working breast; the planks lie on

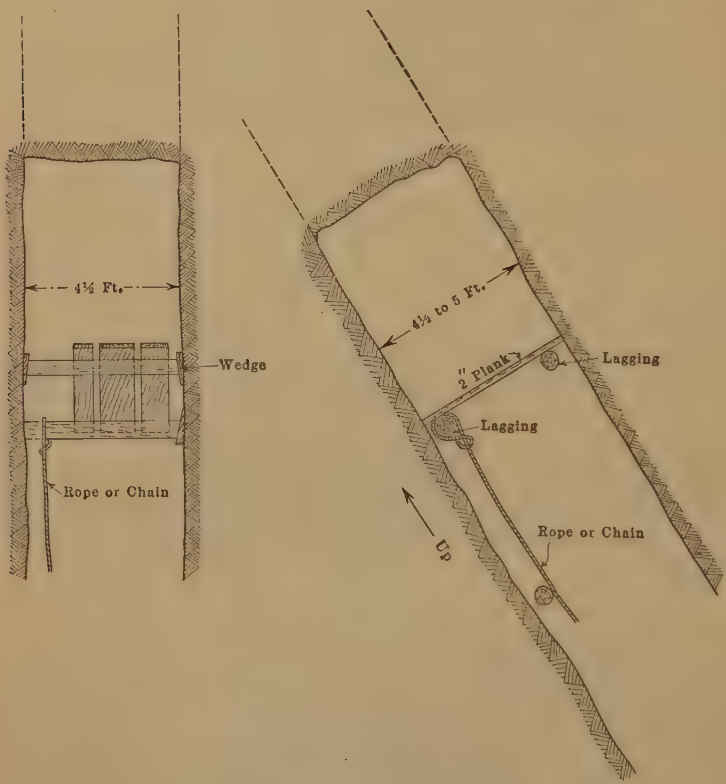


FIG. 38.—STAGE IN A RAISE.

these poles. The stage is not level, but is placed approximately at right angles to the direction of the raise. When the round is fired, the planks are removed, but the poles are left in place to become of assistance in climbing the raise. A chain is generally fastened at the top pole or to a pin anchored in the rock, to facilitate movements of the workmen up and down the raise. With a very long raise a ladder made of ⅝-in. (15.875-mm.) iron rounds clinched around ⅝-in. wire rope has been found of great service. The flexibility of this sort of ladder allows it to stand very hard usage and when not in use it can be rolled up.

Two men work in a raise, a runner and a helper; they share in the bonus plan the same as the men in the drifts. Operating figures showing results in some of the best raises, and also the average figures for 1916 are shown in Table 4.

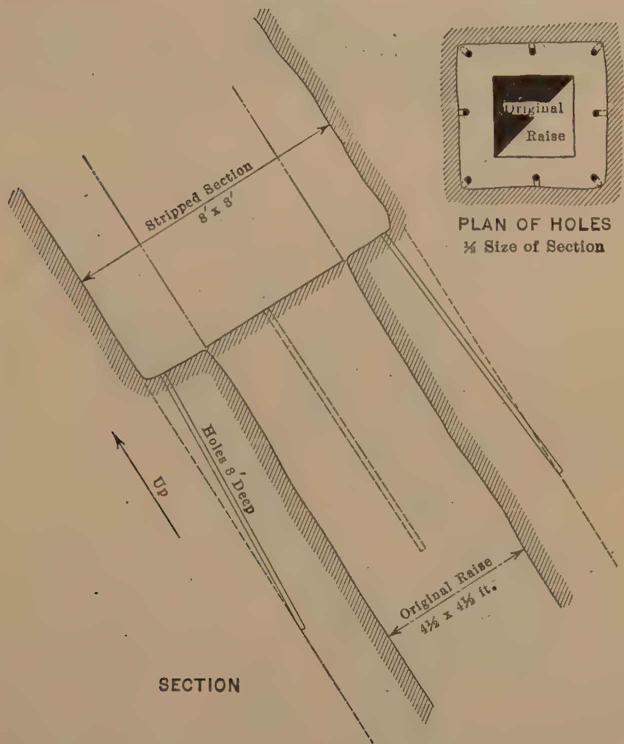


FIG. 39.—STRIPPING RAISE FROM TOP—ARRANGEMENT OF HOLES.

TABLE 4

Drill Shifts per Foot of Advance	Shifts of Breaking Labor per Foot of Advance	Explosive Cost per Foot of Advance	Average Size of Raise	Feet Advanced	Number of Rounds	Advance per Round	Number of Holes Drilled per Round	Number of Feet Drilled per Round	Tons Broken	Explosive Cost per Ton Broken	
0.12	0.44	$0.68	5 by 5	95	16	6.00	17	102	310	$0.21	
0.11	0.43	0.77	4½ by 5	49	8	6.13	17	135	160	0.22	
0.25	0.54	0.98	5 by 5	115	21	5.50	17	102	380	0.30	
0.16	0.53	0.83	4½ by 5	78	15	5.20	17	98	250	0.26	
0.20	0.86	$1.37	4½ by 4½	6,118	1,356	4.50	17	100	16,675	$0.46	Average 1916

Two methods are used for stripping the raises. When possible to work from the top down, benching is employed, using block-hole or sinking machines for the drilling. The men have safety ropes attached

to their belts, and the opening is guarded with plank or lagging. Ordinarily two men with two machines can fire two rounds per shift.

When benching is not practicable, the stoping machines are used, working up from the bottom, by drilling into the sides with over-lapping holes. The drawback to this is the necessity for rebuilding large stages after each firing. Several rounds are generally drilled and all fired at one time to avoid this as much as possible. The methods described above are shown in Figs. 39 and 40.

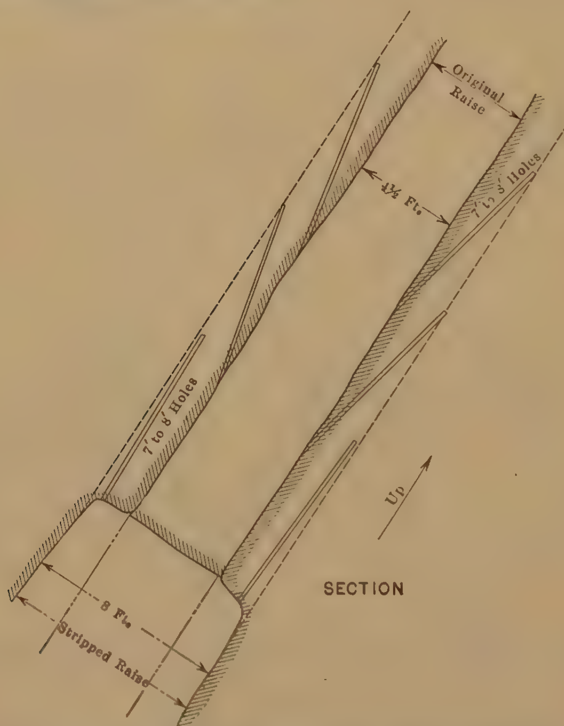


FIG. 40.—STRIPPING RAISE FROM BOTTOM—ARRANGEMENT OF HOLES.

Timber Shafts

One of the old shafts is used for lowering timber to the southern part of the mine. The track is the same that was once used for hoisting ore, but as it has a vertical curve in it, the timber truck is made up of two trucks, connected with wire ropes, of which the lower one has a horizontal platform to hold the timber, but the upper has only the bottom on which to rest it. A hoisting engine on the surface, run by compressed air, controls the movement of the truck. In the same engine house is a small air hoist, with which the timber is loaded into the truck, a snatch block on the headframe enabling the engineer to lift the sticks clear of

the ground so that they may be swung into position by the helpers and then lowered to the truck. The use of this shaft lightens the timber service in the main shaft, and saves long transfers underground, for the timber is delivered by teams and railroad. The shaft extends only to the 400-ft. level.

Below this, and in other parts of the mine, raises are used for the distribution of timber from level to level. These have been stripped to a good size, at least 8 by 8 ft. (2.5 by 2.5 m.) and are planked on the bottom

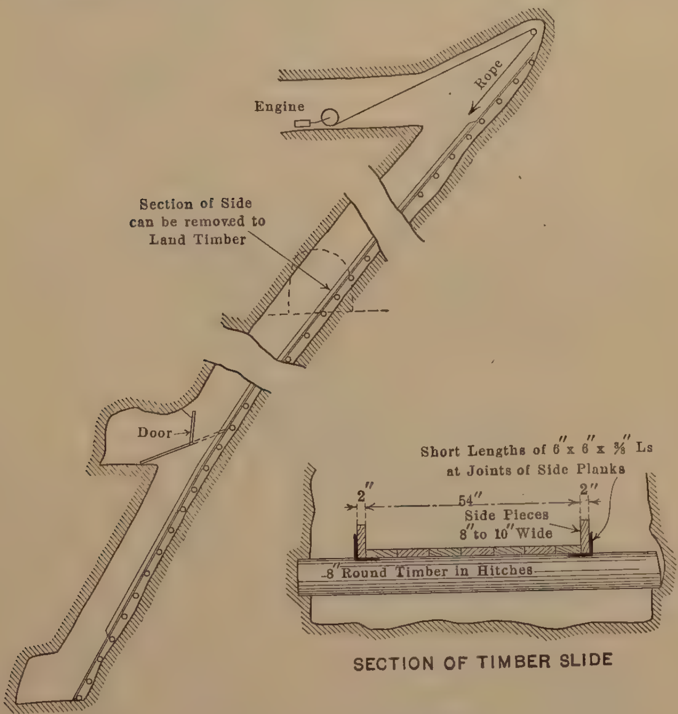


FIG. 41.—TIMBER SLIDE.

and in some cases on the sides. Small Lidgerwood hoists operated by compressed air are used in this service. These are located at the top out of the way, and are controlled by bell signals or by a whistle signal (the whistle being operated by compressed air). These timber slides work very well and besides being used for timber are used for shifting cars, machines, and various other bulky objects (see Fig. 41).

Ore Chutes

For ore chutes, some of the old shafts are used, but several raise series have also been put up. The raise series are much better, for they have been laid out for the purpose, driven on grade and sized according to the

work they are used for. These chutes end either in a shaft loading pocket or at a level on which there is electric haulage, so that the ore may be transferred to the loading-pocket chutes most economically. In this way, except for a few cases, the hand-tramming distances are kept down, generally below 400 ft. (121 m.) and a congestion of cars in a level is avoided. Figs. 42, 43 and 44 show various types of chutes and dumps at the ore passes.

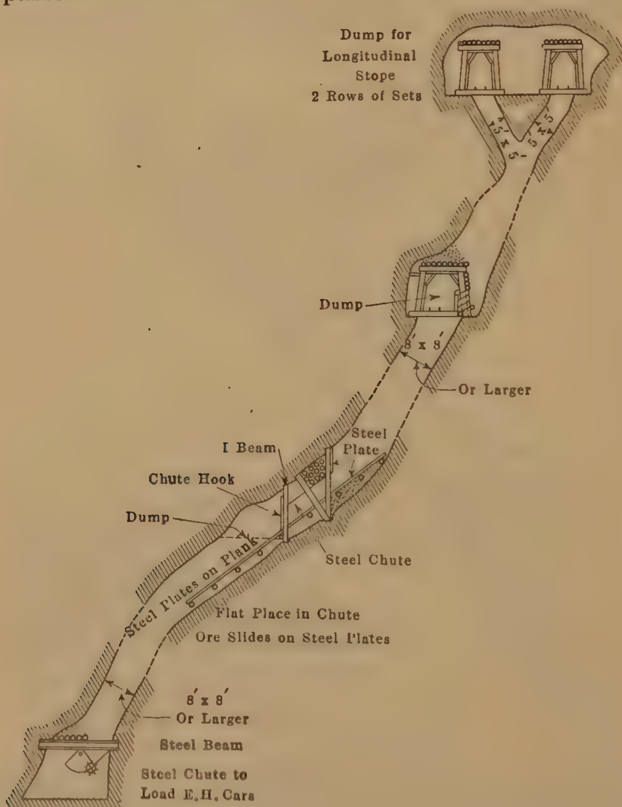


Fig. 42.—Types of ore passes and chutes.

General Mining Operations

Timbering

The timber used in the mine is chestnut and oak, and is separated, according to the diameter of the small end of the stick, into lagging 4 in. (101.6 mm.) and over, 8 in. (203.2 mm.), 10 in. (254 mm.), and 12 in. (304.8 mm.).

Practically all the stopes are worked by means of set timbers. For stopes in which small hand-tram cars are used the sets consist of two legs, 8 ft. (2.5 m.) long, made of 10-in. (254-mm.) timber; one cap, 4 ft.

3 in. long inside of the joggles, made of 12-in. timber; and a sill 5 ft. 4 in. inside of joggles, made of 8-in. timber. Should the sets stand on solid bottom, the sills are dispensed with. The sets are spaced alternately 4 ft. and 3 ft. centers, with chutes placed in the 4-ft. sections, and the chutes are alternated, so that adjoining chutes are not on the same side of the track. Recently these intervals between sets have been increased by about 6 in. without reducing the holding power of the

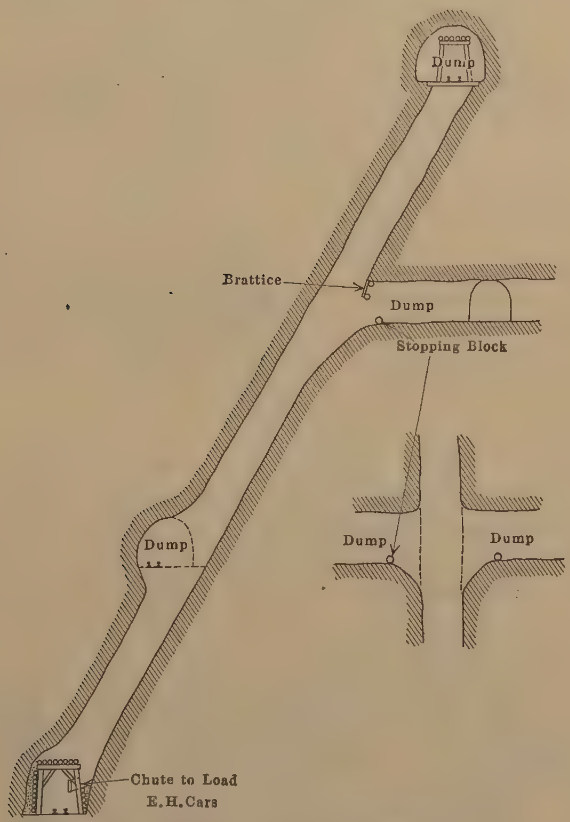


FIG. 43.—TYPES OF DUMPS AND ORE PASSES.

timber sets, thus allowing a wider chute to be used. When any member of a set fails, a new one, called a helping set, is placed alongside of the crippled one. This arrangement of chutes at 7-ft. to 8-ft. centers enables a very accurate control of the ore pile in the stope.

When electric haulage is used, a larger set is employed and slightly heavier timbers are used. The legs are 9 ft. long, the caps are 5 ft. 3 in. to 6 ft., the sills 7 ft. 6 in. to 8 ft. 3 in. These sets are spaced alternately 5 ft. and 4 ft. centers with the chutes in the 5-ft. sections.

An 8-in. timber is placed on the caps, over each leg, and spiked in

place. The space between these timbers is filled with lagging on the small sets, and with the larger sets heavier lagging is used. Care is taken that the joints of the lagging occur between different sets. On the sides of the sets split lagging is placed to hold the rock fill. As a rule the lagging is not spiked, but is held in place by the weight of the material on the top or behind the sets. Studdles which catch both cap and leg are be-

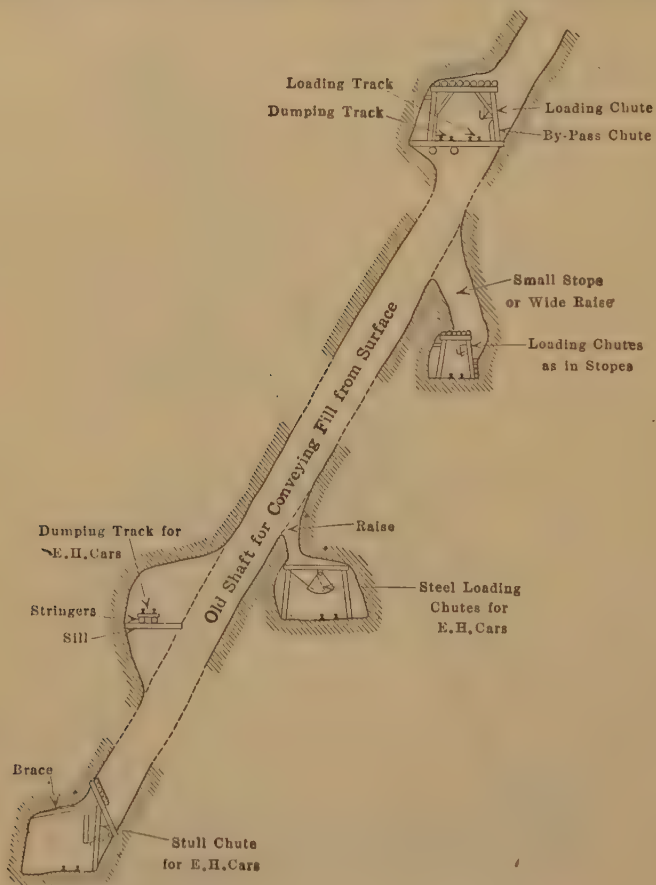


FIG. 44.—TYPES OF CHUTES FOR HANDLING FILL.

tween each set, except at the chutes. In addition, angle braces are used in each set to help support the cap and to hold the sides of the chutes, and perform excellent service in strengthening and maintaining the cap timbers. These features are shown in Figs. 45 and 46.

Several types of steel sets (see Figs. 47 and 48) are now used as experiments; they are working out quite satisfactorily though a slight permanent deflection has resulted in the caps. These sets are held together by bolts

with lock washers. Lagging, studdles, etc., are arranged as with the old timber sets. In all cases the sets are recovered from the stopes and used again, unless the opening is desired to be maintained, or failure has taken place. The increased service given by the steel sets and the

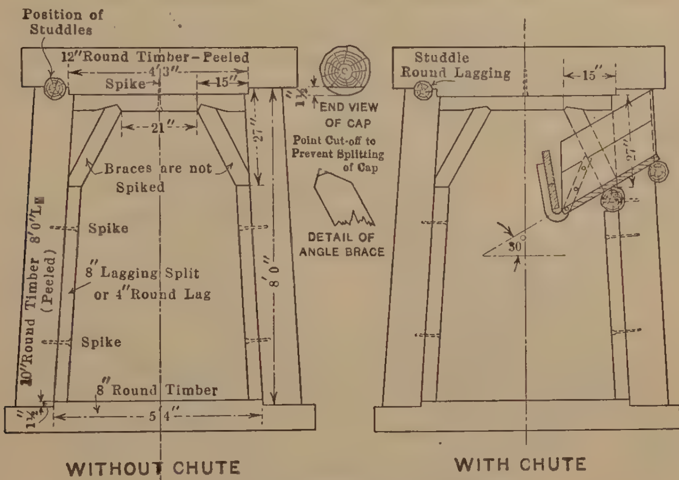


FIG. 45.—DETAILS OF STOPE SETS.

slightly cheaper cost in placing them, would make them rival the timber sets for shrinkage stopes if a greater capital investment were not involved.

A modification of the motor haulage sets, in long longitudinal stopes, on a main haulage road, has been made to give a safe passageway for men. This is obtained by the use of three legs and a long cap, and makes

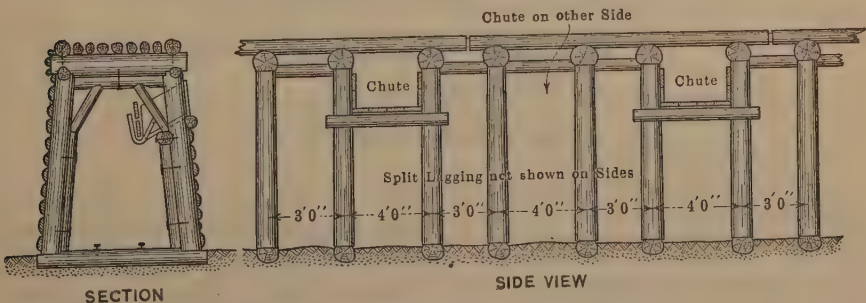


FIG. 46.—ARRANGEMENT OF SETS IN STOPES.

it possible for cars or trains to be passed without climbing over them (Fig. 49).

Whenever drifts or other openings come into a stope, provision has to be made to block the opening in order to hold back the broken ore, and later the fill. This is accomplished by building cribs of 8-in.

timber, or larger, and filling these cribs with broken ore. In case a passage has to be maintained through the opening, the cribs are built around sets. The orebody at Franklin is badly cut up by a miscellaneous collection of large drifts and crosscuts, which made a great many large cribs necessary. The greatest height to which one had to be built was about 50 ft. (15 m.) while some lengths have been as great as 75 ft. (22.9 m.). The width of the cribs depends on the height, varying from 9 to 25 ft. with 15 ft. as an average figure.

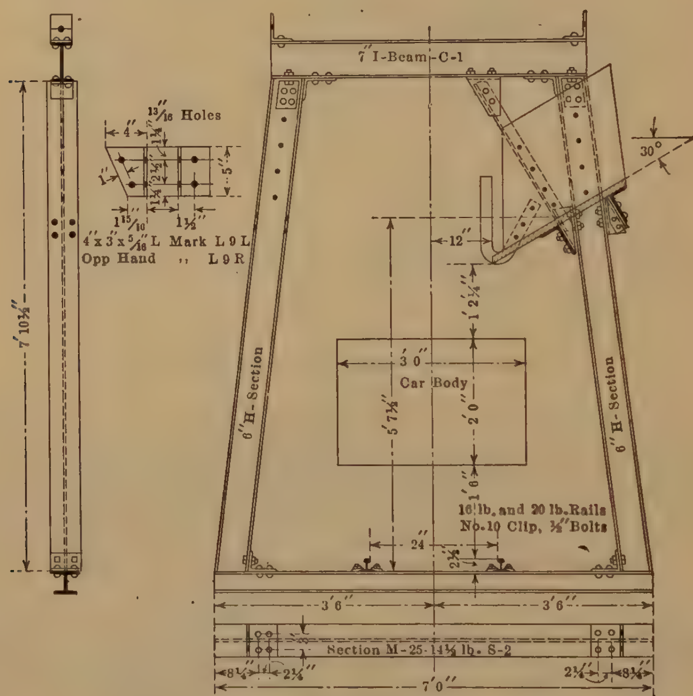


FIG. 47.—STEEL MINE TIMBERS, CLASS A2.

It has been found advisable to have but one notch in a timber at each joint, rather than to cut two in each piece or two in one piece and none in the other. It is also stronger to have the notches in the same timber alternate from the top to the bottom sides. This will bring the notches at each joint, on the same side of the timber, either top or bottom. These cribs make very substantial blocks, which when filled with broken ore will stay in place even after the timbers are fairly well rotted. The advantages of this system of notching over others can easily be demonstrated by models made of twigs or match sticks (Fig. 50).

The broken ore is generally put into the cribbings as the stopes are carried up, so as to have as much resistance to the pressure of the ore in

the stope as possible. In case this is not done, they are filled as the stopes are drained empty. Several types of chutes are used. In the stopes, the chutes used are as shown in Fig. 45; these are easily constructed and repaired, and they stand well under the service. The chute hooks are

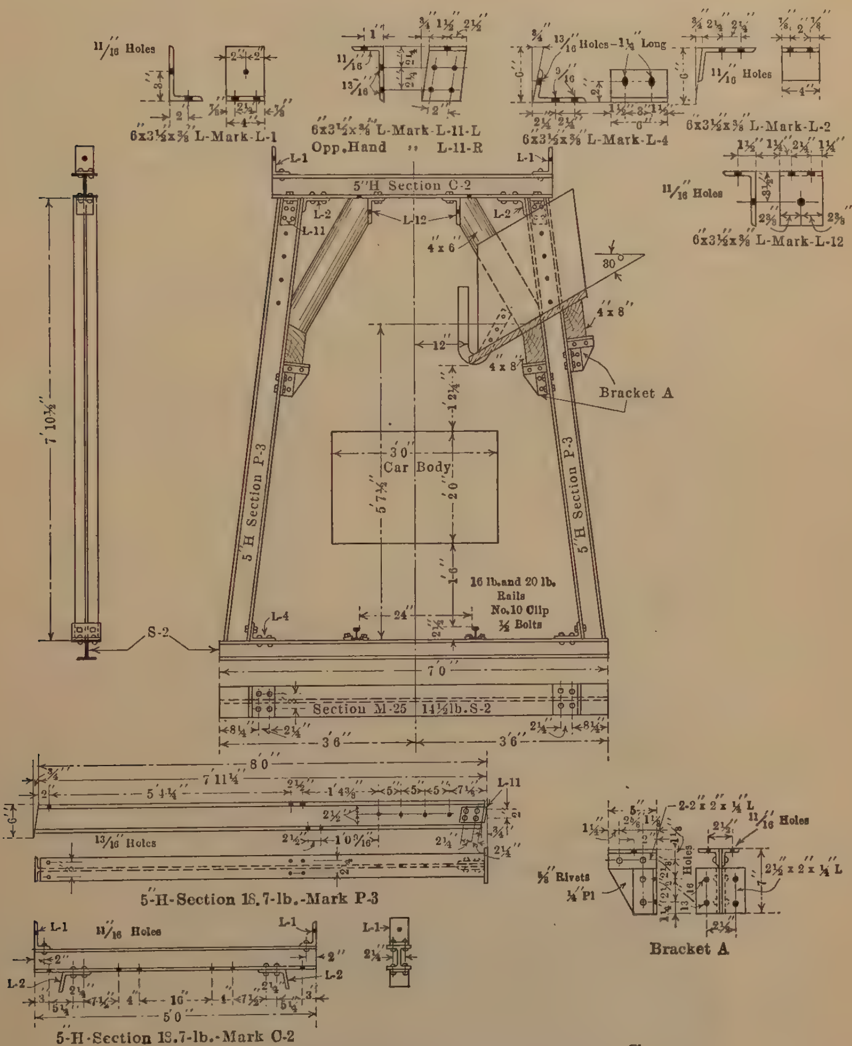


FIG. 48.—STEEL MINE TIMBERS, CLASS C.

made from 2 by ⅝ in. (50.8 by 15.875-mm.) iron, and are attached with ½-in. (12.7-mm.) lag screws and machine bolts (Fig. 51). In addition to the chutes proper, split lagging or old plank is laid on the fill, at the same inclination as the bottom of the chute, to make a better sliding surface, otherwise large chunks become embedded in the fill, and obstruct

the passage of the ore. The chutes are stopped by planks placed in the chute hooks; the ends of these planks are cut to prevent the trammers' hands being pinched between them.

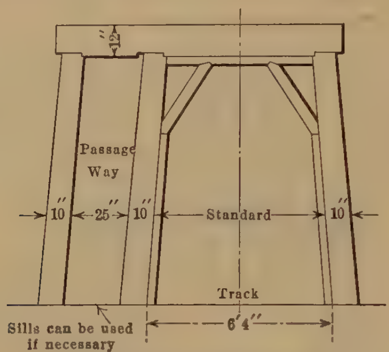


FIG. 49.—THREE-LEG SET.

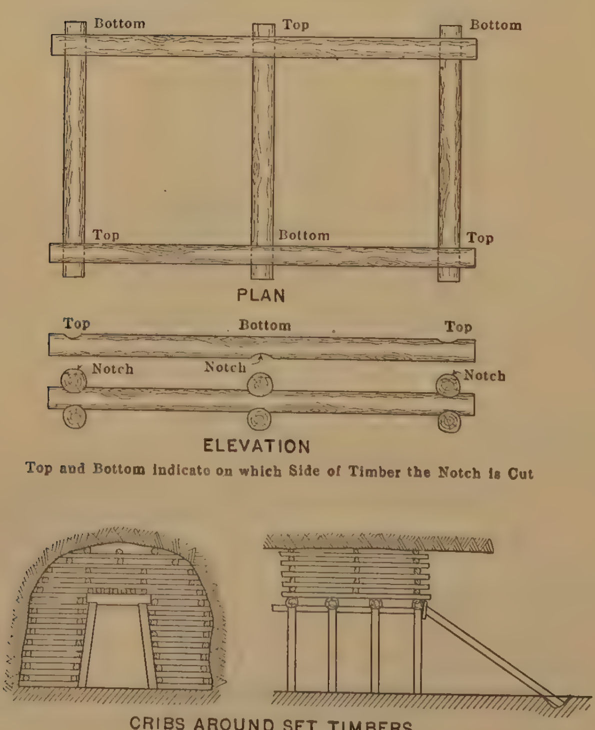


FIG. 50.—CRIB DETAILS.

At the bottoms of ore or rock passes, chutes of similar type but larger are also used, with sets, but the plank and timbers are protected by steel plates and angles.

Stull chutes are also used at the ore raises, the advantage of these over sets being that there is more room on the level, around the cars. The

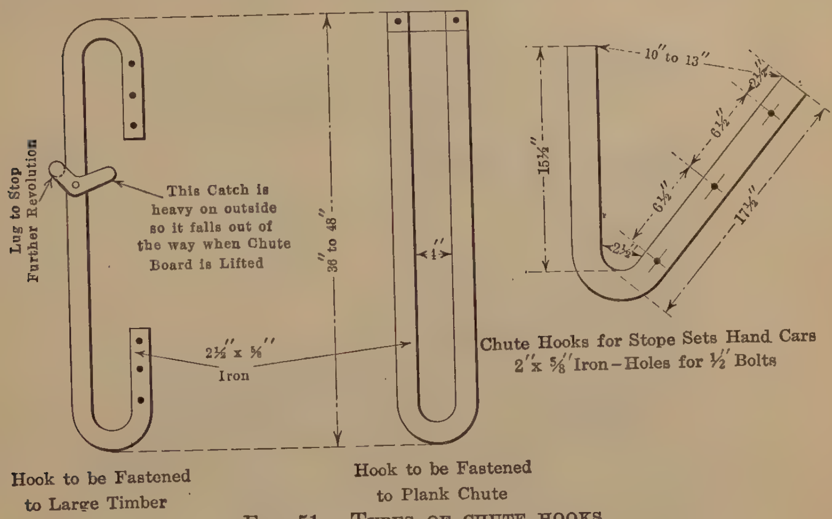


FIG. 51.—TYPES OF CHUTE HOOKS.

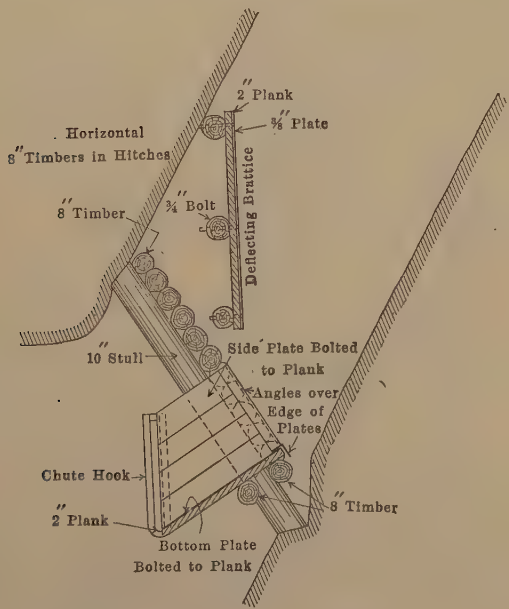


FIG. 52.—PLATING AND BRATTICING CHUTE.

long chute hooks are used on these, too, and catches are placed on them to hold the boards out of the way when the chute is running (Fig. 51).

Another feature of some chutes is an inside deflecting brattice to

protect the chute timbers from blows when the chute is empty. The stulls are put in approximately normal to the inclination of the raise, and hence would get a direct blow from a chunk coming down the raise. The

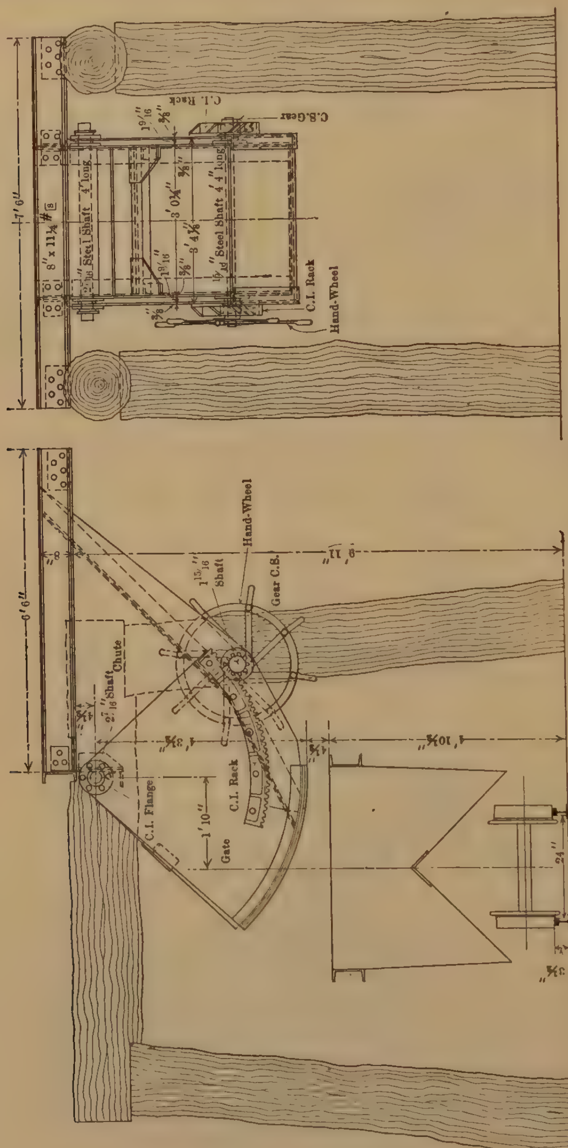


FIG. 53.—FEED GATES FOR CHUTES.

deflecting brattice is put in so that it leans back from the perpendicular, making an angle of 120° or more with the center line of the raise. This brattice is plated and fulfills its purpose very well (Fig. 52).

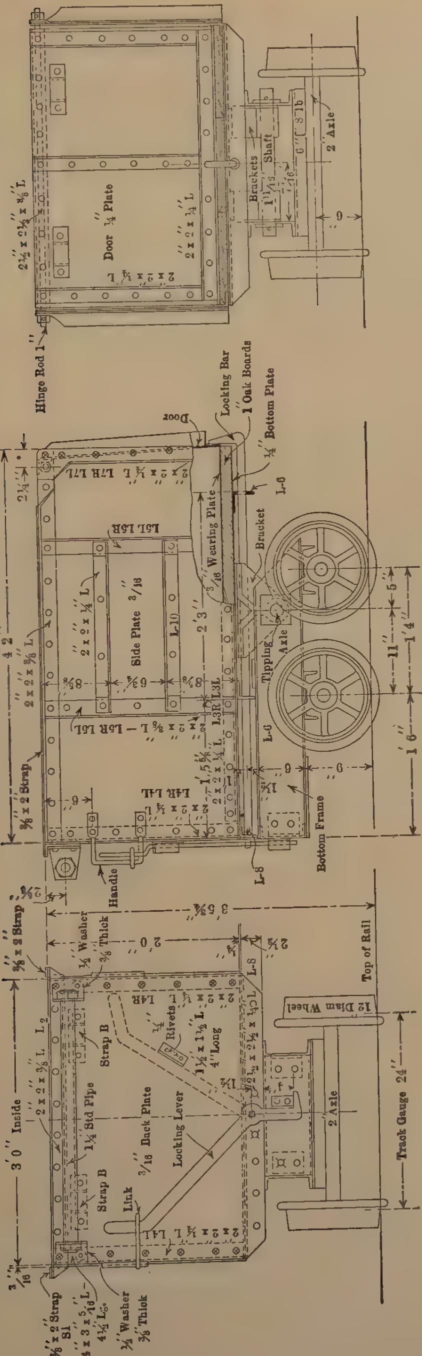


FIG. 54.—END-DUMP MINE CAR.

For loading the electric haulage cars at the ore chute, steel chutes are hung from steel girders or timber sets. Arc gates cut up under the flow of ore to stop it. They are hand-operated with hand wheels, pinions, and racks and are very efficient. The undercut principle prevents the chute being jammed by a large chunk while the fine ore continues to run out, for the chunk is either thrown out of the chute as it closes or is thrown back into it (see Fig. 53).

There are various chutes in the mine which differ in minor details from the types described, but the differences are not great enough to merit special mention. In the sketches of the ore passes and rock passes, these various types are shown on a small scale (Figs. 42, 43 and 44).

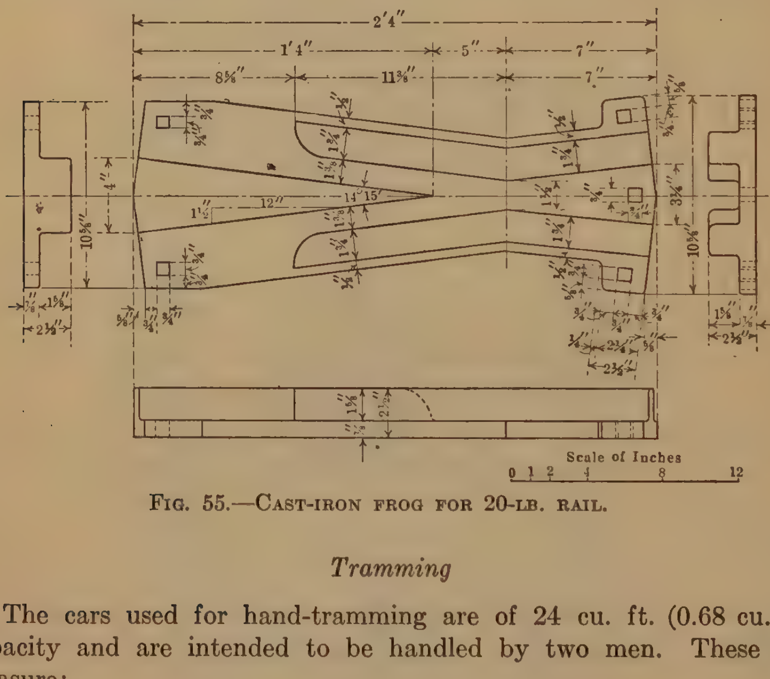


Fig. 55.—Cast-iron frog for 20-lb. rail.

Tramming

The cars used for hand-tramming are of 24 cu. ft. (0.68 cu. m.) capacity and are intended to be handled by two men. These cars measure:

Length overall......	58 in. (1.5 m.)	Length inside......	50 in. (1.27 m.)
Height overall......	42 in. (1.07 m.)	Height inside......	24 in. (0.6 m.)
Width overall......	41 in. (1.04 m.)	Width inside.......	36 in. (0.9 m.)

They weigh complete approximately 2000 lb. (907 kg.) and have an average capacity of 1.4 tons of ore. The wheels are 12 in. (0.3 m.) in diameter and the wheel base is 16 in. (0.4 m.) and 24 in. (0.6 m.) gage. The wheels are a Hyatt roller-bearing type, lubricated with soft grease. A few old ones of a different type are still in use, in which black oil is the lubricant, but the grease wheels are preferred. The weight of the rail used is 20 lb. per yard. Fig. 54 shows the detail of the cars.

In general, track is laid without fishplates at a gage of 24 in. (0.6 m.) and the ends of adjacent rails are spiked to the same tie. This works very satisfactorily, and avoids the necessity of drilling short lengths. The frogs are of cast iron, and are spiked to the ties with blacksmith's spikes (see Fig. 55). The switch points are square iron, pointed, pivoted on spikes and connected with a latch. A kick of the foot serves to throw them.

All cars are now provided with clips on each end so that "safety-boards" can be used to prevent chunks from rolling over the end onto the trammers' feet. This happened frequently when the cars were nearly full, and was a source of many accidents, before boards were used (see

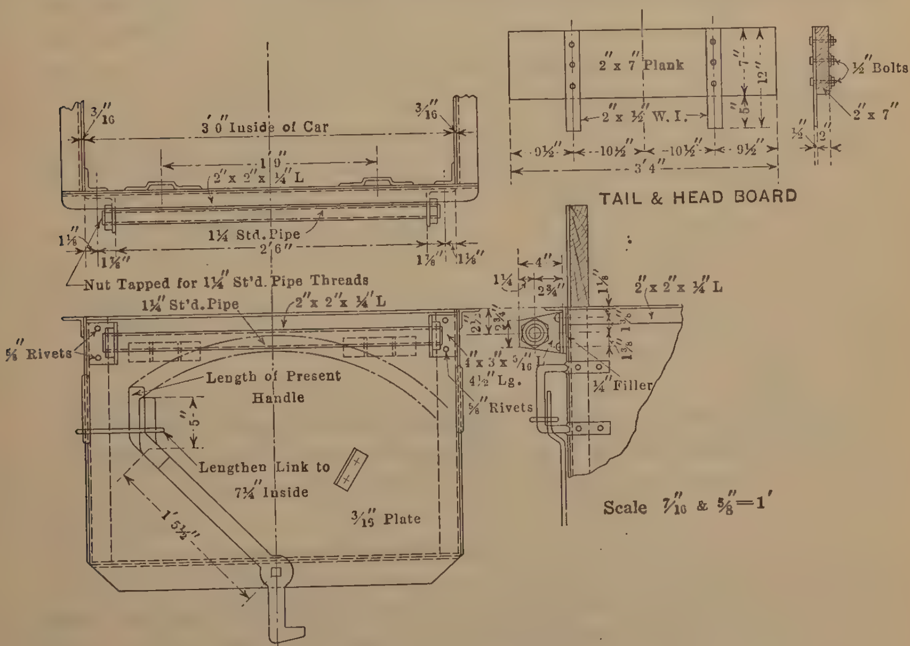


Fig. 56.—Suggestions for End-dump Mine Car.

Fig. 56). Another form of accident arose from chunks on the top of the car rolling on to the trammers' hands as they were pushing the car. A safety handle is now on the back end of each car, to prevent this. In spite of this provision the men frequently neglect to use the handles, but the number of accidents from this source has been materially reduced.

When dumping over the end of a track, curved rails are used to stop the wheels, but a piece of lagging chained to the track is frequently used in front of the curved rails. The block stops the car, and the curved rails act as a second stop if something goes wrong with the timber. A timber is also placed in position to catch the top of the car, at the track

end, to prevent the car overturning into the chute, if for some reason the door fails to open and empty the load.

At some places the cars are dumped between the rails. A stopping block only is necessary, as the rails continue across the hole into which the contents of the car are dumped. These openings are protected by doors made of steel plates, reinforced on the bottom with an angle iron, and hinged on one end for opening. To facilitate the opening and closing of these doors, without stopping the cars or having to pass around them, a system of pulleys and rope is used (Fig. 57). If cars come from two directions, two doors are generally used, but in one instance a combination door was made which operated for either direction. Should it be necessary for some cars to pass over these dumps without dumping, this rope control has to be dispensed with and the doors opened by hand. The stopping blocks, also, cannot be used in this case, so it is necessary

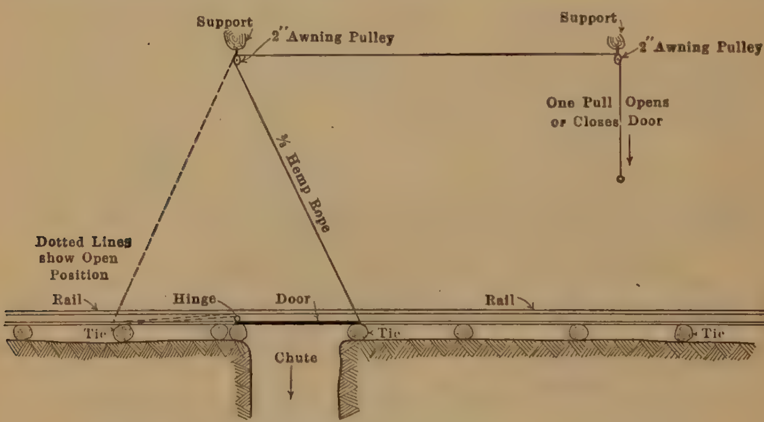


Fig. 57.—Scheme for opening chute door.

for the car to come to the dumping spot very slowly, and stop; the trammers then block the wheels before dumping. This same idea of rope control of doors is used to open the trap doors at ladder roads: the end of the rope is passed through the platform, so that the door may be opened or closed from above or below, by a pull on the rope.

A pair of trammers is expected to load, dump, and tram a car. When a stope has no open chutes two men work at a car, unless large chunks are encountered in numbers, then a third man, called a special trammer, is placed at the chutes to keep the chutes in good shape for loading. He drills block holes in the larger pieces with a stoping machine and blasts them when necessary. It is the general practice to use two cars at a place where a special trammer is working, or to let him take care of the chutes in two adjoining stopes.

Operating figures for tramming naturally vary greatly, as many

variables enter into the work, such as tramming distance, condition of the chutes, etc., but some of the best results as well as some average monthly and yearly figures are given in Table 5.

TABLE 5

Date	Tons Trammed per Man Tramming	No. of Cars	Cars per Man	Tramming Distance	Ton-Miles	Tons	No. of Man-Shifts	Tons per Car
Year, 1916...	16.1	508,316	15	250	27,382	552,625	34,217	1.1
Best figures..	46.6	1,791	36	100	44.3	2,330	50.0	1.3
	52.1	1,504	26	200	114.9	3,030	58.5	2.0
	37.1	2,117	22	50	33.2	3,620	97.0	1.7
	41.0	1,578	41	100	38.9	2,050	50.0	1.3
	36.9	1,406	26	200	75.6	1,990	54.0	1.4

Tramming Bonus System

In January, 1912, a bonus system was instituted to encourage better work from the trammers, and the following tables were formulated, showing the tonnage necessary to be trammed a given distance before a bonus was paid, also the amount per ton paid for every ton above the bonus mark. When this system started, the weights were recorded in gross tons of 2240 lb. and the trammers' wage was $1.70 per 10-hr. shift; net tons of 2000 lb. were later used as a basis of computation and eventually the wages rose to $1.85 per 10-hr. shift. The considerable change in the labor early in 1913, with the acquirement of less capable and ambitious trammers, resulted in few tramming bonuses being earned after the month of April, and the system was discontinued in July, when the length of a working shift was changed from 10 to 8 hr., whereas previously some energetic men had consistently earned good bonuses of say $10 to $30 each month.

The following tables, Nos. 6 and 7, were computed by formulæ and field observations and checked against the 2 years' records of individual stopes and were different for the two cases of closed and open chutes.

Tramming Equations

Let:

T equal total time of a shift, 600 min.

t_1 equal average time of loading, dumping, and tramming one car of ore and returning the empty, inclusive of all delays.

t equal average time of loading, dumping, and tramming one ton of ore and returning the empty, inclusive of all delays.

l_1 equal average time of loading, dumping, and delays per car of ore.

l equal average time of loading, dumping, and delays per ton of ore.

n_1 equal average time taken to tram loaded car 100 ft. and return with empty equals 0.8 min.

n equal average time taken to tram loaded car 100 ft. and return with empty per ton of ore.

x equal distance of stope chute from dump in hundreds of feet.

W equal weight of ore in gross tons, trammed per man per shift to reach bonus mark for a tram lead of any particular distance.

B equal bonus in cents per gross ton on the tonnage trammed above that required by the total shifts of labor in order to reach the bonus mark for any particular tramming distance; this bonus to be divided amongst the total shifts of tramming labor chargeable to that place.

Assume (from past records):

1. The average capacity per car is 1.4 gross tons.
2. The ratio of time $l_1:n_1$ for open chutes is 10:1.
3. The ratio of time $l_1:n_1$ for closed chutes is 16:1.

Now,

$$t_1 = l_1 + n_1X \text{ and } t = l + nX$$

also,

$$t = \frac{t_1}{1.4} = \frac{l_1 + n_1X}{1.4}$$

Since two men push a car,

$$W = \frac{T}{2t} = \frac{T}{2(l + nX)} = \frac{1.4T}{2(l_1 + n_1X)} = \frac{1.4T}{2n_1\left(\frac{l_1}{n_1} + X\right)}$$

therefore,

$$W = \frac{840}{1.6\left(\frac{l_1}{n_1} + X\right)} = \frac{525}{\left(\frac{l_1}{n_1} + X\right)}$$

Allowing labor one-half the profits and assuming a wage of \$1.70 per shift,

$$B = \frac{170}{2W} = \frac{170}{1050}\left(\frac{l_1}{n_1} + X\right) = 0.162\left(\frac{l_1}{n_1} + X\right)$$

Tons trammed per man-shift,

$W = \dfrac{525}{\left(\frac{l_1}{n_1} + X\right)}$ where $\dfrac{l_1}{n_1}$ is the ratio of average time to load and dump 1 ton of ore, inclusive of delays, to the time taken to tram a loaded and an empty car 100 ft.

x equals tramming distance from stope to dump.

Bonus rate per ton (in cents),

B equals $0.162\left(\frac{l_1}{n_1} + X\right)$. If two-thirds of profits go to labor, $B = 0.216\left(\frac{l_1}{n_1} + X\right)$.

TABLE 6.—*Tramming Curves for Bonus*

Distance from Stope to Dump in Feet	Open Chutes $\frac{l_1}{n_1} = \frac{10}{1}$			Closed Chutes $\frac{l_1}{n_1} = \frac{16}{1}$		
	W in Gross Tons	*B* in Cents, to Labor of Profits		*W* in Gross Tons	*B* in Cents, to Labor of Profits	
		½	⅔		½	⅔
0	52.5	1.62	2.16	32.8	2.59	3.46
50	50.0	1.70	2.27	31.8	2.67	3.56
100	47.7	1.78	2.38	30.9	2.75	3.67
150	45.6	1.86	2.48	30.0	2.83	3.78
200	43.7	1.94	2.59	29.2	2.92	3.89
250	42.0	2.02	2.70	28.4	3.00	4.00
300	40.4	2.10	2.81	27.6	3.08	4.10
350	38.9	2.18	2.92	26.9	3.16	4.21
400	37.5	2.27	3.03	26.2	3.24	4.32
450	36.2	2.35	3.13	25.6	3.32	4.43
500	35.0	2.43	3.24	25.0	3.40	4.53
550	33.9	2.51	3.35	24.4	3.48	4.64
600	32.8	2.59	3.46	23.8	3.56	4.75
650	31.8	2.67	3.56	23.4	3.64	4.86
700	30.9	2.75	3.67	22.8	3.73	4.97
750	30.0	2.83	3.78	22.4	3.80	5.07
800	29.2	2.92	3.89	21.8	3.89	5.18
850	28.4	3.00	4.00	21.4	3.97	5.29
900	27.6	3.08	4.10	21.0	4.05	5.40
950	26.9	3.16	4.21	20.6	4.13	5.51
1,000	26.2	3.24	4.32	20.2	4.21	5.62

Calculated on a basis of net tons of 2000 lb.

Tons trammed per man-shift,

$$W = \frac{588}{\left(\frac{l_1}{n_1}\right) + X}$$ where $\frac{l_1}{n_1}$ is the ratio of average time to load and dump 1 net ton of ore, inclusive of delays, to the time taken to tram a loaded and an empty car 100 ft.

X equals tramming distance from stope to dump.

Bonus rate per net ton (in cents),

if one-half of profits go to labor, B equals 0.1447 $\left(\frac{l_1}{n_1} + X\right)$.

if two-thirds of profits go to labor, B equals 0.1929 $\left(\frac{l_1}{n_1} + X\right)$.

TABLE 7.—*Tramming Curves for Bonus*

Distance from Stope to Dump in Feet	Open Chutes $\frac{l_1}{n_1} = \frac{10}{1}$			Closed Chutes $\frac{l_1}{n_1} = \frac{16}{1}$		
	W in Net Tons	*B* in Cents, Labor One-half Profits		*W* in Net Tons	*B* in Cents, Labor Two-thirds Profits	
		$1.70	$1.85		$1.70	$1.85
0	58.8	1.93	2.10	36.7	3.08	3.35
50	56.0	2.03	2.21	35.6	3.18	3.46
100	53.4	2.12	2.31	34.6	3.28	3.57
150	51.0	2.21	2.40	33.6	3.37	3.67
200	48.9	2.31	2.51	32.7	3.47	3.77
250	47.0	2.41	2.62	31.8	3.57	3.88
300	45.2	2.51	2.73	30.9	3.66	3.98
350	43.6	2.61	2.84	30.1	3.76	4.09
400	42.0	2.70	2.94	29.3	3.86	4.20
450	40.5	2.79	3.04	28.7	3.95	4.30
500	39.2	2.89	3.14	28.0	4.04	4.40
550	38.0	2.99	3.25	27.3	4.14	4.50
600	36.7	3.08	3.35	26.7	4.24	4.61
650	35.6	3.18	3.46	26.2	4.34	4.72
700	34.6	3.28	3.57	25.5	4.44	4.83
750	33.6	3.37	3.67	25.1	4.53	4.93
800	32.7	3.47	3.77	24.4	4.62	5.03
850	31.8	3.57	3.88	24.0	4.72	5.14
900	30.9	3.66	3.98	23.5	4.82	5.24
950	30.1	3.76	4.09	23.1	4.92	5.35
1,000	29.3	3.86	4.20	22.6	5.02	5.46

Electric Haulage

The type of cars used for the electric haulage is shown in Fig. 58. These cars are 8 ft. (2.5 m.) overall, 4 ft. 6 in. (1.37 m.) wide, and 55 in. (1.397 m.) from the rail to the top. The rated capacity is 66 cu. ft. (1.87 cu. m.) and for an average their load is about 4½ tons.

The special feature of these cars is the method of operating the doors. As may be seen in the illustration, the toggles controlling the doors are of such shape that when the doors are closed, the lines between the two pivoting points pass below the axle on which the operating wheel works. This locks the doors so well that more pressure on them only tends to lock them more tightly. To open, rather than use the wheel, the men pry the toggles up with a bar, until the line passes above the center of the axle, which releases the lock, and the pressure on the doors swings them open. A quick turn of the hand wheel is all that is needed to close and lock the doors. Should a chunk too large to pass the door be in a car, the arm may be removed from the lugs, thus allowing the door to swing farther open.

To dump a train load of 10 cars, including the closing and locking of the doors, takes two men from 60 to 90 sec., as a rule. Two cars are dumped at a time, then the motorman moves ahead two car lengths, etc. The tracks over the pockets for dumping the motor cars are supported

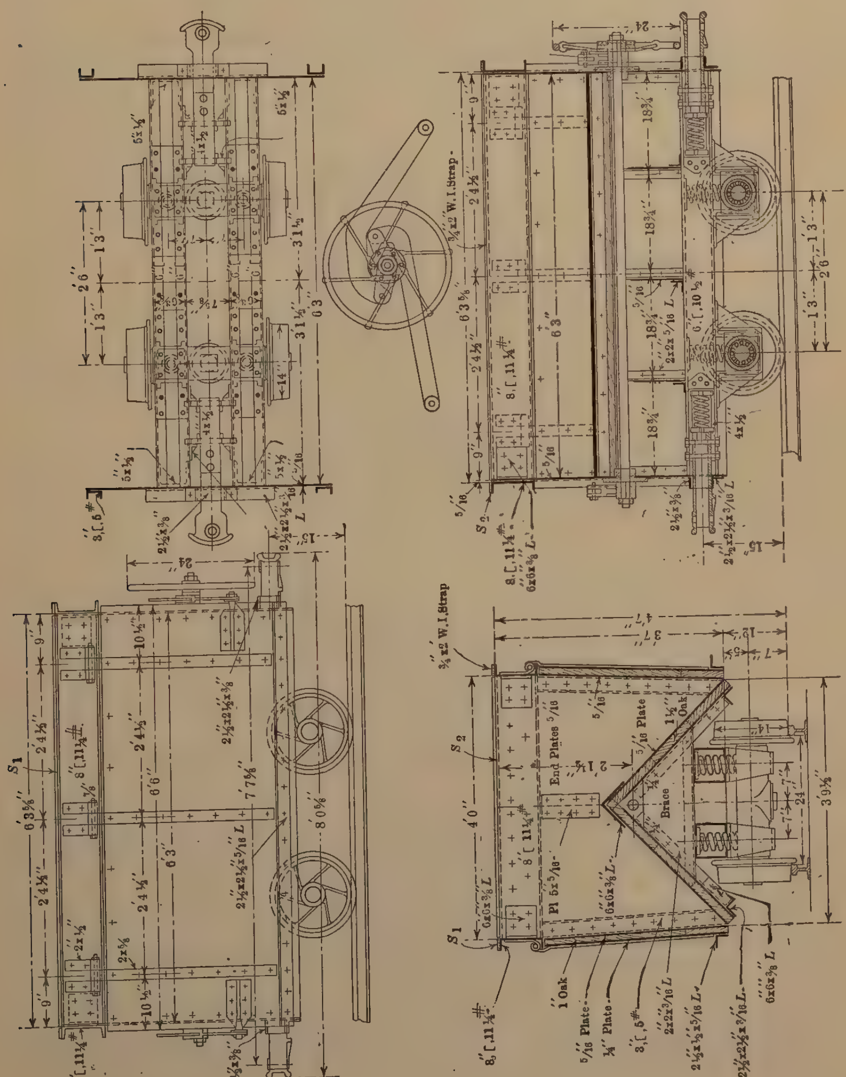


FIG. 58.—UNDERGROUND HAULAGE CAR.

by large *I*-beams. Platforms are built for the men to stand on, and are clear of the run of the load. For other raises, where the span is not so great, three timbers support the track.

The General Electric Co.'s locomotives 6 to 7½ tons, type LM.101–

M1 using 250 volts direct current and giving 2500 lb. drawbar pull at 7½ miles per hour are used. The rails weigh 30 and 40 lb. per yard set to 24-in. (0.6-m.) gage.

No. 2/0 copper trolley wire is used for the current, which has a potential of 250 volts. Standard types of insulators are used, some screwed on pipes, some screwed or bolted to the timber, and some held by suspension guy wires. The wire is bare when 7 ft. or more above the rail, but when it has to be brought lower it is protected by guards of 1 by 8-in. boards. When passing under chutes, special iron brackets are used,

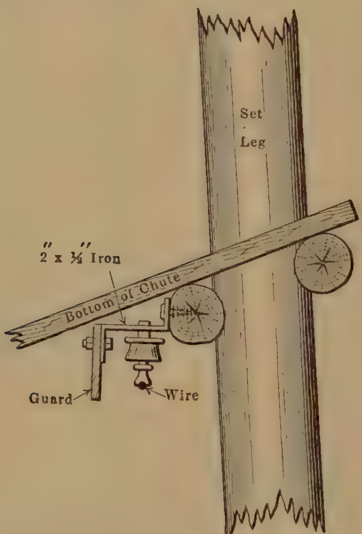


FIG. 59.—SUPPORT OF TROLLEY WIRE UNDER CHUTE.

to support the insulator and the guard boards. These are shown in Fig. 59.

A motor crew consists of one motorman and two trammers. It is rather difficult to give exact figures for the whole of the electric haulage work, as it varies greatly, one day carrying a heavy load and the next a light one, but for pulling the ore from the ore chutes to the pockets the figures of Table 8 will give some idea of the work accomplished.

TABLE 8

Date, 1917	Tons Trammed	Cars Trammed	Tons per Car	Distance Trammed	Ton-Miles	Labor Shifts	Tons per Man
January.......	18,540	6,271	3.0	980	3,455.7	137	135.2
February......	27.950	6,585	4.2	700	3,704.8	128	193.2
March.........	39,580	11,435	3.5	660	4,949.1	205	196.2
April..........	41,180	10,323	4.0	640	5,026.0	199	206.0
Average.......	127,250	34,614	3.7	710	17,135.6	669	191.0

Transverse Stopes

Until about 2 years ago practically all the ore except that obtained from development work was obtained from the shrinkage stopes. Since that time the mining of some of the pillars has been started, but still the greater part of the ore comes from the stopes. These stopes are of two kinds, the transverse stopes, 17 ft. (5.2 m.) wide with their length at approximately right angles to the strike of the orebody, and the longitudinal stopes, worked where the vein is narrow, with their length parallel to the strike of the ore.

These stopes are started at the bottom of the orebody, or in some cases at a level about halfway between the bottom and surface. A floor pillar has to be left under this level in the latter case. The stopes are carried up from level to level, using the same plane for a center, at all times, and the same cycle of operations is carried out from level to level, with such local modifications as the case demands. The description here given covers the general method of procedure shown in Figs. 60 and 61.

For the first lift of the stope, it is necessary to get an opening from the foot wall to the hanging wall. This is done by a crosscut on the lowest level, which is then stripped to the width of 17 ft. (5.2 m.) and to a height of 12 to 15 ft. (3.6 to 4.6 m.). This work has to be done but once, for the subsequent stopes will be carried far enough above the next level to meet the requirements of headroom and width. Another feature met with on the start, but not later, is the ore below the bottom of the level. This has to be benched out, down to the underlying rock, until the whole section is cleaned. Filling is then put on this bottom, until it is filled up to the elevation of the tramming level.

Sills are now laid on this fill, in the proper places, and the sets are erected on them, as described under the timbering work; the chutes are put in the proper places, and the track laid, filling is brought into the stope by cars, dumped on "sollars" on the track, and shoveled behind the sets, up to the height of the chutes. This fill steadies the sets, and keeps the ore, to be broken, above the chute level. As the fill is lighter than the ore, it occupies a larger volume than the same weight of ore, so it is cheaper to muck the fill behind the sets than it would be to muck the ore from behind the sets into cars, also, the amount of time spent under the hazard of an empty stope with remote back is lessened thereby. With the sets erected, covered with lagging, and the fill packed behind the sets, the stope is ready for breaking to begin.

A raise on the foot wall is first carried up to the level above and this is the only time the raise needs to be started from the timbers, for when the stope is completed to the level above, the raise is carried up again before the muck in the stope is pulled down. The object of the raise is

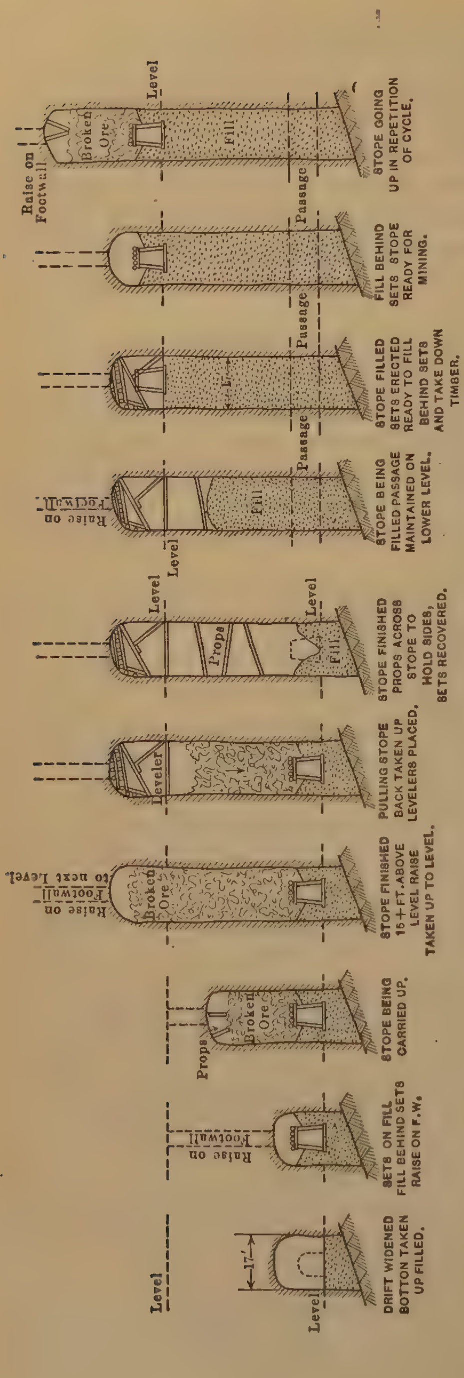


FIG. 60.—TRANSVERSE SECTIONS OF TRANSVERSE STOPE.

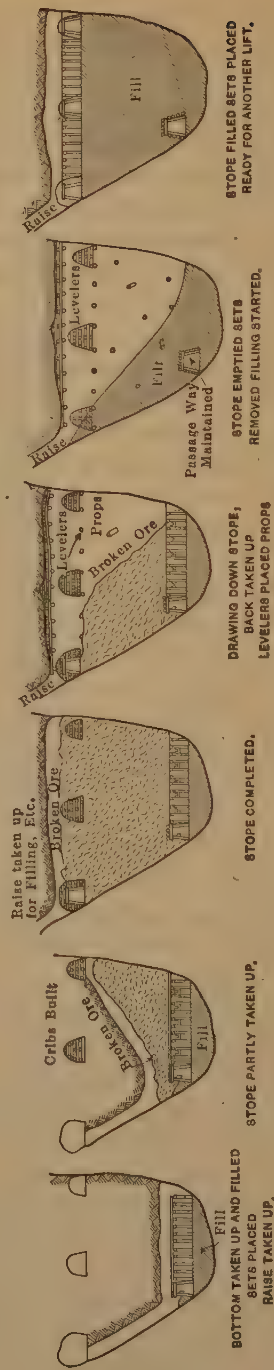


FIG. 61.—LONGITUDINAL SECTIONS OF TRANSVERSE STOPE.

twofold: to afford ventilation, and to give a means of entrance after the sets are covered with the broken ore.

After the raise is completed, stoping begins. The work is started at the hanging wall. Light holes are drilled to break to the hanging wall and to form a cushion on which to blast the stronger holes; 8-ft. holes inclined from 50° to 60° are drilled and fired against this cushion; thus the timbers suffer no damage.

Stoping machines with the direct type of air feed are used with a cross bit of solid steel ⅞ in. (22.225 mm.) in diameter and quarter octagon in section, having a raised center. This subject of drills and bits has been treated at length by Mr. Tillson.[2] The holes are drilled in rows

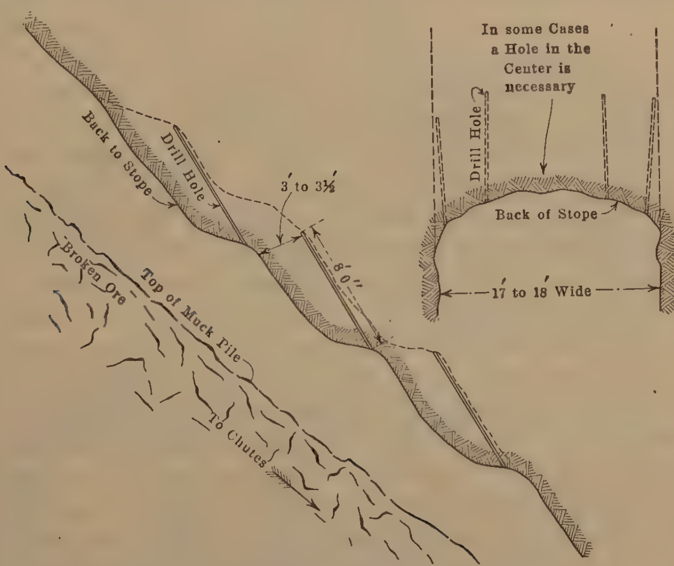


FIG. 62.—HOLES IN TRANSVERSE STOPE.

across the back and three to five are necessary to break the 17 ft. (5.2 m.), depending on the ground. In some stopes the ground has been so bad that hardly any drilling was necessary, for as the loose was barred down, more would open up. Most of the stopes are out of such ground now, so that all the ground broken has to be drilled. The holes are drilled at an inclination of about 50° to 60° from the horizontal, with as much burden as the ground allows. In general, 3 or 3½ ft. burden on a hole will break well with from 6 to 8 sticks of powder in an 8-ft. hole. The firing is done with fuse. Fig. 62 shows the arrangement of holes in a stope.

[2] *Trans.* (1915), **51,** 240. The same paper revised to date was printed in the *Engineering & Mining Journal* for April 7, 1917.

After firing, all the loose ground is cleaned down with bars and gads. Should some bad ground resist these efforts it is securely propped and drilled. Two runners work together and it devolves on both of them to make the working place safe. After the cleaning is completed, one takes the stoping machine, and the other with the block-hole machine drills short holes in the large chunks and slabs so that they may be broken with powder. Those chunks that are large, but not large enough to need a block hole, are broken with 16-lb. sledges.

The opening at the hanging-wall end gradually becomes larger as the stope opens up, and ultimately the timbers become covered and the raise on the foot wall has to be used for entrance, unless the stope has already broken into some opening on the hanging-wall side of the level above. As the stope is carried up, the back is kept as nearly as possible at the slope of the muck pile, so that the miners have about 6 to 8 ft. (1.8 to 2.5 m.) of head room in which to work. Whenever cribbings are encountered in the progress of the work, they are filled with broken ore to give them greater stability. This has to be done mostly by mucking, though some muck runs into them.

During the mining or "breaking" operations, trammers draw some of the broken ore from the chutes, but this drawing has to be regulated so as to keep the muck pile the proper distance from the back.

The breaking continues until the back of the stope has been carried 15 to 18 ft. (4.6 to 5.5 m.) above the next level above that from which it started. The raise for the next stope is then put up on the foot wall, which finishes for that section of the stope the work of mining.

The broken ore is then pulled down for the length of the stope until the top of the pile is a few feet below the elevation of the upper level. While this is being done, timbermen have been started at one end, either hanging or foot wall, to take up the back of the stope with timber, and if necessary the sides are propped or laced, as shown in Fig. 63. This work is a precautionary measure, to prevent the fall of chunks that have become loose after the muck pile is so far below the back that inspections cannot be made. As this timber is placed, levelers, or horizontal timbers, are placed across the stope at the proper elevation for the track which will later be placed for the fill cars. Bridges are thrown across the stope when needed for haulageways.

As soon as the trammers have finished pulling the muck low enough for the timbers, the tramming is concentrated at either the foot-wall or hanging-wall ends, depending on the direction of tramming, until a "chute is open" (*i.e.*, the ore has reached the position where no more will roll into the end chute). During this time men have occasionally inspected the sides that were being exposed; all small loose rock pieces have been cleaned down and heavy pieces propped with timbers. From the time an open chute is obtained, two men work above the timbers and

trim the sides, block hole, and sledge large chunks, prop the sides, and keep the angle of the ore pile trimmed, so that no sudden run of ore may start. This results in better muck reaching the chute, and a consequent reduction in time necessary to load the tram cars. The direction of tramming, mentioned above, is so arranged that the car going from the

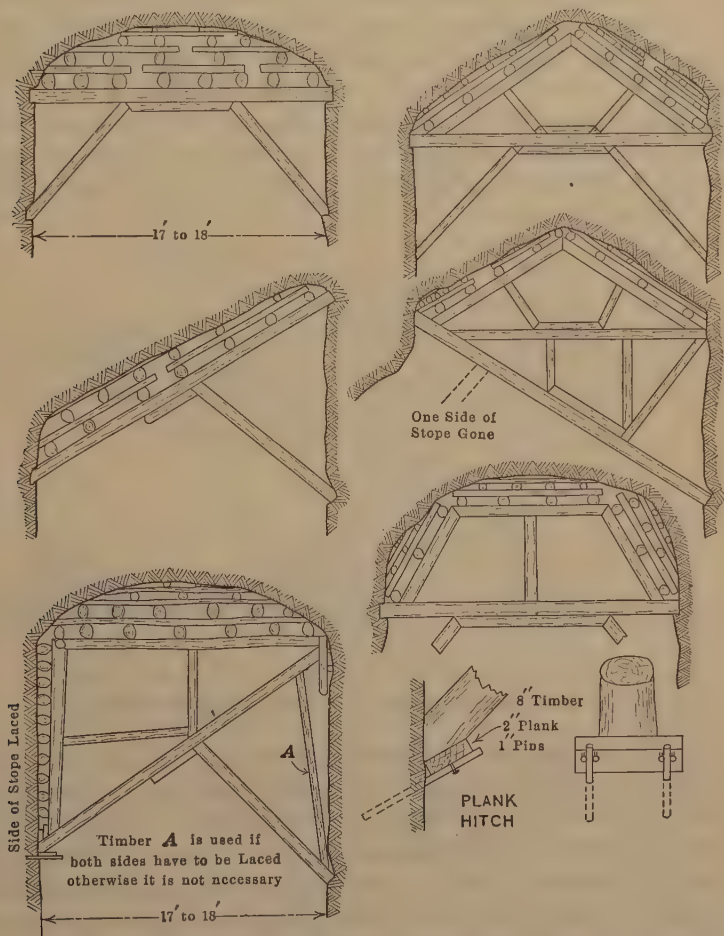


FIG. 63.—METHODS OF TAKING UP BACKS OF TRANSVERSE STOPES.

chute will always be under timber sets that still have ore on them. In this way, should something fall from the back of the stope and strike bare timbers, the trammers will not be injured, and the work tied up. This tramming continues until the ore is all out of the stope. The last stages of work are shoveling into chutes, the ore that will not run, and lifting the lagging on top of the sets, so that the ore on and between them will fall to the track below, and may be shoveled up. The track and timber

sets are now removed, together with all the lagging and timbers that are suitable for use again. This timber is generally pulled to the level above, with a small air hoist, mounted on a drill column, and stored near the stope, for use when the stope is filled. In case there is not room for storage on the level above, the timber is stored where convenient, and taken to the stope again when needed. Where passageways have to be maintained through the filled stope, the sets for them are erected, covered, and braced both inside and out, against the direction from which the fill is to come.

The stope is now ready for filling. When possible, it has been the custom to run the foot-wall raises up to a motor level, even though that is several levels above the working level. When this has been done,

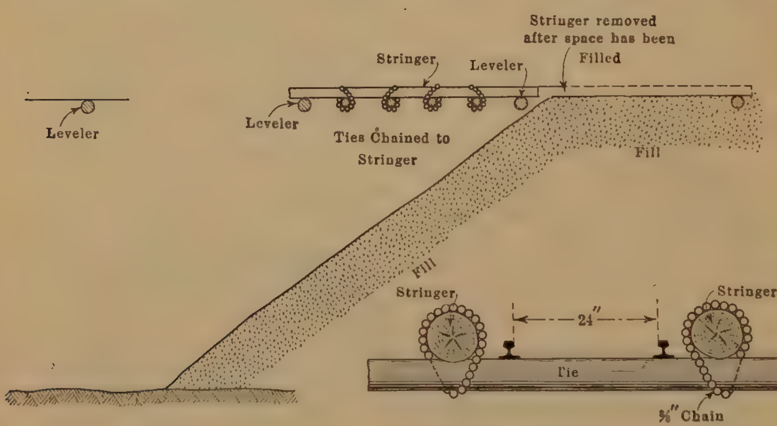


Fig. 64.—Support of fill track in transverse stope.

the rock filling is dumped by the motor down the raises until there is enough in the stope to make room for a chute and car. To do this, the filling pile is leveled to the proper grade by shoveling, and the chute and track put in place. If possible, the chute is built with set timbers, placed so that they may later be used for mining the next lift, but where this cannot be done a temporary chute is used. The remainder of the stope is then filled with the tram car, which is loaded at the rock chutes. When the stope cannot be filled to this stage by dumping through the raise directly from the motor car, the filling is brought on the upper level by car from some other raise series to the edge of the stope and dumped into it. When the filling reaches the upper level, the tracks are extended and the remainder of the stope filled.

The track for filling purposes is supported on the leveler timbers already mentioned. Formerly stringers were laid on these timbers and ties on the stringers, but this made it necessary, when recovering timbers,

to dig down several feet in the fill in order to get them. The present method is to place the levelers with their tops at the elevation of the rails. The stringers are laid on the timbers as before, but wide enough apart for the car to pass between them, and the ties are chained to the under side of the stringers, so that, when the stope is filled, all timbers are in sight (see Fig. 64).

Sometimes it is possible to fill the stope so that the ore-track grade is the same as the grade used for filling, but frequently the final grade is opposite to that of the filling grade. In such a case the filling is either trammed down grade or an engine of the "puffer type" is used to pull the car up grade. In the former instance, after the stope is filled, the track is blocked up to the final grade, and the fill completed by pushing the car up the 1½ per cent. grade. The sides are not reached by the

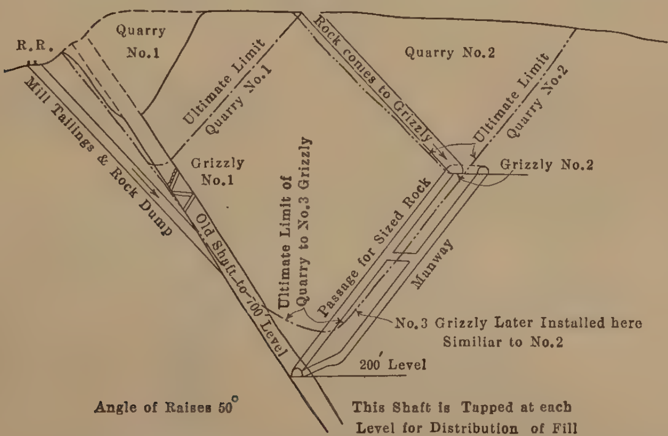


Fig. 65.—Arrangement for quarries at Ding Dong shaft.

car, so they have to be filled by mucking the fill into place, and this has been found, in the majority of cases, to be as economical as the moving of the track to the sides would be.

In cases where the sides of the stope are known to be bad the filling is introduced into the empty part of the stope as soon as possible, and from then on follows the drawing of the ore as closely as practical, the toe of the fill pile being kept 20 to 25 ft. (6 to 7.6 m.) from the toe of the ore pile. It has been necessary in a few cases to leave the timber sets in place and cover them with the filling.

Originally filling was brought in railroad cars from a gravel bank where it had been loaded by a steam shovel, but as soon as practical it was produced from quarries on the property and dead work in the mine and sent directly into the mine through shafts, the Ding Dong shaft particularly. The collar of the shaft was accordingly concreted, and *I*-beams placed to support a railroad track. All the rock rejections

from the picking table and the tailings from the mill are dumped here, and join the filling broken from the quarries. Some of these quarries break directly into the shaft, while the product from others is brought to the shaft by the electric haulage cars, unless it is utilized where the motor cars can take it directly. Fig. 65 shows the arrangement of the quarries at this shaft. The other quarries break to grizzlies placed over raises that lead to other portions of the mine. The grizzlies are rugged in construction, and are so built that parts that become broken may easily be replaced. Fig. 66 shows the construction of one of the rock grizzlies.

Until the latter part of 1912, the rock in the quarries was broken with vertical holes nearly 30 ft. (9 m.) deep. These were drilled with piston machines mounted on tripods. Since then the piston machines have been discarded, and all the drilling is done with 90-lb. hammer drills, of

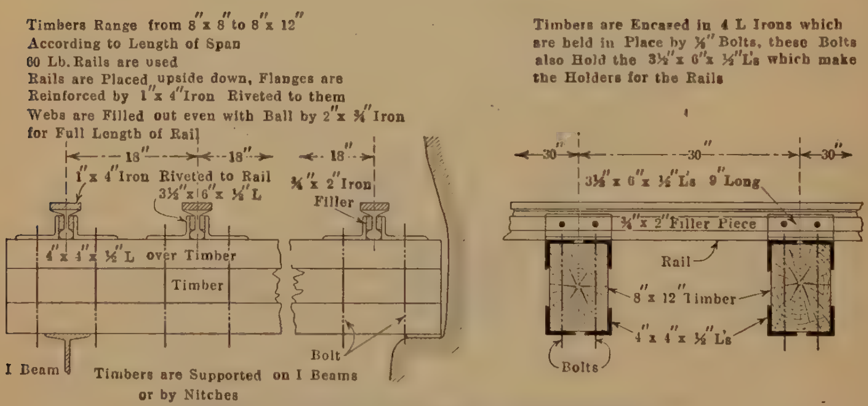


FIG. 66.—DETAIL OF GRIZZLY CONSTRUCTION.

the self-rotating type, used unmounted, and drilling holes 12 ft. (3.6-m.) deep. With the old method a machine crew took a shift or more to drill a 30-ft. hole, but with the hammer machines one man can generally drill close to 100 ft. of holes in a shift. The present practice is a decided improvement over the former, both in regard to amounts of rock broken and to the size of the chunks. There are not nearly so many large boulders which have to be block-holed. Of course considerable block-holing has to be done on the grizzlies, but some boulders have to be broken before they can go through the raises leading to the chutes.

When the filling is completed and the grade properly made, the usual stope sets are erected on sills and, when in position and braced, filling is shoveled behind them to the proper height, and the stope is ready to mine again. The timber gang then removes the timber that was used to take up the back, and the blocking and the large timbers are stored

TABLE 9.—*Operating Figures, Open Cut*

Tons Rock Broken	Tons Broken per Man Breaking	Tons Broken per Drill Shift	Explosives, Cost per Ton Broken	Tons Through Grizzly per Man	1913
11,280	80.0	274	$0.032	13.5	Tons broken per man breaking includes block-holing large chunks after benches were fired. Explosives' cost includes block hole and grizzly powder.
4,640	34.5	280	0.060	18.2	
5,490	22.5	292	0.058	16.0	
3,010	28.4	235	0.015	20.2	
7,640	20.1	210	0.070	12.5	
7,480	39.4	345	0.066	31.3	
5,230	21.1	266	0.065	15.1	
4,440	20.6	364	0.053	13.0	No block holes necessary as rock was shot directly into place through mill hole. Muckers cleaned benches.
6,940	39.7	305	0.018	30.2	
10,710	47.6	402	0.011	40.2	
7,470	58.5	233	0.016	37.8	

for future use, but the lagging is lowered into place on top of the sets. The machines are then started and operations proceed as already outlined.

TABLE 10.—*Operating Figures for Stopes*

Tonnage Broken	Tons Broken per Man-Shift	Tons Broken per Drill-Shift	Tons Broken per Foot of Hole	Explosives, Cost per Ton	Remarks
3,300	68.8	264	2.3	0.030	These are some of the highest figures made in larger stopes over period of 1 month.
1,030	64.5	206	2.3	0.033	
2,310	48.4	170	1.3	0.032	
2,880	62.6	125	1.0	0.064	
2,560	60.9	176	1.4	0.036	
4,880	50.9	130	1.0	0.053	
7,390	69.0	140	1.3	0.039	Two machines to break this tonnage.
636,920	33.0	102	1.0	0.046	Average, year 1916.

Timber cost, 10 c. per ton, includes labor and timber, average for year 1916.

The work expected of the men in stopes is to drill 80 to 125 ft. (24 to 38 m.) of hole per shift per machine, with a resultant breakage of 50 tons per man in stope. The average for 1916 does not show this, but this figure includes many working places in which the average of large stopes could not be expected.

Safety Devices

It is hard to design many mechanical safety devices for use in a stope, but all that is possible to promote the safety of the work is done. The use of props for all loose ground is insisted upon; the territories of the shift bosses are small enough so that each place may be visited four to six times per shift. The holes are so drilled that the entrance to the stope allows the man to approach the newly broken back, on the side of the collars of the holes, hence they can see all the bad back before they come under it. To prevent flimsy stages from being used, each stope has an extension board (Fig. 67) on which to support the machine. When loading holes, the men are not allowed to wear their lamps in their hats,

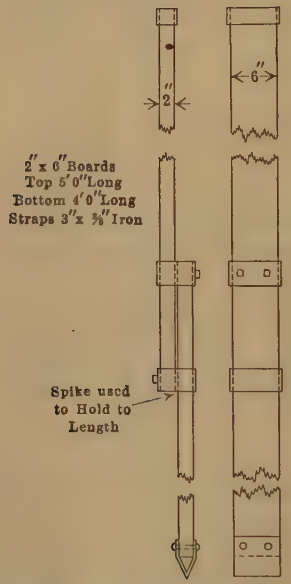


FIG. 67.—STOPE BOARD.

as a fuse hanging down from the back might easily become lighted without the miners' knowledge and result in an accident. When "spitting" fuse, two men always work together, so that should the lamp of one be put out, the other can immediately light it for him. With a large number of holes to be fired, extra men are sent to the stope. The type of carbide lamps now used has the automatic lighting device, a milled steel wheel rubbing on a pyrophoric alloy, so that lamps can be lighted almost instantly. When drawing a stope empty, the men working above the timber sets are required to keep the muck pile at or close to the angle of repose, to prevent a sudden run of the pile and to keep the sides of the empty part well cleaned and propped. With all the precautions taken that can

be thought of, accidents will occasionally happen, but many are undoubtedly saved by care. Lack of care and disregard of instructions are the sources of many.

Longitudinal Stopes

As the transverse stopes were carried nearer the upper portion of the west vein, the distance from foot to hanging walls became smaller and ultimately a point was reached where the lengths of the stopes were not much more than the widths. When this condition was reached, a change in mining methods was made, and the stopes converted to the longitudinal types, where the length is parallel to the foot and hanging walls, and the width is determined by the distance between them. At first, two transverse stope slices and the pillar between were taken out as one longitudinal stope, which gave a length of about 70 ft. (21 m.). Later three stopes and the included pillars were tried, and successfully carried out, a length of 120 ft. (36.6 m.). In several cases, longer stopes than these, even with bad hanging walls, were successfully made, by

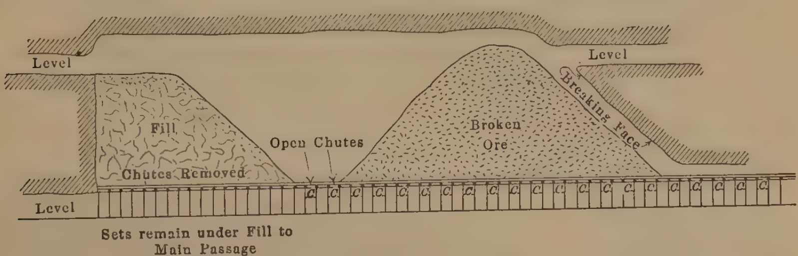


Fig. 68.—Breaking, drawing, and filling in longitudinal stope.

introducing the fill as soon as possible, so that three operations were going on at once—breaking ore at one end, drawing down the stope at the center, and filling at the other end (see Fig. 68).

While the arrangement of the timber sets varies from that of a transverse stope, the same type of sets and chutes are used. Some stopes are wide enough to have two rows of timber sets, one on the foot wall and one near the center. The most important point about the timber of a longitudinal stope is to have the sets as close to the foot wall as possible; it even pays to bench out 4 or 5 ft. of the foot to obtain this condition. The sets should also be braced against being pushed toward the hanging wall, for as the stope goes up the weight on the foot side of the sets becomes much greater than that on the hanging side. Another feature of the longitudinal stope is the difficulty of keeping the ore stripped from the foot wall, for as the stope advances upward, the chutes, even though right on the foot wall, on the level, become under the hanging wall. Consequently the foot side of the stope has to be kept higher than the hanging side, especially in a wide stope, in

order to reach the back all over the stope. The hanging side should be kept the higher to have the face or back normal to the dip, but unless the stope is narrow this is not possible. When the foot wall is very flat, muckers have to be constantly employed to throw the broken ore to the hanging side so that the foot wall may be reached by the drill holes. In some cases, this is too costly, then an imaginary foot wall is carried and this is benched to the true foot wall as the stope is cleared out. Where the dip of the vein is 50° to 60° from the horizontal, experience has indicated 100 ft. (30.5 m.) as the greatest vertical distance that a wide longitudinal stope may be carried because of the above difficulty (see Fig. 69).

The same type of machine is used in the longitudinal stope as in the transverse stope, though occasionally it is necessary to use hollow steel when drilling in the foot wall, as very flat holes are necessary. Block-

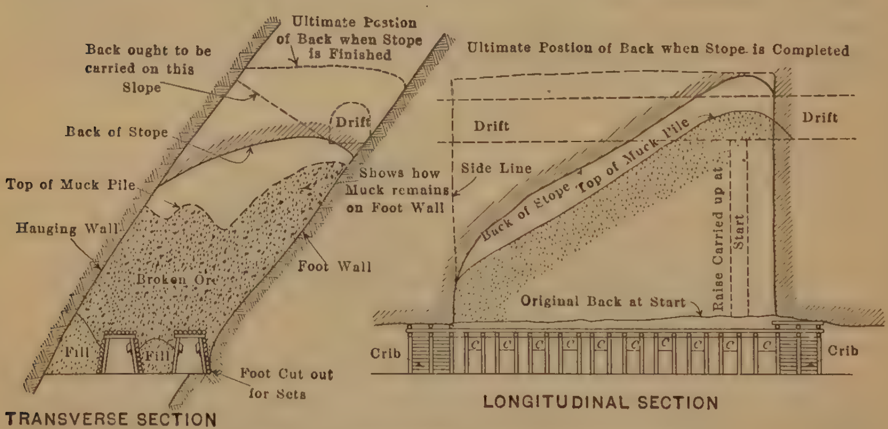


FIG. 69.—SECTIONS OF LONGITUDINAL STOPE.

holing is essential as the ore has a tendency to break in larger masses. Props are used as in other cases, and a thorough cleansing of the back is necessary at all times.

A raise is first carried up on the foot wall, generally at one end, and breaking starts from this, so that the stope first breaks through to the level above at the raise.

As a general rule, it is not desirable to have the width over 20 to 25 ft. (6 to 7.6 m.), but it has happened that the vein has widened as the stope advanced, so that horizontal measurements from foot wall to hanging wall have been 60 ft. (18 m.) and the normal distance 35 to 40 ft. (10.7 to 12 m.). In stopes of this size it is impossible to prop any loose hanging wall, hence it is imperative that the wall be cleaned thoroughly and left absolutely sound, before the muck is withdrawn, because the back cannot be taken up, as in a transverse stope. Fortunately,

only one accident has occurred from a chunk falling out of the back after the muck pile was drawn down, although there have been several cases when chunks have fallen between shifts when no one was present, but this circumstance also applies to the transverse stopes.

When the width of the stope permits, the hanging wall is propped from the foot wall if necessary, and in all cases the foot wall is carefully examined, and any ore left on it is blasted off, block holes or deeper holes being used if necessary.

When the longitudinal stope is empty, it is necessary to leave set timbers standing to afford passage along the level, through the stoped section. The sets in use are straightened into position if they have been moved, new ones put in to replace the crippled ones, and they are braced as thoroughly as possible; this is difficult sometimes, as interior braces cannot be used on account of traffic. When all is ready, the fill is intro-

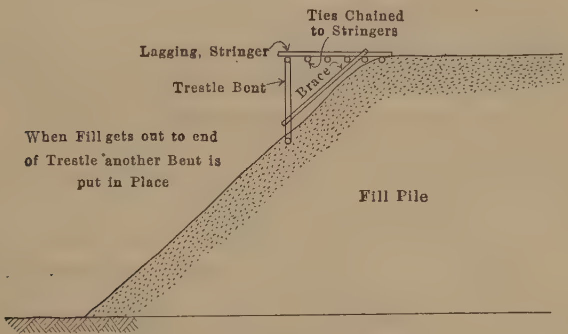


Fig. 70.—Filling trestle.

duced into the stope as in the transverse stopes, either through raises or by cars at one end. No levelers can be placed for the track, so a trestle has to be erected to carry the track over the edge of the fill pile, and the caps on these trestles take the place of the levelers, otherwise filling proceeds as usual (Fig. 70). In cases of bad hanging wall, the filling is introduced at one end, as soon as possible, and is kept close to the talus of the ore pile, but in the majority of longitudinal stopes this has not been necessary.

In several stopes the vein has widened sufficiently to make it unsafe to complete the stope in one operation. These stopes are then carried to completion by what is locally termed "cut-and-fill" methods: The back is carried at an angle of about 40°, the angle of repose of the fill, and when this is obtained, the ore is drawn out and fill run in. Lagging is placed on the fill, and a layer 10 to 12 ft. (3 to 3.6 m.) thick is taken from the back, but keeping it at the same angle. The ore is again drawn down and more filling introduced, etc. By this method it is always possible to prop the back either from the broken ore or from the filling.

This method, of course, is slower and costlier than the ordinary methods, but has a greater element of safety where the ground is treacherous (Fig. 71).

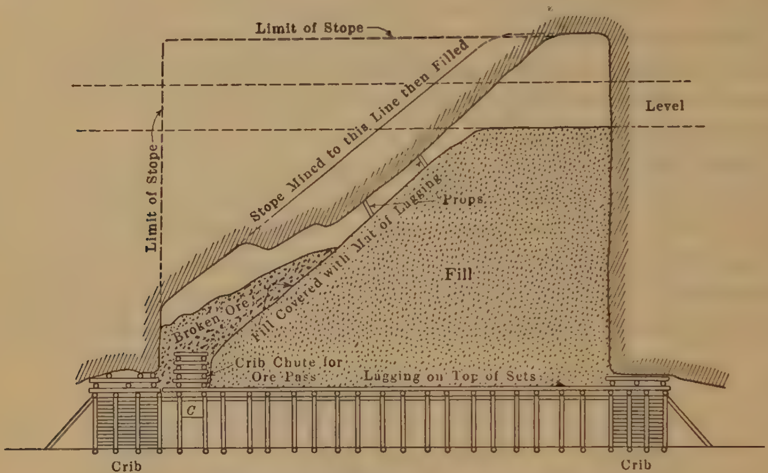


Fig. 71.—Longitudinal stope, cut-and-fill method.

In another instance, a very flat hanging wall slabbed so badly that it was necessary to take horizontal sections of 10 ft. in height, replacing the ore by trammed filling, and shoveling and passing all the ore down through cribbed chutes. Had it been possible to bring filling from the

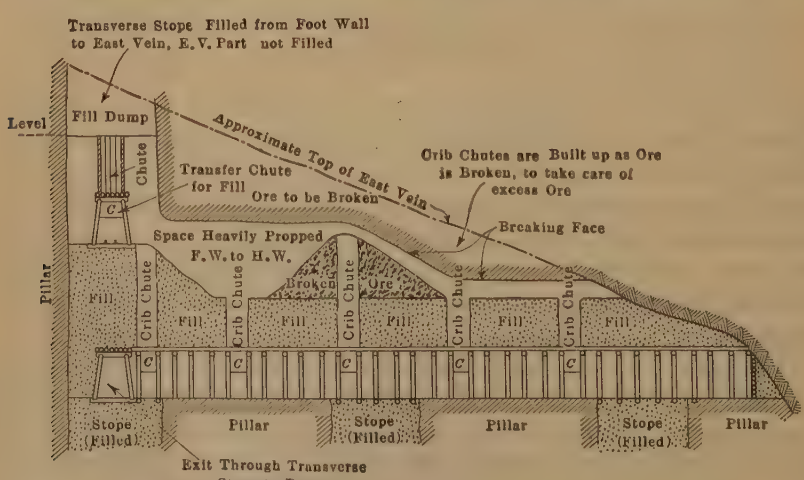


Fig. 72.—Longitudinal stope, special cut-and-fill method.

other end of the stope, the cut-and-fill method could have been used, but this was impossible without an excessive amount of rock drifts and raises (see Fig. 72).

No separate operating figures of the longitudinal stopes have been kept, but monthly figures showed about the same results as the transverse stopes. Some were better when the stopes were wide, the walls good, and the foot wall steep, others with bad walls or flat foot walls were much lower than transverse stope results. Wherever longitudinal stopes have been mined, however, ore has been obtained at a low cost which otherwise would have been left as pillars to be mined out in the future by the more expensive and slower top-slice methods.

Pillars

About 3 years ago it was possible to begin the work of mining some of the pillars between the filled stopes. This work is still in the development stage, for, even though about 20 pillars are being worked at present, a unit has to be completed as originally planned, before any other method can be started in the same pillar. A unit consists of the portion of the pillar between two levels. The conditions under which the pillars have to be mined are approximately as follows: The pillars are, as a rule, 35 ft. (10.7 m.) wide, with lengths determined by the distance from the foot wall to hanging wall, from 25 to 300 ft. (7.6 to 91 m.) or more. On each side is a filled stope, filled with mixed rock and mill-tailings; the filling abuts against the ore with no timber between to form a line of separation. In the majority of cases the tops of the pillars have been covered with timber mat and this covered with filling. Access to the pillars is possible on each level, which in most cases are 50 ft. (15 m.) apart. At the present time the pillars are against the hanging and foot walls at each end, but, in some places, parts of them have to be left for future work; this will leave these portions open on three sides.

Several methods of developing the pillars have been tried in the endeavor to find the best. These differ only in the central arrangement. As the methods of approaching the fill are the same in all cases, the differences will first be described and then the details of approaching the fill will be taken up.

Caving Methods

In the first plan tried, a drift was driven in the center line of the pillar from foot wall to hanging wall, and at intervals of 30 ft. (9 m.), inclined raises, 60° dip, were carried to the top of the pillar, as shown in Fig. 73. The raises were then stripped to 6 by 8-ft. (1.8 by 2.4-m.) size and, by stulls and lagging, divided into a 4 by 6-ft. (1.2 by 1.8-m.) ore chute, and 2½ by 6-ft. (0.76 by 1.8-m.) ladder road. Loading chutes were installed at the bottom of each ore chute.

A sublevel drift about 6 by 8 ft. in size was driven in the center of the pillar, with its bottom about 13 ft. (3.9 m.) below the top of the pillar.

Drifts were then started toward the fill at the foot wall, the hanging wall, and at points between the raises. When the fill was reached by drill holes, the faces of the drifts were left standing. Then, at the center, an opening was made to the mat and sets erected to catch up the mat. The drifts to the sides were then opened to size, and sets erected to the fill, which was breasted off and kept from running into the working place. The sets used had 10-ft. (3-m.) legs or posts and a 5-ft. (1.5-m.) cap and the posts had a batter of 1 to 12. When the level was nearly completed, a lower drift, 13 ft. below, was started for the next sublevel; this had to be started in time to be finished before the upper level was completed. The object of the many places worked in the pillar was to get a larger production, but on account of the inexperience of

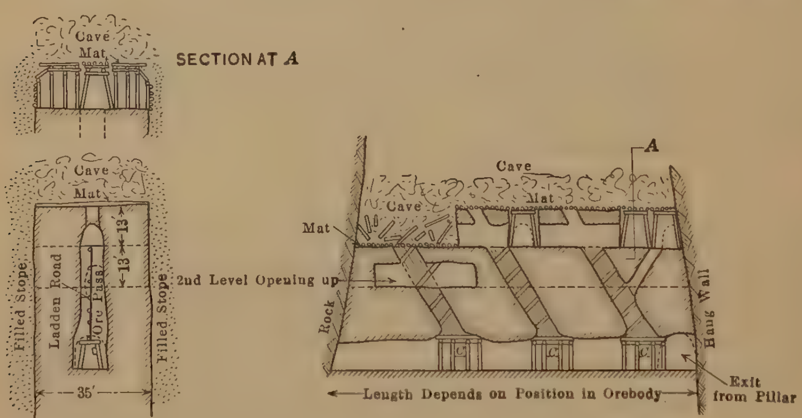


FIG. 73.—FIRST METHOD OF PILLAR MINING—CAVING.

the men, there was not enough concentration of effort, and the working parties were soon reduced to two. It also soon became evident that the height adopted, 13 ft., was not the one best suited for the work, as a stage was necessary to drill and trim the top portion, and much blocking was necessary. The long legs were also difficult to handle.

The ore at Franklin is not particularly suited to top slicing, as it is necessary to use powder for breaking, which means that all trimming for timber, etc., has to be done with "pop" holes. The excessive blasting is not the best thing for the set timbers.

Several of the pillars had not been graded level on top, so in places the 13-ft. height became 16 or 17 ft., which resulted in the necessity of double sets. These were far from satisfactory. No attempt to use square sets was made, as the amount of work was considered too small for the expenditure of time and effort to train the men. None of the workmen at the mine had ever seen any caving work, but some of the shift bosses were familiar with it.

As soon as possible, therefore, a change was made to 10 ft. as the height of a slice, and this height is still used. The set timbers are made of two legs 8 ft. (2.44 m.) long, set on a batter of 1 to 8, and the cap remains the same size, 5 ft. (1.53 m.) inside of joggles. These are all made from timber 8 to 10 in. (203.2 to 254 mm.) in diameter. With this height, the top can be reached with drill steel in machines resting on the floor, and the shorter timbers are more easily handled in the cramped quarters.

The broken ore was thrown or shoveled into the chutes, as these were close enough together to make wheelbarrows unnecessary. After the ore was taken out to a width that made caving necessary, a mat of split lagging, with full-size lagging over the chutes, was laid on the floor to cover the ore entirely, then 1¼-in. (31.75-mm.) holes were drilled halfway through each leg of the sets ready to be caved; a small quantity of

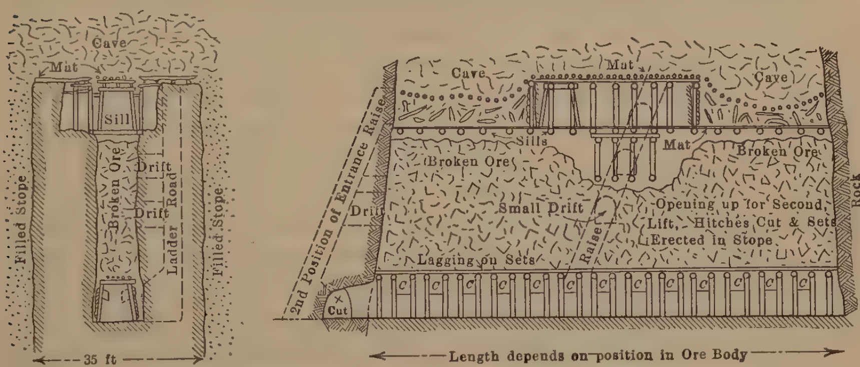


FIG. 74.—CENTER-STOPE METHOD OF PILLAR MINING—CAVING.

dynamite, with fuse and cap, was inserted and tamped in each hole, and at the end of the shift all were fired.

A second method of development, which had many good features, was tried out, but unfortunately it did not work as well as was hoped for. In this a small stope 8 to 10 ft. wide was carried up to the top of the pillar from the crosscut on the level, after the usual stope set timbers had been placed. This left two narrow pillars about 13 ft. wide, with an 8-ft. stope filled with broken ore between. In one of the small pillars at the center, a 4½ by 4-ft. raise, inclined at 70°, was driven parallel to the sides of the pillar and at the proper 10-ft. intervals this raise was connected to the stope by small crosscuts. A ladder road and a pipe line were installed in this raise, thus giving a means of access. After the small stope was completed, it was drawn down until the top of the muck pile was 13 to 15 ft. below the mat. Hitches were then cut for sill timbers so that the top of the sills were 10 ft. below the mat. The sills used were 10-in. timbers and were spaced about 4½-ft. centers. Sets were erected

on these sills and blocked to the mat. At each end of the pillar small drifts were run to the fill as before, opened up and timbered, and the ore mined back towards the entrance raise (see Fig. 74).

The advantages claimed for this method were:

1. Cheap breaking of the ore in the small stope.
2. Much of the ore broken, when mining toward the fill, rolled directly into the stope, saving handling.
3. All the ore that had to be handled needed but a straight throw to get into the stope reservoir.
4. The broken ore in the stope acted as a support to the pillars on each side, the same as if filled.
5. The control of the ore in the stope was obtained through the numerous chutes in the stope sets on the level.

Unfortunately, the bases of support over the stope failed, as the sills were continually breaking, even though 12-in. timber was substituted for 10-in., and angle bracing was resorted to. When a sill broke another had to be placed beside it, and another set erected with shorter legs, thus reducing the head room. Although the stopes were to be kept as nearly as possible to an 8-ft. width, this could not be carried out in practice on account of slips and cracks in the pillars. Often too, after the sills and sets were in place, the further work would cause the bottom of the hitches to slip off and make additional sets necessary. In several cases the head room was reduced to 4 ft. (1.2 m.), with sets so close together that there was hardly room to throw the muck between them. In one case an entire side pillar settled 2 ft., distorted the sets and ruined the ladder-road. It became necessary to stop pulling through the chutes so that the sills could be placed on the muck in the stopes, a new chute and ladder-road had to be constructed, and a small car built to carry the muck to that chute.

The numerous hitches that had to be cut formed another drawback in practice. To gain sufficient bearing for the timber, they had to be at least 8 in. deep, with room enough at one side to swing in the timber. These were costly, even though drilled and blasted before trimming to size with pneumatic hitch cutters. The supports for the angle braces also consumed time and labor. Then, too, the stopes were not wide enough for the broken ore to be drawn down evenly, so the ore would hang up and then go down with a surge, sometimes with disastrous results to the set timbers below.

As originally tried, the entrance road was placed in the center of the pillar, work was started at each end and retreated toward the center; so the final work on a sublevel, therefore, was in a small pillar with fill on both sides and caved ground on the top and ends, hence a great weight had to be supported, and the removal of this last portion of the

ore was quite costly. To avoid this, a new raise was carried up at one end, but outside of the working area of the pillar, either in ore or rock, and in the center line of the pillar. This was connected to the stope by small crosscuts. The method of working was then changed so as to start mining at the end of the pillar farthest from this raise, and to retreat from that point. This always left solid ground at one end of the working place, and has done away with the excessive weight.

Some pillars that are known to be more or less shattered have been developed on a straight top-slicing basis. The entrance raise, as shown in Fig. 75, is carried up in the center line of the pillar but outside the working limits. Raises, inclined 70° to the horizontal, and spaced 20 to 30 ft. (6 to 9 m.) apart, are driven for chutes, and sublevel drifts 4 by 6 ft. (1.2 by 1.8 m.) in size driven from the entrance raise to the opposite end of the pillar. This work is all completed before the slicing work is

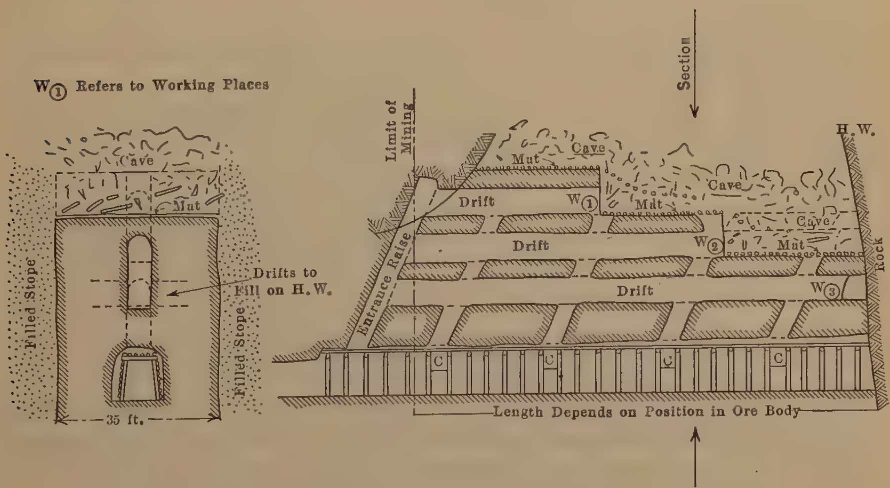


FIG. 75.—SUB-LEVEL METHOD OF PILLAR MINING—CAVING.

started; also the crosscut on the level is timbered with stope sets from the entrance raise to the far end, but chutes are placed only at the raises. Originally the crosscuts on the main levels were timbered only at the raises, but all had to be timbered throughout, in the end, so now it is all done at once, and much more cheaply. The sublevel drifts are made narrow, so that the sets placed on a level above will always be on the solid portion of the pillar. Although this method of placing drifts and raises is more expensive than a very narrow stope, 4 ft. wide, it is better, for the raises permit a better drawing of the ore than the stope would, while in addition there is always the possibility of the stope getting too wide. When development is completed, the slicing work is begun; as soon as the top sublevel has been worked back about 30 ft. from the end, a

second party is started on the next lower one. Three and even four working parties may be engaged at once on a pillar if it is long enough.

The idea of the stope in the center of the pillar, with its cheaper ore, still seems good, so that another method has been developed to retain this feature, but to eliminate the drawbacks of the original stope scheme. In this latest scheme, illustrated in Fig. 76, the sets on the level are spaced on 4-ft. 3-in. (1.37-m.) centers, with chutes placed at 17-ft. (5.2-m.) centers, and alternating from side to side. When the sets on the level are completed the stope is carried up 8½ to 9 ft. (5.64 to 5.79 m.) wide to a point about 15 ft. (4.6 m.) above the top of the sets, and the ore is all drawn out, except for the amount that is behind the sets. Sills are then placed on the lagging above the sets, and immediately over the sets below, and these sills are cut as long as possible and wedged tightly to the sides. Regular slice sets are then placed on these sills, blocked

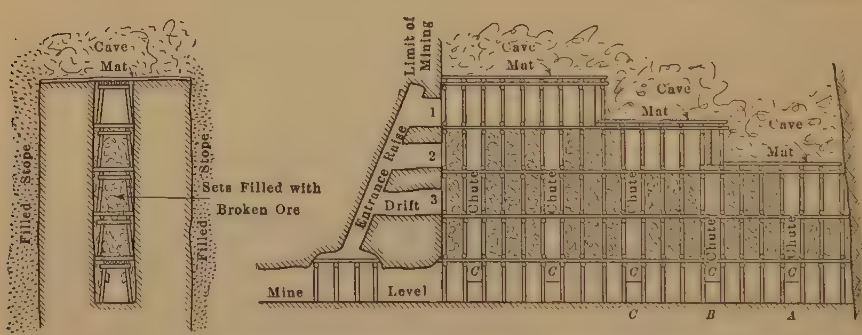


FIG. 76.—SETS-ON-SETS METHOD OF PILLAR MINING—CAVING.

and studdled, but are covered with lagging spaced 12 in. apart. The spaces over the chutes are planked up to make chutes. The stope is then carried up another lift. The broken ore, however, is drawn between the lagging to fill the space between the sets and the excess is drawn through the chutes. The lagging on top of the sets is then removed, except for the pieces on each side immediately over the legs, and sills are again placed with sets on them, and the process repeated the same way, until the mat is reached. The top sets are blocked to the mat, and are left empty. A floor is laid on the sills for better shoveling and the pillar is ready for slicing.

The sets that have been placed are all necessary to mine the pillar; each set has a foundation under it which ultimately reaches to solid bench on the level, and all the spaces not needed at once are filled with broken ore. The latter remains in position until wanted; it is not a moving mass, as though it were drawn off at the bottom and replaced with the new at the top, and therefore is a real support. When the top level is completed, the plank sides of the chutes of the next lift are

knocked out. A great deal of the broken ore will immediately run into the chute, and that below the angle of repose is then mucked into the chute, after which the next level is ready for slicing. In actual practice it is not necessary to wait for one sublevel to be entirely completed before starting on the next lower one, for when the first cave has been made, about 30 ft. (9 m.) wide, work can generally be started below, so that pillars developed by this method can be worked in stages, too.

After the sublevel is established in whatever method is used, drifts 6 ft. (1.8 m.) high and 7 ft. (2.2 m.) wide are started at one end of the pillar toward the filled stopes on each side. These drifts are not broken to the fill, for if a drill steel breaks through, only the cut is fired and the rest is left standing for the time. Starting from the center the drifts are then opened up large enough for the timber sets, a set at a time, and the sets, spaced on 3 to 4-ft. centers, are put in place and blocked to the mat. The set timbers have 5-ft. caps, and 8-ft. legs, cut to a batter of 1½ to 12, and are made from sticks 8 in., 9 in., or 10 in. in diameter. The firing has to be very lightly and carefully done, with only a few holes at a time, so that any timbers, mainly braces, that may be knocked out, can be replaced before more holes are fired.

A drift is not driven for the separate lines of sets, only for the first at the end of the pillar, for this opening leaves the remainder of the slices with two faces to which to break, thus permitting faster progress than it would if the drift had to be driven and opened up.

The machines used in the pillar work are the same as for the stopes and drifts. The drifts are drilled with the mounted block-hole machines, which are also used dismounted for block-holing and drilling bottom holes. Most of the other holes are drilled with the stoping machines, though at times the block-holer is mounted and worked from a column. No water is used in any pillar drilling, except for development work, and the pneumatic feed mounting is strongly favored.

No regular system of holes is in use, as advantage is taken of all chances possible, though in general the holes are drilled toward the fill, and approximately parallel to the face left when the ground was opened for the previous line of sets. Not much trouble is encountered in the work until it becomes necessary to breast against the fill. It is important, of course, that all braces be kept in place. The breasting of the fill is the slowest part of the work, for care has to be taken that a "run" is not started, for if such a misfortune happens considerable extra work is involved in stopping it and disposing of the rock.

When the sets have been carried as close to the fill as is deemed safe, an opening to the fill is made at the bottom of the face so that the floor mat timbers may be shoved to the fill, and thus insure that the pillar is properly covered for the next lower slice. Then a short set, shown in Fig. 77, 5 to 6 ft. (1.5 to 1.8 m.) high, is put in, and lagging, spaced

about 8 in. apart, is laid on this set, and the one behind, also from this set to the floor. Holes are then drilled in the ground ahead, fired, and the broken ore then barred between the lagging, generally working from the bottom up, allowing the fill to run against the lagging. In this way practically all the ore is obtained. Various other schemes of breasting to the fill have been tried, but this method works best, for even after the ground between this short set and the fill is blasted, it practically remains in place until taken out with the bars, thus preventing any sudden run of filling. Very little ore is lost in the slices, and the greatest portion of the ore that is lost is due to small spalls being mixed with the filling or flying into the caves when holes are fired.

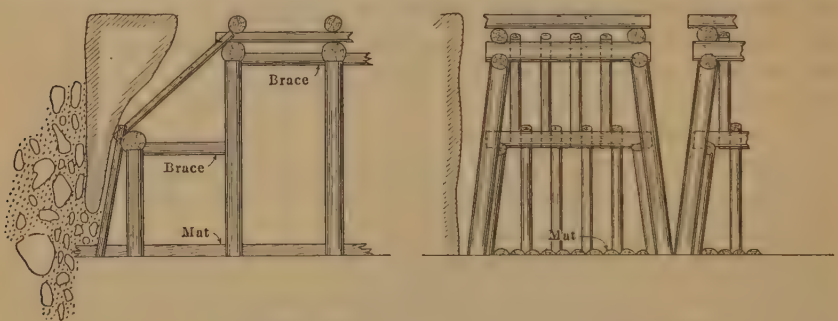


FIG. 77.—BREASTING AGAINST FILL.

Non-Caving Methods

Several other methods of extracting pillars, when the conditions differed from those ordinarily encountered, have been used, but they were intended to get better results by taking advantage of the special conditions and were not expected to become permanent systems. Short descriptions might be of interest.

Fig. 78 shows two stopes, with centers 52 ft. (15.85 m.) apart, which had been carried to the top of the East Vein. Stope "A" was filled from above to the angle of repose of the fill as shown, but stope "B" was filled only to the level. The pillar between these stopes was then divided by a small stope 10 ft. (3 m.) wide which, when completed to the cap rock, was drawn completely empty. Timber bulkheads built with all timbers touching (skin to skin), 9 ft. (2.75 m.) square outside dimensions were then built on the set timbers, and also on sets placed on the fill in stope "B." The bulkheads were about 5 ft. (1.5 m.) apart and extended up to the cap rock.

The pillar between stope "A" and the bulkheads in the small stope was taken out first by undercutting the pillar and pulling the fill in to replace the extracted ore. It was necessary to shovel the ore into cars until the fill had been built up to the level of the chutes. As the fill was

built up above the set timbers, a brattice was laid against the bulkheads and the space between them used for chutes and passage. Ultimately a point was reached where no more fill was encountered and from there

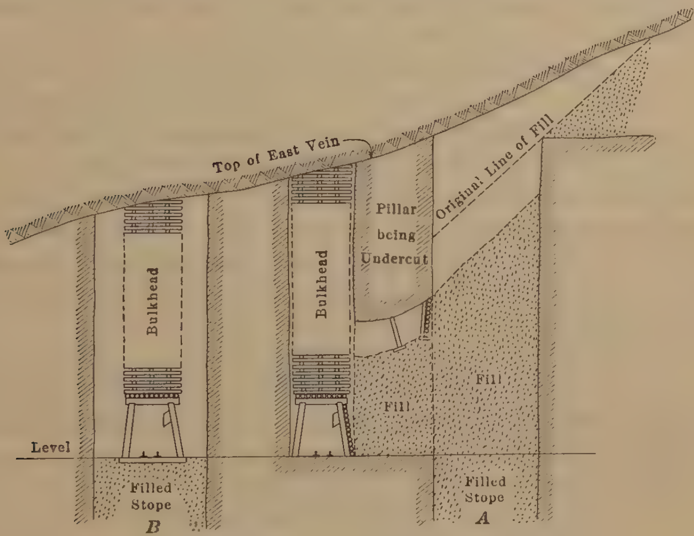


FIG. 78.—METHOD OF UNDERCUTTING PILLAR.

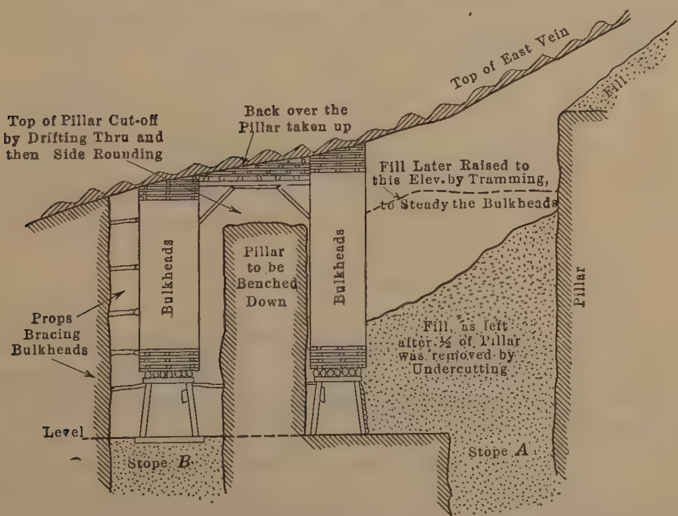


FIG. 79.—METHOD OF BENCHING PILLAR UNDER SUPPORTED ROOF.

on it became a matter of breaking and shoveling, as the cap rock was self-supporting and gave no trouble.

In this work the fill had to be breasted off with lagging held in place by props until the broken ore was cleaned up, but this did not prove difficult as only a small portion of the fill was exposed at a time.

A different method was used to extract the pillar adjoining stope "B." The conditions at the start are shown in Fig. 79. The first step was to drift through the pillar, near the top, and after that to extract all the ore between the level of this drift and the cap-rock. The back was then taken up as in a stope, using the bulkheads for the support of the timber, and the pillar was then benched down, with the ore passing through chutes in the set timbers on each side. As soon as there was room, the timbers holding the back were supported by angle braces.

The major part of this work was completed when that part of the pillar which was not to be mined at the time began to crush and the cap rock broke off in large slabs upon the back timbers. It became necessary to cover the remainder of the pillar with a mat, and to fill the opening with waste. In spite of the weight and movement, the timbers held until the filling was completed.

General Remarks on Pillar Mining

It has been found best to cave a top slice after a width of three sets, about 22 to 25 ft. (6.7 to 7.6 m.) has been opened. The timbers may be

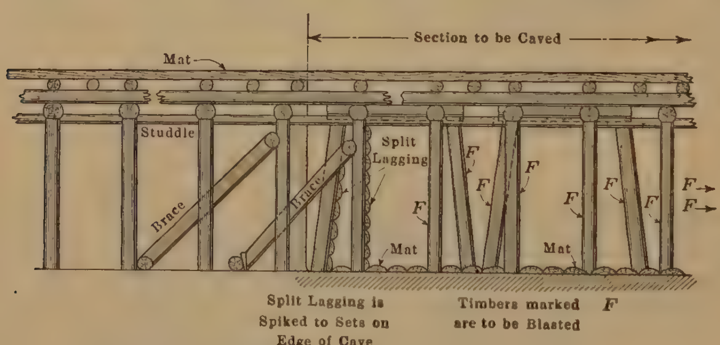


FIG. 80.—TIMBER BRACING FOR CAVING SETS.

all in good shape but generally, before a fourth set can be completed, the weight begins to come on the timbers, and handicaps progress. In places, under especially favorable conditions, more than three can be taken, but not as a rule. When caving a "room" the legs that are against the solid are not blasted, though the other legs of these sets are. Split lagging is spiked to the legs that are not to be blasted and the two or three sets in the center of the pillar next to the room to be caved are carefully braced by inclined braces. This has been found necessary to prevent the center sets being knocked out of position or at the least out of plumb (see Fig. 80).

Timber is brought on small timber trucks to the entrance of the pillars and is hoisted to the working sublevel by small column hoists using ¼-in. (6.35-mm.) wire rope. The hoists are mounted on the level and the

required direction of the rope is obtained by sheaves. A column is generally set up at each pillar, but a single hoist is sufficient at present for each level, as it is easily transported from pillar to pillar. Formerly the timbers were slid up the ladders, resting on straps of iron which fitted the sides of the ladders, but the present practice is to build small slides of plank, about 18 in. (0.46 m.) wide, with sides about 6 in. (152.4 mm.) high, at one side of the raise.

This pillar mining is still in the development stage, but more and more ore is being produced from pillars each month. The men have all been trained at Franklin, but the condition of the labor market has been a great handicap to the training. Table 11 shows some of the results obtained by this work, and while the stoping methods produce the ore much more cheaply, the results from the pillars are gradually becoming better. It is not expected that the pillar ore ever can be obtained as cheaply as that from the stopes, but practically all the ore in the pillars is recovered and the loss is considered less than 1 per cent. It is possible that much better figures for the work could be obtained at the sacrifice of ore, but the value of the ore is such that it is cheaper in the end to get it all, even though at a higher cost per ton recovered. Table 11 gives some illuminating cost figures.

TABLE 11.—*Top Slicing Figures*

Tonnage Broken	Tons Broken per Man in Slice	Tons Broken per Drill-Shift	Tons Broken per Foot of Hole	Explosives, Cost per Ton	Timber Charges	Timber, Cost per Ton
1,790	12.1	42.4	0.40	$0.116	$35.00	$0.020
1,110	8.6	46.2	0.46	0.104	25.78	0.025
1,110	5.5	34.6	0.40	0.040	55.00	0.050
830	17.3	63.8	0.70	0.060	16.24	0.020
1,140	5.2	39.6	0.50	0.040	121.33	0.121
840	5.3	40.0	0.40	0.080	115.10	0.136
1,010	8.1	56.7	0.40	0.050	50.00	0.050
890	5.8	34.2	0.40	0.070	61.38	0.069
1,150	8.8	65.8	0.80	0.059	76.27	0.066
1,310	7.2	33.6	0.33	0.071	190.69	0.146
1,170	7.8	50.0	0.60	0.062	190.14	0.163
870	7.5	33.4	0.38	0.105	210.71	0.242
Average for year 1916:						
95,095	6.1	36.9	0.40	0.089	21,871.65	0.230*

These are monthly averages of individual working places and are somewhat above the average. They are chosen to show what has been accomplished at times and what is hoped will soon be regularly exceeded.

For 1916 the tons per man from solid ore in pillar, to broken ore in loading pocket for pillar work were 3.1 tons.

* Includes development timbering.

Open-cut Mining

Some of the earliest mining in the history of the orebody was done in the open cut at its south end. From one of the early pits sunk there, the ore was mined that was used in producing the original brass weights and measures used as standards by the U. S. Bureau of Standards. In the early operations, a short tunnel was driven and the open-cut ore was transported through it by mule haulage until the pit reached a depth of about 40 ft. (12 m.), then ore was hoisted up small shafts, removed by cableway systems, also hoisted up inclines. The present method is to drop it through mill holes and raises to the 300-ft. level, where it is loaded into haulage cars and carried by electric haulage to the delivery point into the main raise series at Palmer shaft.

In order to mine the ore, it has been necessary to strip a considerable tonnage of rock both from the hanging wall and the included wedge rock. This has not been an excessive cost, however, as the rock was needed for filling the stopes and was milled to raises delivering to the 300 level electric haulage system for distribution.

The south open cut is now very far advanced and measures approximately 600 ft. (182 m.) long by 350 ft. (106 m.) wide and 250 ft. (76 m.) maximum depth. The accompanying photograph, Fig. 81, gives only an imperfect idea of its size and is taken looking from the south toward the north end of the open cut.

Table 12.—*Nationalities of Employees at Franklin*

Nationality	Mine, Per Cent.	Mill, Per Cent.	Salaried, Per Cent.	Total, Per Cent.
Russians	21.0	2.0	....	23.0
Hungarians	12.7	14.1	0.3	27.1
Americans	2.3	19.6	4.2	26.1
English	0.9	0.6	0.7	2.2
Polish	4.4	0.4	0.1	4.9
Slavish	10.1	1.5	0.2	11.8
Germans	0.2	....	0.2	0.4
Lithuanians	0.5	0.2	....	0.7
Romans	0.1	....	....	0.1
Ruthenians, Ukranians	0.1	....	....	0.1
Italians	....	2.2	0.1	2.3
Scotch	....	0.2	0.1	0.3
Spaniards	....	0.6	....	0.6
Irish	....	0.1	....	0.1
Welsh...., Norwegian	....	0.1	....	0.1
Canadians	....	....	0.1	0.1
Total	52.3	41.6	6.0	99.9

FIG. 81.—SOUTH OPEN CUT.

GENERAL DISCUSSION

The labor on the plant varies from time to time, but now totals about 2003 men, whose nationalities are about in accordance with the tabulations of per cent. in Table 12. Many men have been at the mine for several years, and are quite expert in their work. The usual number of floating laborers has to be contended with; they are much less desirable than plain "greenhorns" who are willing to learn. Since the war started, no "greenhorns" have come over. In spite of the nationalities of the men, no trouble has occurred between them. The shift bosses, and the higher class of skilled labor, are mainly American, English or Cornish.

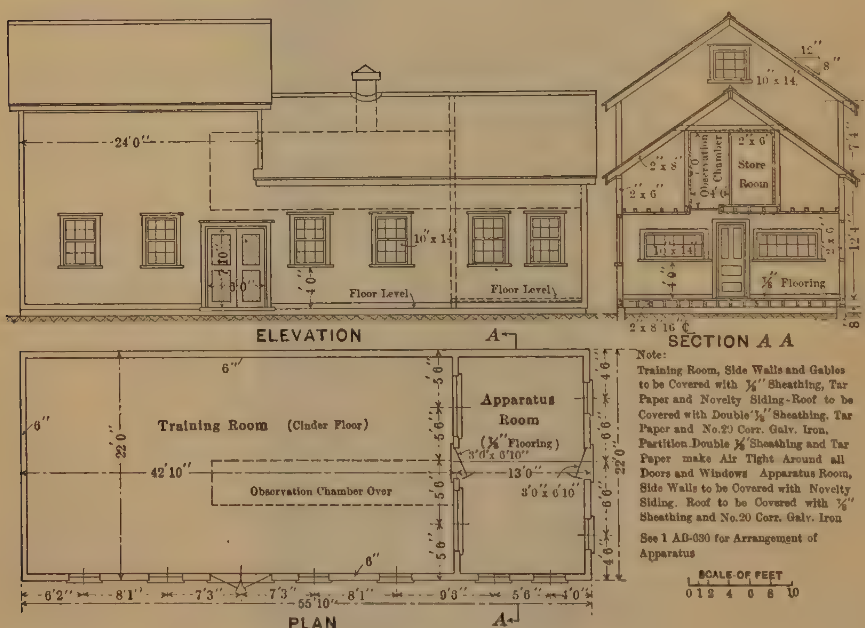


FIG. 82.—FIRST-AID BUILDING.

The company has established a social service department, which has done much for the town, especially for the foreigners. For instance, 5 years ago, 50 per cent. of the lockers in the change house were uncalled for and only 40 per cent. of the men on the pay roll used them, but now even with the equipment doubled, there are not enough lockers to fill the demand. The Community House or Neighborhood House is the center of this work, with four bowling alleys, card and game tables, a pool table, reading room, and library. A competent corps of workers lives at the house and directs the activities. These are kindergarten, boys' and girls' clubs, English classes for foreigners, and night school. A trained nurse

makes regular visits in the town, and by instruction and suggestion has accomplished a great work in Americanizing the habits, if not the actions, of the foreign population. A smaller Neighborhood House is also maintained for the exclusive use of the foreigners who are backward about enjoying the privileges of the large house. This building is heated by hot air and has a drinking fountain, a game room with pool table, library and reading room, and a band room in the basement.

A well-equipped hospital and a competent staff look after all the medical work, but many of the workmen have been trained in first aid, both by the instruction from the Bureau of Mines, and by the safety engineer. In the plant, near the shaft, a first-aid and rescue building

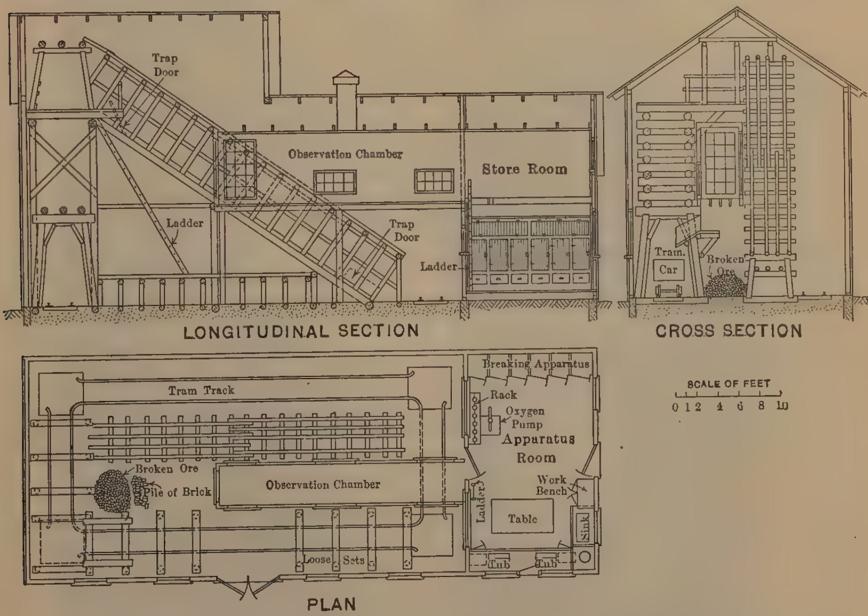


FIG. 83.—ARRANGEMENT OF APPARATUS IN FIRST-AID BUILDING.

has been erected for training and practice in helmet work, under smoky and gaseous conditions. Many mining conditions have been simulated by timber sets, inclines, a cribbed raise, a chute, a tram car, and passageways with low head room, etc., in one portion which can be filled with smoke. Ample provision for observation has been made, and also provision for the quick release of ventilators and doors in case of accident. This building with equipment, etc., and the safety work carried on has been fully described by Mr. Tillson[3], so will not be duplicated here (see Figs. 82 and 83). One result of safety discussion and observation is the substitution of stairways for travel between levels, in place of

[3]*Engineering & Mining Journal* (Apr. 7, 1917).

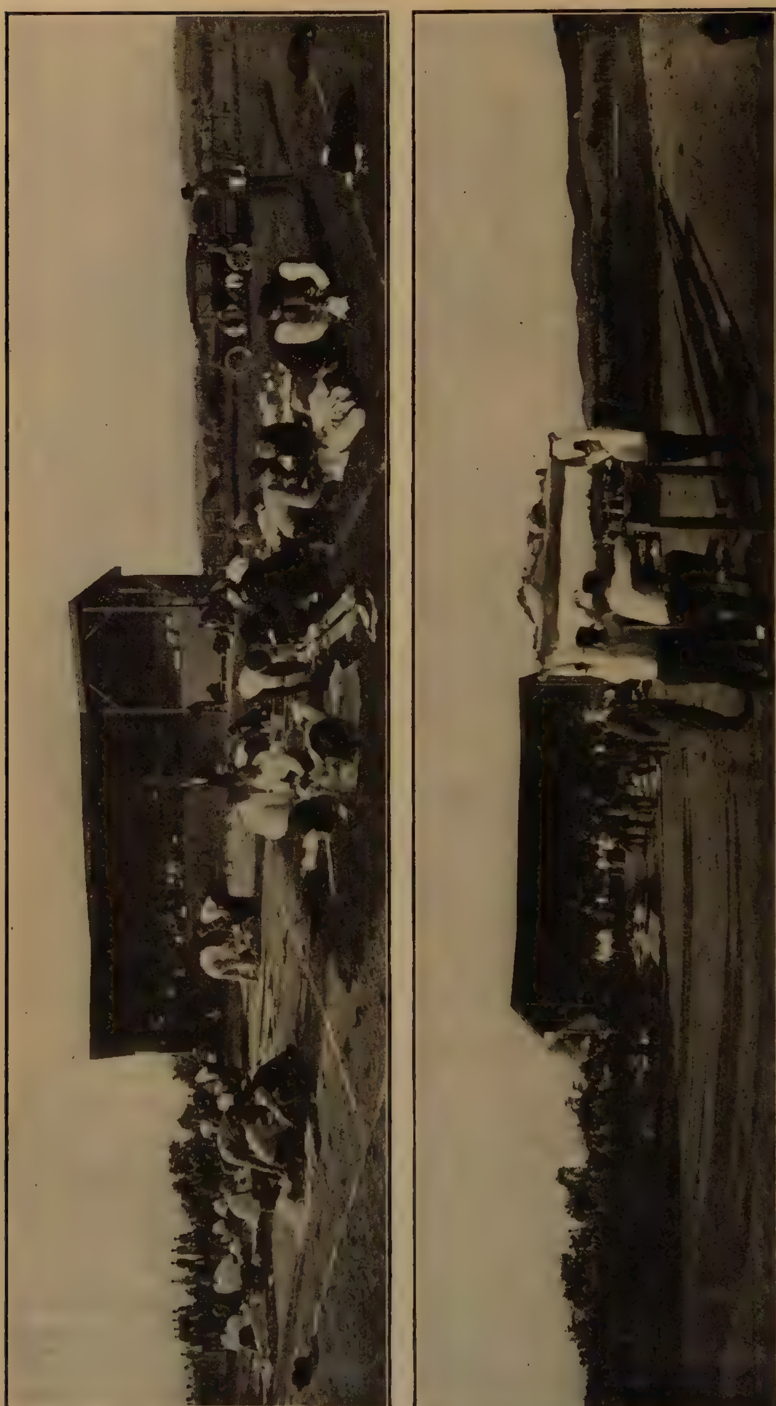


Fig. 84.—Views of First-Aid Contest at Franklin.

ladders. These are made of 2-in. plank with hand rails and so constructed that stretchers may be slid on them without jar to the patient. Their superiority to ladders is apparent after one trial.

Over 200 of the mine workmen have received U. S. Bureau of Mines instruction in first aid and 80 have also been practiced in the use of oxygen helmets, and other rescue apparatus, in smoke and non-respirable gases. In October, 1916, a contest between ten teams of six men was held to choose the team that would be entered in the National Interstate First-Aid Contest held in Detroit the same month. The two photographs, Fig. 84, give an idea of interest displayed by the men.

The schedule of training in oxygen-helmet work is substantially as follows for all men on the mine rescue squads:

First Day.—Physical examination.

Instruction in construction, assembling, charging, and testing of apparatus.

Wear apparatus 1 hr. without oxygen, in fresh air in smokeroom.

Work schedule—walk, climb, and crawl around smokeroom (average rate of walking, 3 miles per hour).

Carry five bags of sand (50 lb.) to top platform, and return same to place from which taken.

Instructions in dismantling and cleaning apparatus and charging bottles.

Second Day.—Supplementary instructions in assembling, charging, testing, and wearing apparatus.

Wear apparatus 1 hr. with oxygen, in fresh air in smokeroom.

Work schedule—same as first day.

Supplementary instruction in dismantling and cleaning apparatus and charging bottles.

Third Day.—Assemble, charge, and test apparatus.

Wear apparatus 2 hr. in smoke in smokeroom.

Work schedule—walk and climb around smokeroom at moderate pace for about 10 min. Take down and set up four timber sets. Connect line of pipe as instructed by captain.

Finish period by alternately carrying bag of sand up inclined manway and down ladder, and walking around smokeroom.

Dismantle and clean apparatus and charge bottles.

Fourth Day.—Assemble, charge, and test apparatus.

Wear apparatus 2 hr. in smoke in smokeroom.

Work schedule—walk and climb around smokeroom at moderate pace for about 10 min. Build wall on south side of smokeroom across track, made of sand bags backed by stone. Tram stone in car from chute.

Carry sand bags up inclined manway and down ladder.

Dismantle line of pipe.

Finish period by walking and climbing around smokeroom, sawing timber, etc.

Fifth Day.—Assemble, charge and test apparatus.

Wear apparatus 2 hr. in smoke in smokeroom.

Work schedule—walk and climb around smokeroom for about 10 min.

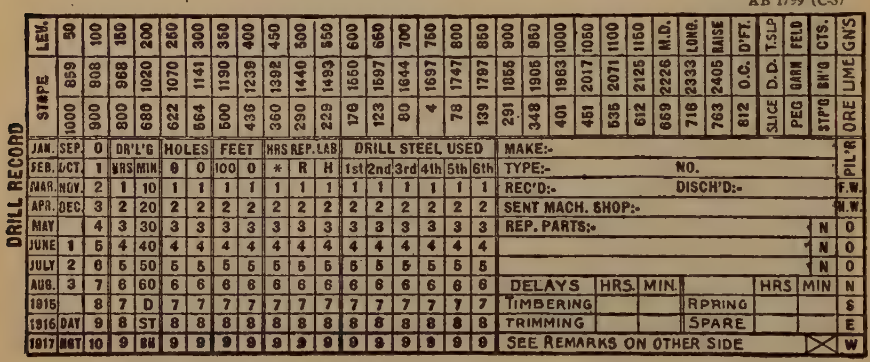
Tear down wall and return materials to place, carrying sand bags up inclined manway and down cribbed chute.

Tear up and relay track.

Take down and set up two timber sets.

Finish period by walking and climbing around smokeroom, sawing timber, etc.

The engineering department makes an estimate each month of the amount of work accomplished in each working place, all surveys are by


AB 1799 (C-5)

DRILL RECORD

LEV.	50	100	150	200	250	300	350	400	450	500	550	600	650	700	750	800	850	900	950	1000	1050	1100	1150	M.D.	LONG.	RAISE	D'FT.	T.SLP	FELD	CTS.	GN'S
STOPE	859	808	968	1020	1070	1141	1190	1239	1392	1440	1493	1550	1597	1644	1697	1747	1797	1855	1905	1963	2017	2071	2125	2226	2333	2405	O.C.	D.D.	GARM	BH'G	LIME
	1000	900	800	680	622	564	500	436	360	290	229	178	123	80	4	78	139	291	348	401	451	535	612	669	716	763	812	SLICE	PEG	STP'G	ORE

JAN.	SEP.	0	DR'L'G		HOLES		FEET		HRS REP. LAB			DRILL STEEL USED						MAKE:-		PIL'R
FEB.	OCT.	1	HRS	MIN	0	0	100	0	*	R	H	1st	2nd	3rd	4th	5th	6th	TYPE:- NO.		
MAR.	NOV.	2	1	10	1	1	1	1	1	1	1	1	1	1	1	1	1	REC'D:- DISCH'D:-		F.W.
APR.	DEC.	3	2	20	2	2	2	2	2	2	2	2	2	2	2	2	2	SENT MACH. SHOP:-		N.W.
MAY		4	3	30	3	3	3	3	3	3	3	3	3	3	3	3	3	REP. PARTS:-	N	O
JUNE	1	5	4	40	4	4	4	4	4	4	4	4	4	4	4	4	4		N	O
JULY	2	6	5	50	5	5	5	5	5	5	5	5	5	5	5	5	5		N	O
AUG.	3	7	6	60	6	6	6	6	6	6	6	6	6	6	6	6	6	DELAYS HRS. MIN. HRS MIN		N
1915		8	7	D	7	7	7	7	7	7	7	7	7	7	7	7	7	TIMBERING RPRING		S
1916	DAY	9	8	ST	8	8	8	8	8	8	8	8	8	8	8	8	8	TRIMMING SPARE		E
1917	NGT	10	9	SH	9	9	9	9	9	9	9	9	9	9	9	9	9	SEE REMARKS ON OTHER SIDE		W

Fig. 85.—Drill record.

instruments, with results that are in the main quite close to scale weights of mill products and to approximate amounts as gaged from car counts, or labor, and any small errors are naturally compensated in the course of time. Against these estimates, the mine department's clerical force enters all the labor and material charges so that unit figures for each working place are obtained, which give a basis for comparison with similar places, or with the same place for previous months.

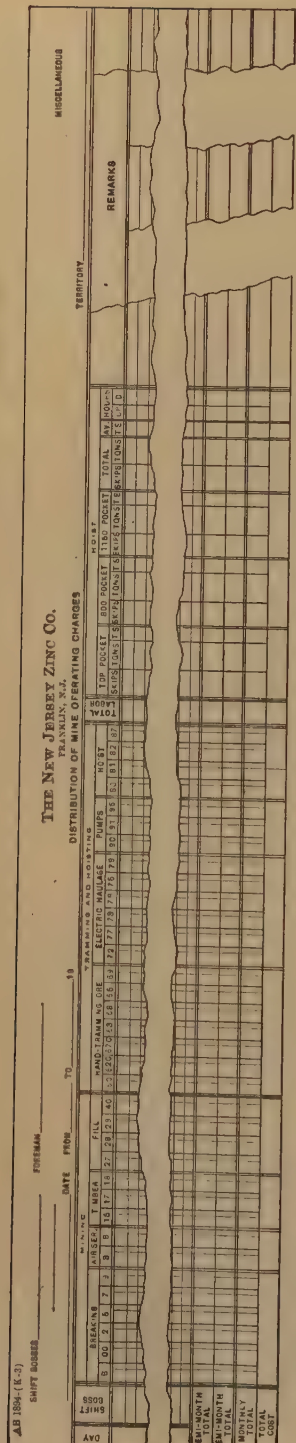
Method of Recording Mine Operating Data

Because, in general, costs are not of great value for comparisons unless they are based upon the same unit costs of labor and supplies, and because the operating staff is not responsible for the wages per hour, emphasis has been placed upon the amount of work accomplished as rated per hour of labor or per unit of bulk or weight of supplies. Furthermore, an analysis showing the rate of production for each working place in the mine permits a more intelligent supervision of the labor

MINE OPERATING AB 1892 (C-8)

BRK. FILL				PDR		FUSE		CAPS		EXP.		FILL					CARS		PDR		FUSE		CAPS		E.H.FILL			CARS		L'G'N		PL'K		SL'BS		DATE			STOPE			LEV.
20	21	24	26	100	200	100	200	100	200	100	200	TRAM		H'D'L.			100	200	100	200	100	200	100	200				100	200	TOP		800		1150		J	Nv	1		56	1239	Sur
1	1	1	1	1	300	1	300	1	300	1	300	41	46	42	44	47	1	1	1	1	1	1	1	1	51	54	56	1	1	1	100	1	100	1	100	F	Dc	2	960	4	1392	100
2	2	2	2	2	400	2	400	2	400	2	400	1	1	1	1	1	2	2	2	2	2	2	2	2	1	1	1	2	2	2	200	2	200	2	200	M		3	920	78	1440	150
3	3	3	3	3	500	3	500	3	500	3	500	2	2	2	2	2	3	3	3	3	3	3	3	3	2	2	2	3	3	3	300	3	300	3	300	A	10	4	880	139	1493	200
4	4	4	4	4		4		4		4		3	3	3	3	3	4	4	4	4	4	4	4	4	3	3	3	4	4	4	400	4	400	4	400	My	20	5	840	291	1550	250
5	5	5	5	5	0	5	0	5	0	5	0	4	4	4	4	4	5	5	5	5	5	5	5	5	4	4	4	5	5	5	1	5	1	5	1	J	30	6	800	348	1597	300
6	6	6	6	6	2	6	2	6	2	6	2	5	5	5	5	5	6	6	6	6	6	6	6	6	5	5	5	6	6	6	2	6	2	6	2	Jy		7	740	401	1644	350
7	7	7	7	7	4	7	4	7	4	7	4	6	6	6	6	6	7	7	7	7	7	7	7	7	6	6	6	7	7	7	3	7	3	7	3	Ag	D	8	680	451	1697	400
8	8	8	8	8	6	8	6	8	6	8	6	7	7	7	7	7	8	8	8	8	8	8	8	8	7	7	7	8	8	8	4	8	4	8	4	S	A	9	622	535	1747	450
30	31	34	36	9	8	9	8	9	8	9	8	8	8	8	8	8	9	9	9	9	9	9	9	9	8	8	8	9	9	9	5	9	5	9	5	O	N		564	612	1797	500
ORE				PDR		FUSE		CAPS		EXP.		ORE					CARS		PDR		FUSE		CAPS		E.H. ORE			CARS		TIMBER			12 IN		10 IN		8 IN		500	669	1855	550
BREAKING				100	200	100	200	100	200	100	200	TRAM		H'D'L			100	200	100	200	100	200	100	200				100	200	LABOR			100	200	100	200	100	200	460	716	1905	600
0	1	4	6	1	1	1	1	1	1	1	1	61	66	63	64	67	1	1	1	1	1	1	1	1	71	74	76	1	1	10	11	16	1	1	1	1	1	1	436	741	1963	650
1	1	1	1	2	2	2	2	2	2	2	2	1	1	1	1	1	2	2	2	2	2	2	2	2	1	1	1	2	2	1	1	1	2	2	2	2	2	2	360	763	2017	700
2	2	2	2	3	3	3	3	3	3	3	3	2	2	2	2	2	3	3	3	3	3	3	3	3	2	2	2	3	3	2	2	2	3	3	3	3	3	3	290	812	2071	750
3	3	3	3	4	4	4	4	4	4	4	4	3	3	3	3	3	4	4	4	4	4	4	4	4	3	3	3	4	4	3	3	3	4	4	4	4	4	4	229	859	2125	800
4	4	4	4	5	5	5	5	5	5	5	5	4	4	4	4	4	5	5	5	5	5	5	5	5	4	4	4	5	5	4	4	4	5	5	5	5	5	5	202	908	2226	850
5	5	5	5	6	6	6	6	6	6	6	6	5	5	5	5	5	6	6	6	6	6	6	6	6	5	5	5	6	6	5	5	5	6	6	6	6	6	6	176	968	2333	900
6	6	6	6	7	7	7	7	7	7	7	7	6	6	6	6	6	7	7	7	7	7	7	7	7	6	6	6	7	7	6	6	6	7	7	7	7	7	7	150	1020	2405	950
7	7	7	7	8	8	8	8	8	8	8	8	7	7	7	7	7	8	8	8	8	8	8	8	8	7	7	7	8	8	7	7	7	8	8	8	8	8	8	123	1070	O.C.	1000
8	8	8	8	9	9	9	9	9	9	9	9	8	8	8	8	8	9	9	9	9	9	9	9	9	8	8	8	9	9	8	8	8	9	9	9	9	9	9	80	1141	TSlp	1050
MIS	B	00	2	3	5	7	8	9	27	28	29	40	60	C	63	C	65	68	69	90	91	96	72	73	75	77	78	79		15	17	18			80	81	82	87	Plr.	1190	D.D.	1100
	TIMB.		BRK.		FILL		HDL		M.D.		D.W.	TRAM			CHTS		SAF'Y		MUCK		PUMP		STRIP		CL'N		GRA		SLICE		RAISE		DRIFT		HOIST		L'G'L		N S E W H F			1150

FIG. 86.—MINE-OPERATING SLIP.



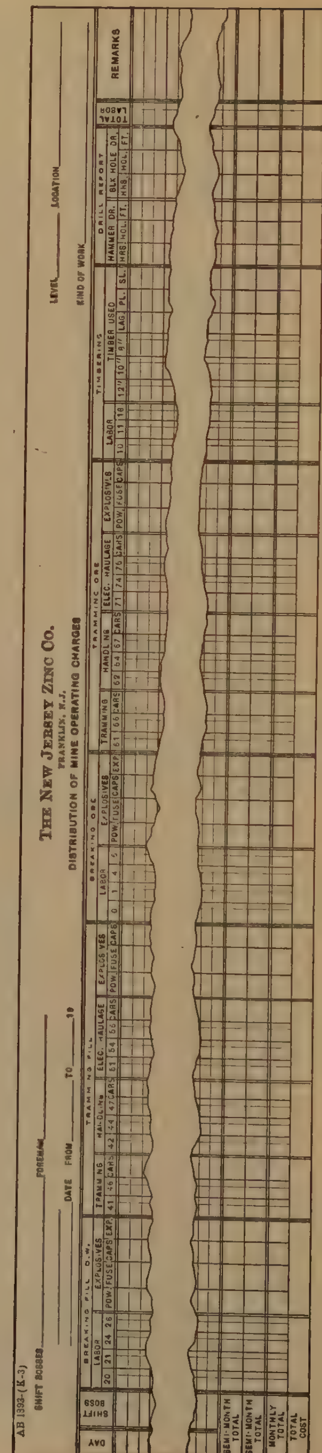


FIG. 87.—MINE OPERATING ANALYSIS SHEETS.

and greater efficiencies in operation. For this purpose a distribution of labor, explosives, timber and the amount of drilling is recorded daily by the clerical force in the mine captain's office and at the end of the month are summarized for each working place and entered on a sheet which also shows the surveyors' estimates of the results produced (ore or rock broken from the solid state or change in the amount of any reserve of ore or filling placed, and footage and tonnage advance of drifts, raises and stopes).

The basis of the mine office distribution is the daily reports of the shift bosses, which are made on separate slips for each place where they have men working (see Fig. 86). These slips are furnished bound in a book like a conductor's transfers and the information is supplied by punching out the figures representing the report, such as, date, coördinate of the stope, the level, and under the different job numbers (which are printed in red at the heads of the columns) are the numbers of shifts and hours (the latter indicated by punching a small round hole) of labor worked on the job; also, the amount of timber used, number of cars trammed, and explosives supplied, are indicated, as well as the nature of any special work such as "cleaning," "grading," "safety," etc. Another similar punch book of slips is provided for especially keeping track of the work performed by and maintenance of the rock-drilling machines and quantities of drill steel used at each working place (see Fig. 85). The classification of mining operations and the corresponding jobs and occupations applying to the same are shown by the following table:

Classification	Job Numbers
A. Mining	
1. Stoping and slicing	0 to 19 inclusive
(*a*) Operating labor	
(*b*) Maintenance labor	
(*c*) Air service	
(*d*) Shift bosses	
(*e*) Explosives labor	
(*f*) Illuminants labor	
(*g*) Timbering labor	
2. Filling	
(*a*) Breaking	20 to 29 inclusive
(*b*) Hand-tramming	40 to 49 inclusive
(*c*) Electric haulage	50 to 59 inclusive
(*d*) Miscellaneous	
3. Dead work	
(*a*) Operating labor	30 to 39 inclusive
4. Surveying	
B. Tramming, Hoisting, etc.	
1. Hand-tramming ore	60 to 69 inclusive
(*a*) Operating labor	
(*b*) Maintenance of tracks	
(*c*) Maintenance of cars	

THE NEW JERSEY ZINC CO.
FRANKLIN, N.J.
MINE OPERATING DRIFTING RAISING SUMMARY

Level	Stope	Drill Shifts per Ft. Advance	Breaking Labor per Ft. Advance	Tr. Hdl. E. H. Lab. per Ft. Advance	Explosives per Ft. Advance	No. of Holes per Round	No. of Ft. Drill'd per Round	Advance per Round	Drill Shifts per Ton	Breaking Labor per Ton	Tr. H'd'l E. H. Lab per Ton	Explosives per Ton	Advance in Ore	Advance in Rock	Total Advance	No. of Rounds	Tons Rock Broken	Total Tons Broken	Size	Inclination	Material	Remarks	Surveyors Estimate							Breaking Labor											
																							Broken Ore in Stope	Gain or Loss	Ore Broken	Ore put in Stope	Ore put in Cribs	Rock Removed	Ore Trammed	Ore				Rock				Total Breaking Labor	61	66
																														0	1	4	6	20	21	24	26			
Total Drift																																								
Total Raise																																								
Tot. Drift Raise																																								
Grand Tot. Drift																																								
Grand Tot. Raise																																								
Grand Tot. Dft R'se																																								

Mine Operating Misc. Summary

Levels	Territory	Remarks	Fill																	Ore																	Timbering						
			Breaking				Tram		Handling			El. Haul			Explosives				Breaking				Tram		Handling			El. Haulage				Explosives			Labor			Timber			
			20	21	24	26	41	46	42	44	47	51	54	56	Powd	Fuse	Caps	Exp	0	1	4	6	61	66	62	64	67	71	74	76	Cars	Pow	Fuse	Caps	10	11	16	12	10	8	Lag
Total Misc.																																									
Grand Tot. Misc.																																									

Remarks

Summary

	Broken Ore in Stopes	Gain or Loss	Ore Broken	Ore put in Stope	Ore put in Cribs	Rock Removed	Ore Trammed	Cu. Yds Fill Stopes	Cu. Yds Fill Placed	Tons Rock Broken	Mill Fill Dumped Ding Dong	Explosives Cost Ore	Explosives Cost Rock	Cost of Timber Used	Palmer Hoist Pocket	Skips	Tons	T/S	No of Working Days in Month	
															Top				Time Office Tally of 8 hr Shifts	
Grand Total Ore															800				Mine Office Tally of 8 hr Shifts	
Grand Tot Dft R'se															1150				Discrepancy	
Grand Tot. Fill															Total					
Grand Tot. Misc.															Estimated Hoist				Tons Ore and Rock Broken	
Summary															Surveyors Error				Gain of Ore in Chutes and Pockets	

THE NEW JERSEY ZINC CO.
FRANKLIN, N.J.
MINE OPERATING ORE SUMMARY

Level	Stope	Breaking Ore					Tramming Ore							Remarks	Surveyors Estimates							Breaking Ore															
																						Labor						Hand Tramming							Handling		
		Tons/Man in Stope	Tons per Dr. Shift	Tons per Ft. Hole	Explosives Cost Ton	Timber Charges	Tons Tram'd per Man Tramming	Ton Miles Hand Tram'd	Tons Tram'd per Man Elec. Haul	Ton Miles Elec. Haul	Tons Tram'd per Man Tr. Hdl. Eh.	Explosives Cost Ton			Broken Ore in Stope	Gain or Loss	Ore Broken	Ore put in Stope	Ore put in Cribs	Rock Removed	Ore Trammed	0	1	4	6	Total Breaking Labor	Explosives Total Cost	61	66	Total Tram'g Labor	Cars	Tons/Car	Tram'g Dist.	Tons Hand Tram'd	62	64	67
Tot. Inactive Stopes																																					
Tot. Active Stopes																																					
Total Top Slices																																					
Total Ore																																					

Remarks

Grd Tot. In'tive St.																																				
Grd Tot. Active St																																				
Grd. Tot. Top Slices																																				
Grand Tot Ore																																				

THE NEW JERSEY ZINC CO.
FRANKLIN, N.J.
MINE OPERATING FILL SUMMARY

Level	Stope	Breaking Fill				Tramming Fill							Total Timber Charges	Remarks	Surv Estimates			Breaking Fill						Tramming Fill									
																		Labor						Hand Tramming							Handling		
		Tons/Man Breaking	Tons per Dr. Shift	Tons per Ft. Hole	Explosives Cost/Ton	Cu. Yds. Tram per Man Tramming	Cu. Yds. By 1000' Hand Tr'm'g	Cu. Yds. Tram per Man El. Haul	Cu. Yds. By 1000' El Haul	Cu. Yds. Placed per Man Tr. Hdl. E. H.	Explosives Cost/Ton				Tot Cu. Yds Fill in Stope	Cu. Yds. Fill Placed	Tons Fill Broken	20	21	24	26	Total Breaking Labor	Explosives Total Cost	41	46	Total Tram'g Labor	Cars	Cu. Yds./Car	Tram'g Dist.	Cu. Yds. Hand Tram'd	42	44	47
Total Fill																																	
Grand Tot Fill																																	

Remarks

FIG. 88.—MINE OPERATING SUMMARY SHEETS.

THE NEW JERSEY ZINC CO.
FRANKLIN, N.J.

MINE OPERATING DRIFTING RAISING SUMMARY

DRIFTS RAISES MISC.

MONTH OF 19

Tramming Handling El. Haul Labor															Tramming				Total Drilling						Explosives								Total Cost
Ore						Rock									Ore		Rock		Hammer			Block Hole			Ore				Rock				
62	64	67	71	74	76	41	46	42	44	47	51	54	56	Total Tr. Hdl. E.H. Labor	Cars	Tons per Car	Cars	Tons per Car	Shifts	Holes	Feet	Shifts	Holes	Feet	Powder	Fuse	Caps	Expl'd	Powder	Fuse	Caps	Expl'd	

Mine Operating Misc. Summary

Timber		Drilling						Mining															Tramming and Hoisting																			
		Hammer			Block Hole			Breaking						Air Ser		Timber			Fill				Hand Tramming Ore							Electric Haulage						Pumps			Hoist			
Pik	Sibs	Shft	Holes	Feet	Shft	Holes	Feet	B	00	2	5	7	9	3	8	15	17	18	27	28	29	30	60	02c	07c	63c	68	05	69	72	77	73	78	75	79	90	91	96	80	81	82	87

Remarks

Summary

Ore				Rock				Ore and Rock	
Shifts on Brk'g. Timb'g. Air Serv.		Tons Ore Brok'n p. Man Brk'g etc.		Shifts on Breaking		Tons Rock Brk'n p. 8hr. Sht. Bk.		Tons Ore Rock Broken p. 8h Shift	
Shifts on Tram'g, Handl'g, Elec. Haul.		Tons Ore Tram p. Man Tra'g. etc		Shifts on Placing		Cu. Yds. Pl'd p. 8hr. Shift on Pl'g.		Tons Ore Broken per 8h. Shift	
Shifts on Pump, Hoist				Mill Fill Shifts not Rep'ted on Slips		T. Dump'd p. 8 hr. Sht. Mill Fill		Tons Ore Hoisted per 8h Shift	
Shifts not Reported on Slips									
Total 8 hr. Shifts for Mon. on Ore		Tons Ore Brk'n. p. 8hr. Shts on Ore		Total 8hr. Shifts for Mon on Rock		T. Broken p. 8hr. Shift on Rock		Total 8h. Shifts for Month	
Aver. No. 8 hr. Shifts p. Day on Ore		T. Ore Hoist'd. p. 8hr. Shts on Ore		Aver No 8hr. Shifts p. Day on Rock		Cu. Yds. Pl'd. p. 8hr. Shift on Rk.		Average No. 8 hr. Shifts per Day	

THE NEW JERSEY ZINC CO.
FRANKLIN, N.J.

MINE OPERATING ORE SUMMARY

ORE

MONTH OF 19

Tramming Ore														Timbering					Drilling						Explosives								Timber					
Handling		Electric Haulage												Labor				Cost of Timber	Hammer			Block Hole			Breaking				Tramming									
Total Hdlg. Labor	Supply Cars	71	74	76	Total El. Haul Labor	Cars	Tons/Car	Tramg. Dist.	Tons El. Haul	Total Lab. Tr. Hdl. E.H.	Explosives	Total Cost		10	11	16	Cost of Labor		Shifts	Holes	Feet	Shifts	Holes	Feet	Powder	Fuse	Caps	Expl'rs	Powder	Fuse	Caps		12"	10"	8"	Lag	Pik	Sibs

Remarks

THE NEW JERSEY ZINC CO.
FRANKLIN, N.J.

MINE OPERATING FILL SUMMARY

FILL

MONTH OF 19

Tramming Fill														Timbering					Drilling						Explosives								Timber					
		Electric Haulage												Labor				Cost of Timber	Hammer			Block Hole			Breaking				Tramming									
Total Hdlg. Labor	Supply Cars	51	54	56	Total El. Haul Labor	Cars	Cu. Yds/Car	Haulage Dist.	Cu. Yds El. Haul	Total Lab Tr. Hdl. E.H.	Explosives	Total Cost		10	11	16	Cost of Labor		Shifts	Holes	Feet	Shifts	Holes	Feet	Powder	Fuse	Caps	Expl'rs	Powder	Fuse	Caps		12"	10"	8"	Lag	Pik	Slps

Remarks

FIG. 88.—CONTINUED.

	Job Number
2. Electric haulage............................	70 to 79 inclusive
(*a*) Operating labor	
(*b*) Maintenance of way	
(*c*) Maintenance of cars	
(*d*) Maintenance of lines	
(*e*) Maintenance of motors	
3. Hoisting.......................................	80 to 89 inclusive
(*a*) Operating labor	
(*b*) Maintenance of way	
(*c*) Maintenance of equipment	
4. Pumping..	90 to 99 inclusive
(*a*) Operating labor	
(*b*) Maintenance of pipes	
(*c*) Maintenance of pumps	
5. Departmental expense	
(*a*) Safety	
(*b*) Light, heat and water	
(*c*) General plant maintenance	

These reports of each active working place are checked daily by the assistant mine captains in charge of a number of shift bosses in various

A B 1665 (C-4)

THE NEW JERSEY ZINC COMPANY

FRANKLIN, N. J.

DRILL RECORD

MAKE;............TYPE;............SHOP No.............
PURCHASED ON REQ.; A B............, 191   IN SERVICE;............191
CYLINDER DIAM.;............inches   PISTON STROKE;............inches   PISTON WEIGHT;............lbs.   oz.
AVGE. DRILLING SPEED (to date)............inches per min.   AVERAGE AIR CONSUMPTION;............cu. ft. per min. free   FACTOR;............
ORIGINAL:—   DRILLING SPEED............inches per min.   AIR CONSUMPTION............cu. ft. per min. free   FACTOR;............
PRESENT (   ):—DRILLING SPEED............inches per min.   AIR CONSUMPTION............cu. ft. per min. free   FACTOR;............

DATE		WORKING PLACE		DRILLING TIME		No Holes Drilled	Footage Drilled	Drilling Speed in Ft. per Hr.	Material Drilled	Pcs. Drill Steel Used						Hrs. Repair Labor			Repair Parts (Maker's Symbols)		M'T'CE Charges		
		Lev.	Loc.	Hrs.	Min.					1st	2nd	3rd	4th	5th	6th	.40 Special	$2.88	$2.56			Supplies	Mine Labor	Mach. Shop Labor
Brought Over																							
1st	DAY																			N			
	NGT																			O			
2nd	DAY																						
	NGT																						
3rd	DAY																						
30th	DAY																			N			
	NGT																			O			
31st	DAY																			N			
	NGT																			O			
TOTAL FOR MONTH																							
Carried Forward																							

FIG. 89.—DRILLING SUMMARY SHEET.

territories, and are then entered by clerks on distribution analysis sheets which are filed in loose-leaf ledgers and totaled twice each month (see Fig. 87 and 88). The month's summaries are then transferred to the summary sheets (Fig. 88).

The time-keeping system is a double check one. Every workman not on the staff enters the plant through a turnstile gate, by presenting his pay check, a special brass check with his number on it. The time office records are made from this. Each shift boss reports his men in a time book which is checked each day against the time office records so

that any errors may be looked up and corrected. Should a man go home early, or work overtime, he is provided with a card showing this, so that again a check is had against the shift bosses' time book. Each occupation is known by a symbol which, when placed in the time book, indicates the occupation at which a man has worked, and should he be employed at his regular job, no symbol is necessary. These symbols are keyed, so that the unit's digit indicates the rate of pay.

There has been no attempt to make this article in any sense complete or it would be entirely too lengthy. Features in regard to power plant and mining equipment have not been mentioned, although they may be evident in the illustrations, as, for instance, the coal-storage device shown in one of the views of the plant.

Discussion

Robert Peele, New York, N. Y. (written discussion*).—This excellent paper is one of a number of recent detailed descriptions of the methods and plant in use at prominent mines of the country. The publication of such papers cannot be too highly praised. In the past, the interchange of inside information has often been hampered by the disinclination of some mining companies to permit publication of details of their work. These companies apparently have not realized fully that free discussion is of mutual benefit; that members of mine staffs are aided in keeping up to date, in improving old methods and devising new ones to meet changes in conditions, and, by a wholesome feeling of rivalry, are stimulated to introduce economies and do better work.

In writing this paper, Messrs. Haight and Tillson have packed a large amount of information into a relatively small space, and the numerous illustrations, chiefly reductions of working drawings, are valuable. The authors are also to be commended for their highly detailed Table of Contents, which adds to the usefulness of the subject matter.

The method of mining at Franklin does not fall readily into any of the accepted classifications. It is a combination of shrinkage stoping and mining of the pillars by a modified top-slicing method. Attention may be called to the fact that, in the wider parts of the orebody, the stopes, 17 to 18 ft. (5.2 to 5.5 m.) wide are transverse; in the narrower parts, they are longitudinal, that is, in the direction of the strike. This variation is made necessary by the shape and character of the deposit, and distinguishes practice at Franklin from that in some of the western mines. The direction of the stopes is changed from transverse to longitudinal when the orebody narrows sufficiently to make the length of a transverse stope approximately equal to its width. Some of the longitudinal stopes exceed 120 ft. (36 m.) in length, and when the hanging

* Received Oct. 8, 1917.

wall is bad, requiring prompt filling, three operations proceed simultaneously in the same stope, viz., ore is broken at one end, while drawing down goes on in the middle, and filling at the other end. Each transverse stope or chamber is begun by a crosscut from foot to hanging wall, which is afterward enlarged to the stope width and to a height of 12 or 15 ft. (3.7 or 4.6 m.). This preliminary work is done only once for each stope, because the stope is carried up far enough to continue stoping from the level above, after the lower lift has been filled. By adopting this plan, development work is minimized in both time and cost. An ore lying below the lowest level, on the bottom rock, is benched out by underhand stoping, and the space filled up to the floor of the level. Then a raise, for ventilation and access, is put up to the level above and stoping begins. In the longitudinal stopes, the orebody is in places wide enough for two opening drifts, one on the footwall, the other near the middle, the raises being started from the footwall drift. Local conditions, such as a flat or bad hanging wall, have demanded certain minor variations in both types of stope, which are fully discussed in the paper.

It may be useful to compare the above method of opening and working the stopes with those employed elsewhere in deposits of somewhat similar character. In the King mine, of the Arizona Copper Co., for example, there are two orebodies, 500 and 700 ft. (152 and 213 m.) long, having a maximum width of 30 ft. (9 m.) and a dip of 70°; the walls are good. Here, haulageways were driven in wall rock (sometimes in both walls), with crosscuts 25 ft. (7.6 m.) apart, center to center. Then the entire sill floor was taken out as a breast stope, and the work carried up as a flat-back shrinkage stope. Raises were at 100-ft. (30-m.) intervals along the strike. The back stopes were 25 ft. high, and advanced toward each other from adjacent raises.

In stoping of this kind, in veins of rather flat dip, shovelers must be employed to throw broken ore from the footwall to the hanging side, so that the footwall may be reached by the drill holes and to keep the stope properly filled. It is stated that, at the Franklin mine, because of this difficulty, when the dip of the orebody is less than 50° or 60°, experience indicates that 100 ft. is the greatest vertical distance that a wide longitudinal stope can be carried up.

At the Alaska-Treadwell mine, the orebody, though in places reaching a thickness of over 400 ft. (dip 50° to 65°), is worked by large rooms treated as shrinkage stopes. The pillars between the rooms are continuous to the top of the orebody, but the stopes themselves terminate when they break into the level above. Stoping begins by breasting out a crosscut, driven from the main level (or from an intermediate drift driven from a raise), over the entire area between pillars and from wall to wall. It is then carried up as an arched-back shrinkage stope.

Since the dips, and other physical features of the orebodies in the King and Alaska-Treadwell mines do not differ greatly from those at Franklin, it might be supposed that the stopes in these mines could be carried continuously for considerable heights, as at Franklin. But, instead of doing this, the initial development, consisting of relatively expensive crosscuts and sill-floor excavation, or flat-breast work, is repeated on each level; a matter of some consequence in point of first cost.

The Franklin mine method of development, varied in detail from time to time to meet local changes in dip, thickness of orebody, and character of walls, has solved the difficult problem of coördinating the smaller scale and less economical work previously carried on by several different companies.

The interesting description of pillar mining, given by Messrs. Haight and Tillson, indicates that the Franklin mine management has made a most careful study of the methods employed in western and northwestern mining districts. A number of methods, of both caving and non-caving types, have been tried. The method found best thus far for the pillars, but which, as the authors of the paper state, may not yet have reached its final form, is top-slice caving. Caving is begun after a width of three sets (22 to 25 ft.) has been opened the length of the pillar. A slice of four sets can sometimes be taken, but the timbering usually begins to crush before that width is reached. Practically all the ore in the pillars is recovered, though at a considerably higher cost than for ore from the stopes.

Lucien Eaton, Ishpeming, Mich. (written discussion*).—Upon reading the paper by Messrs. Haight and Tillson, on Zinc Mining at Franklin, N. J., I was impressed first of all by the minuteness with which each detail has been worked out and recorded. The same impression is given by a visit to the mine. Last winter I had the good fortune to visit the Franklin mine, and was struck by the similarity in size and shape of the orebody there to that of the Lake mine of the Cleveland-Cliffs Iron Co. at Ishpeming, Mich., with which I have been intimately acquainted for over 16 years. My knowledge of the mine covers a period during which radical changes in mining practice have been effected. On account of the similarity between the two orebodies, a short history of these changes may be of interest.

The original plan for mining the orebody at the Lake mine was to use square-set rooms and pillars, each three sets wide (21 ft.–6.4 m.), laid out at right angles to the long axis of the orebody, and three levels were opened up according to this plan. It was also planned to fill the rooms with rock before mining the pillars, and many of the rooms were so filled

* Received Nov. 16, 1917.

before the mining method was changed. At this time the ore was hoisted through an inclined shaft with four compartments, two compartments being used for hoisting ore and rock, one for lowering timber and supplies, and one for pipes and ladderway. As the depth of the bottom level was only 350 ft. (106 m.), no provision was made for handling men.

The ore at the Lake mine is a hydrated hematite, as a general rule considerably softer than the ore at the Franklin mine, but there are places where it is very similar in hardness and toughness. The ore in both sides of the trough was wider than it is in the corresponding parts of the Franklin orebody, but at the bottom of the trough under the capping the width is about the same in the two mines. In the early years, when the softer ore near the top of the orebody was mined, drilling was done by hand with jumpers and augers or with hammer and drill. The ore has become harder in depth, however, and during the last 10 years drilling has been done with machine drills of one type or another.

About 20 years ago, mining by square-set rooms and pillars was discontinued, and top slicing and caving, taking the ore in slices two sets high, was adopted. During the last five years this has been gradually changed, until now nearly all the slices are only one set high. I was not present when the method of square-set rooms and pillars was in use, but I had the misfortune to mine many of the pillars left between the rock-filled rooms, and it was a disagreeable and expensive job. During the time that mining was confined to the two sides or legs of the trough, the weight of the crushed capping, timber, etc., above workings was not sufficiently heavy to cause serious inconvenience in mining, but when the house-of-rock between the two sides was undercut, the pressure increased enormously, and, if there had been any pillars left to mine, much of the ore in them would have been lost. Even the hardest ground in the middle of the vein is crushed quickly, and it is not uncommon to have a drift go down bodily, the ore above the timbers remaining intact and so hard that drilling and blasting is necessary before new timber can be put in. Before the central house-of-rock was undercut, it was not unusual for a drift in the harder ore, near the capping, to last for a couple of years or more, but now a drift near the top of the orebody seldom lasts more than a month without retimbering.

The ore at the Franklin mine is so valuable that complete extraction is more to be desired than low mining costs, and if the same effect is produced by the weight of the capping, when it has been undercut over a large area, as at the Lake mine, great difficulty will be experienced in mining the pillars between the transverse rooms, and the attendant loss of ore will more than offset the saving in cost achieved by the use of these rooms. This saving is already largely offset by the higher cost of slicing in small pillars.

As crushing proceeds and the weight of the capping increases, more and more timber is necessary, but this additional cost is very nearly offset by the reduction in the cost of explosives. In fact, in the mines on the Marquette Range in normal times, the cost of timber plus explosives at different mines is remarkably uniform.

The handling of the large amounts of timber required by a mine of the size of the Franklin or the Lake raises the question as to the relative value of inclined and vertical shafts. Vertical shafts require longer cross-cuts to the orebody, but are cheaper to sink and require less shaft per foot of depth. Maintenance of timbering and skips is much less with vertical than with inclined shafts, and for large orebodies this item alone usually offsets the higher cost of cross-cuts. Moreover, the greater facility for handling timber afforded by a vertical shaft is an item of great importance in a large mine. For this reason, when it became necessary to sink a new shaft at the Lake mine in 1903, it was decided to sink a vertical shaft, in which a large enough cage could be used to permit the transfer of timber underground directly on the trucks without rehandling. On this cage, poles 9.5 ft. (2.9 m.) long can be handled on the trucks. This design of shaft was so successful that a slight modification of it was adopted as standard in the company, and no new inclined shafts have been sunk at any of the company's mines. The standard size of shaft is 10 ft. 10 in. by 14 ft. 10 in. (3.3 by 4.5 m.) inside of steel members, and the cage will carry 10-ft. timbers laid horizontally on the trucks. It seems to me that a vertical shaft of this general design would be preferable to the inclined shaft now serving the Franklin mine, and it would be interesting to hear from Mr. Haight or Mr. Tillson the reasons which led to the selection of an inclined instead of a vertical shaft.

Characteristics of Zinc Deposits in North America

Their Bearing on Origin and Tenure of Economic Life

BY FRANK L. NASON, M. A., WEST HAVEN, CONN.

(St. Louis Meeting, October, 1917)

THE complete statistics of zinc-ore production in the United States for 1916 are not yet available. The following figures are, therefore, only approximate. The total production of concentrates for this year, including New Jersey and Tennessee, amounts to 1,280,000 tons. Of this tonnage, five mines produced approximately 536,000 tons, or about 42 per cent. The remaining concentrates came mainly from the "Valley States," Missouri, Wisconsin, Kansas, Oklahoma, and Arkansas. The 1,280,000 tons of concentrates represent about 25,000,000 tons of crude or run-of-mine ore. Of the five mines producing the 42 per cent., the dimensions of three are approximately 25 by 1500 by 2000 ft. (7.6 by 457 by 610 m.) or more. Four of the five mines are undoubtedly veins in the common acceptance of the term. They are located in great vein systems. The mines of "Valley States" have not, so far, been classed as veins; simply as "secondary deposits." The origin of these secondary deposits has been a warmly debated subject. Many geologists contend that they are the results of leaching from overlying and adjacent rocks carrying but traces of zinc, these traces leached out and deposited in favorable places. If this theory is correct, deposition is strictly limited to shallow depths. No volume of ore comparable to that of "true fissure veins" is to be expected. Just what this means to the future of zinc mining in these States is here indicated. The total zinc-bearing areas of these States amount to about 9124 sq. miles (23,631 sq. km.). Assuming that 10 per cent. of this total will be more or less productive, there are 912 sq. miles (2,362 sq. km.) of "mineral" ground. The average depth of the operated mines is something like 100 ft. or less. In *Bulletin* 606, *U. S. Geological Survey* (1915), p. 202, Siebenthal makes the following statement: "The writer is informed by William Waugh, one of the drillers of the Stone City well, that cuttings from other wells which he drilled in the vicinity of Pittsburgh, Kan., have shown galena and sphalerite taken from a depth of 591 ft. (180.14 m.) and that the showing of ore at some places would be considered good enough to justify the sinking of a shaft had it been found in the Joplin district and at shallower depths." Leaving out of the question the prohibitive cost of such a shaft, there is

more than a hint that the ore reserves of these States may be tentatively reckoned as being increased by nearly 600 per cent. This, of course, means the total past production plus orebodies that may yet be found at the present average depth. In the body of this paper I have cited examples of drill holes having cut zinc ores at a depth of over 1000 ft. (305 m.) below base level of drainage. Ores occurring at this depth below base-level drainage could hardly be accounted for by downward-flowing waters; this, according to Mr. Siebenthal, is not necessary. In the *Bulletin* already referred to, p. 42, he says: "The writer believes that, directly or indirectly, the Cambrian and Ordovician limestones and dolomites have furnished the major part of the ores now found in the Joplin district; that these ores were precipitated from ascending alkaline-saline sulphureted waters."

Mr. Siebenthal's theory is not new; he was anticipated by 28 years. Walter P. Jenney[1] comes to the following conclusion: "No good reason appears for placing these ore deposits (Valley States) in a separate class from fissure veins because of distinctions based upon the form and position of the orebodies relative to fissures. On the contrary, these runs, considered in connection with the fissures through which they were formed, belong to the great class of ore deposits in which the origin of the ore is in mineral-bearing solutions, ascending through the fissures from some source of unknown depth in the crust of the earth, of which class fissure veins are the simplest typical form." Both Jenney and Siebenthal base their conclusions on long and close study of the mines in the fields of the Valley States. The recorded conclusions of two such hard and conscientious workers seem to make this paper superfluous. I can add little, if anything, to their evidence. The most that I can expect to do is to record the observations made during my years of experience in the zinc fields of the United States, Canada, Mexico, Guatemala, and San Salvador. I record the results of my observations under the following heads: Formative Influence of Eruptives; Contact Phenomena; Fissuring and Faulting; Minerals Accompanying Fissure Deposits without Eruptives; Minerals Accompanying Fissure Deposits with Eruptives; Zinc-Bearing Fissures in Limestones and Other Rocks Accompanied by Eruptives; Contact Zinc Deposits; Secondary Enrichment of Zinc Deposits; Depths at Which Zinc and Lead Deposits Have Been Found.

The conclusions which I have personally drawn from the above phenomena seem to strengthen the position taken by Jenney and Siebenthal that fissuring and faulting are the prime prerequisites for the formation of zinc-ore deposits; Jenney seems to go farther than Siebenthal in that he concludes that the zinc-ore deposits of the Valley and Appalachian States are, in effect, fissure veins with all of the essential characteristics of this class of deposits; that in no essential do they differ from the recog-

[1] *Trans.* (1893), **22**, 217.

nized veins of the Rocky Mountain region. My observations compel me to follow Jenney.

Formative Influence of Eruptives

The greater number of zinc deposits in the Rocky Mountain region in British Columbia, the United States, Mexico, Guatemala and San Salvador are characterized by the presence of eruptives. This is also true of the zinc-lead deposits accompanying the fluorspar deposits of the Kentucky-Illinois field. In Butte, Mont., the ores of the great zinc mines of the Butte and Superior, the Elm Orlu, Lexington, Alice, Moulton, Emma, and Nettie and other minor deposits, occur in great fissures; in the granite. The eruptives, aplite and rhyolite, seem to have intruded later than the mineralization of the zinc veins. Certainly many of the copper veins have been broken and displaced by the intrusion of later dikes. The later eruptives in this district thus seem to have played no part, even indirectly, in the zinc mineralization.

In the Baker-Neihart district the silver-lead-zinc veins are in fissures in probably pre-Cambrian gneisses and in Paleozoic limestones accompanied by eruptives in the form of dikes. In this district evidence as to formative influence of eruptives is inconclusive. In the Troy district in northwest Montana, the lead-zinc deposits occur in fissures in the pre-Cambrian slates. The same is true of the Highland-Surprise and the Douglas on Pine Creek near Kellogg, the Interstate-Callahan and the Success near Wallace. In these mines the country rock is faulted heavily, and in all of them occur dikes and bosses of quartz-monzonite and dikes of other eruptives.

The present working adit of the Interstate-Callahan is 1 mile (1.60 km.) long. It followed in on a thin leader of zinc-lead ore. At a distance of about 1 mile it cut through a trap dike and picked up the great vein on the far side. One wall of the shoot was slate, the other quartz monzonite. Heavy faulting as evidenced by the juxtaposition of two distinct sedimentary horizons, is present.

In Madison, N. H., a wide vein of silver-lead-zinc ore occurs in a fissure in the granite. No later eruptives are noted in this locality. In the zinc fields of Missouri, Kansas, Oklahoma, Illinois and Wisconsin, and of eastern Tennessee, Virginia and Pennsylvania, the ore deposits are characterized by heavy faulting and are also remarkable for the entire absence of eruptives of any kind.

The presence of fluorite, supposedly an evidence of the presence of eruptives, is found sparingly at Austinville and Cedar Springs. In one Virginia locality an undeveloped deposit of zinc is characterized by the fact that at least 50 per cent. of the gangue is fluorspar. The color of the latter is the characteristic deep purple. It may be that white fluorite occurs and has escaped detection in the calcite and dolomite gangue,

but if it occurred in any great quantity its presence would have been noted by the smelters. Fluorite is the only hint of the possible presence of eruptives in the vicinity of the zinc mines.

The Hanover, N. M., zinc deposits are, in places, accompanied by rhyolite and basic dikes in addition to the great body of porphyry directly east and west of the railroad.

In Pembroke, Maine, there are a few remarkably rich pockets of blende in persistent fissures in diorite. On Deer Isle, at Cape Rozier, are numerous lenses of rich zinc-lead ore carrying silver. These probably occur in pre-Cambrian sheared eruptives. These deposits are accompanied by diorite dikes.

In the Charcas district, San Luis Potosi, Mex., which almost identically reproduces the Hanover district, numerous acid and basic dikes are present in addition to the basal porphyry. These cut the orebodies in several places, as a rule, displacing them. At Charcas, in the highly metamorphosed hill known locally as "La Bufa," numerous acid dikes cut the orebodies, obviously later than the orebodies themselves. Feldspar porphyry is an abundant rock. It appears flanking the limestones, and in dike-like masses in the limestones. Whether the porphyry is really an eruptive or only has the appearance of it, depends wholly on whether it is basal or not, taking the appearance of an eruptive by means of faulting. The latter explanation appears to me to be the only tenable one. This much is certain in either case—no important mineralization is found either at the contact with bedding planes or at the contact of apparent dikes. This condition is particularly evident at Hanover.

Near Wallace, Idaho, one prospect, carefully examined, showed a zone about 7 ft. (2.13 m.) wide in which three stringers of blende could be seen. These occurred in pre-Cambrian slates. No eruptive was nearer than 100 ft. (30.48 m.). Crushed rock evidenced faulting. On the slope of the hill was a small outcrop of quartz-monzonite.

Typical illustrations of the lack of effect of eruptives on mineral deposition and metamorphism are shown in Fig. 6. West of the Nason tunnel, dikes and sills have produced neither effect. A dike occurs cutting through both the orebody and the garnet at the east end of the above tunnel. Evidently the dike could have had nothing to do in this place with the mineralization or with the metamorphic garnet. Fig. 5 seems to present even stronger positive evidence. At the western end of the main tunnel a block of ore and limestone is cut off entirely from the main body by two dikes; farther in, the ore zone is faulted in contact with dikes, probably because of them. Here again the eruptives are younger and have no formative effect. These, as was stated, are types selected from a large number of similar examples.

To sum up briefly, in some of the cases cited, eruptives are to be noted; in every case, faulting and fracturing occur. The only conclusion to be

drawn seems to be this: Whatever the formative influence of eruptives on mineral deposits may be, they are not absolutely essential. On the other hand, faulting and fracturing are essential. Certainly this is true: In very many instances eruptives have followed mineral deposition; not preceded it.

Contact Phenomena

It seems to be the general impression that when an eruptive cuts across bedding planes of sediments, cuts through an older eruptive, or interpolates between sedimentary strata as blankets or *mantos*, the plane of contact is a favorable locus for mineralization. This mineralization may take the form of economic minerals or metamorphic minerals not useful.

Indirect negative evidence is more than suggested in the paragraphs immediately preceding. Eruptives are not universal accompaniments of ore deposits. Several examples of eruptives which, alone, have been entirely lacking in metamorphic influence can be cited.

In the San Carlos mountains, about 90 km. (55.9 miles) southeast of Linares, Mex., are copper mines which have been worked extensively. The mountains are comparable to a great volcanic crater; the village of San Carlos lies in the cup of the crater. The crater rim on the north, west and east, as well as large areas in the cup itself, are massive limestones. Within the cup, and in places on the rim, are immense masses of solid garnet. In this garnet the copper veins are found. In more or less intimate relation with this garnet are dikes and, possibly, eruptive bosses. The eruptives have been supposed to be laccoliths, mushroom-shaped eruptives, where the volcanic neck is the stem and the eruptive, spreading from this neck, the umbel. It has been assumed that the garnet and the cupriferous veins were the results of contact.

A large part of the cup is soil-covered; the rim, bare rock. Careful study of several miles of the rim disclosed the following facts: Starting on the unchanged blue limestone, and following along the exposed strata, white spots began to appear and along the bedding planes a slight bleaching was noted. The bleaching extended through the entire mass; the limestone changed to a white, crystalline marble; then clots of amorphous garnet; finally, great, vertical, dike-like masses of pure garnet. Still following along the line of strike, the changes were reversed until another garnet dike was approached. The above phenomena were observed on the three limestone rims. For miles, between the garnet dikes, absolutely unchanged limestone lay above and below the intrusive sheet and in absolute contact with it.

At Charcas, San Luis Potosi, Mex., are two mines, the "Tiro General" and "La Bufa." These mines lie in a U-shaped valley whose axis runs nearly due east and west. The mines are about 2 miles (3.22 km.) apart.

They lie on the south slope of a range of mountains that rise about 1500 ft. (457 m.) above them. The slopes and crest of the mountain are about equally divided between a coarsely crystalline feldspar porphyry and a blue limestone.

The Tiro General is located on a well defined east and west fissure. La Bufa is not certainly located on a fissure. The orebody, zinc and lead, is, so far, confined to bedding planes that dip to the north at an angle of about 45°. The hanging wall of the orebody has not been definitely located; the foot wall appears about 1500 ft. above the outcrop and on the slope of the mountain. It has also been located immediately under the orebody between the fourth and fifth levels in the Malacate shaft. (See Fig. 8, "Probable Foot Wall.") Within the great orebody are masses of garnet and epidote with equal masses of wollastonite. These rocks, especially the garnet, carry large masses of pure hematite. Along the east and west strike and up the slope of the mountain is what was formerly a limestone, now entirely changed to massive wollastonite. Brecciated porphyry carrying interstitial blende is found a little above the fifth level (Fig. 8) and below. The limestone is unchanged.

At the extreme east end of the first level, a supposed porphyry mass was driven through, seemingly indicating a dike rather than a massive body. The rocks, however, are so rotted that on the east ends of levels 1 and 2 positive identification is impossible. A porphyry contact indicated as possible is thus unproved. The garnet, hematite and wollastonite noted in the section cannot thus be certainly ascribed to contact reactions. Fissuring, only, is absolutely certain. For a long distance around the mine, the chert of the limestone is changed to the same mineral. The same mineral with crystalline quartz occurs between bedding and joint planes. For 2 miles or more west of La Bufa, the above conditions obtain except that zinc and lead ores are interrupted along the line. The western end of the valley is terminated by a connecting north and south ridge that unites the two ranges. Where the axial line of the Tiro mine crosses the ridge, there is an immense dike-like mass of garnet and wollastonite rock. North and south of this dike the limestones are unchanged, though there are many eruptives in the form of dikes. Numerous draws on either flank of the valley expose sills of eruptives in immediate contact with both unchanged limestone and slates. There is a perfect exposure of both upper and lower contacts, but at no point is there any sign of metamorphism or mineralization.

In Animas Forks, Colo., is a great fault fissure 100 ft. (30.48 m.) thick. At one end is the Sunnyside mine; at the other, the Gold Prince. Both of these mines have free gold and this is accompanied by sulphides of zinc, lead and copper. There is also much iron pyrites with rhodochrosite in a gangue of crystalline quartz. The mines are about 3 miles (4.83 km.) apart. Between, the vein is exposed as crystalline quartz

with much silica in the colloidal form. In small bunches, galena and sphalerite occur with rhodochrosite. These are of uneconomic size. I saw no eruptives in the locality. The throw on the fault vein is, as I recall, not less than 3000 ft. (914.4 km.). The vein filling was, in the less metamorphic portions, composed of partly digested fragments of the original rock breccia.

At Breckenridge, Colo., limestones and shales are exposed in the various gulches with sills of eruptives in immediate contact but with no metamorphic changes. The mines here are sometimes located on faults or fissures in the eruptives, often at seeming contacts, but invariably accompanied by faulting. Judging by cited examples—and localities have by no means been exhausted—simple contact between eruptives and sedimentaries, between older and newer eruptives, both metamorphism and mineralization depend either wholly or in part upon other phenomena than simple contact. In the case of the great vein near Animas Forks, it appears to be evident that the metamorphism here exhibited depended upon causes wholly independent of eruptives. We can go a step further and say that there are many zinc and lead deposits that are wholly independent of eruptives, apparently depending upon fissuring alone.

The citations from the San Carlos mountains and the Charcas district in Mexico, at the most, can only be put into the neutral class since, in the instances cited, the extreme metamorphism is invariably accompanied by fissuring or faulting and nowhere else along the line of contact is any metamorphic change to be noted. The same conditions prevail to a striking extent at the zinc mines of Alotepec in Guatemala and near Metapan in San Salvador. In the last cases cited, veins in the same eruptives that form dikes in the limestones and occur in the zinc-lead veins, are filled with sulphides of lead, zinc and copper. There can be no question in this case that the minerals in these veins are later than the eruptives; by natural inference, if not by direct evidence, these same eruptives had nothing to do with the mineral veins in which they occur. In the Malacate shaft at La Bufa mine, at a depth of about 700 ft. (213.36 m.) the shaft cuts through the foot-wall limestones that appear under the zinciferous blankets. At about this point in the shaft brecciated porphyry appears. The interstices are filled with blende and galena. The dike is nearly vertical and until the dike is passed through the lead and zinc minerals are found. Below the dike (in the shaft) lead and zinc appear in brecciated limestone.

In the "southwest opening" of the New Jersey Zinc Co.'s mine at Mine Hill, N. J., a large trap dike cuts the willemite-zincite-franklinite orebody in an east and west direction. Between the dike and the regular orebody there is a more or less persistent vein consisting mainly of cleiophane, a white sulphide of zinc. It is accompanied by a considerable

amount of fluorite. This occurrence is to be contrasted with an occurrence of honey-yellow blende in a fault plane at the Stirling Hill mine. No eruptive accompanies this last occurrence.

Fissuring and Faulting as Related to Zinc Deposits

In eastern Tennessee and southwest Virginia are many striking examples bearing on the above subject. The most noted example, owing to its successful commercial development, is the Mascot mine about 14 miles (22.53 km.) east-northeast of Knoxville. This mine is located on a brecciated fault line that can be almost continuously traced from Eves Mills in Loudon County to Talbot in Hamblen County, a distance of about 75 miles (120.7 km.). Zinc and lead are found at Eves Mills, Unitia, Friendsville, Phillips farm near Caswell, Loves Creek, McMillan, Roseberry, Mascot, New Market, and Mossy Creek on this line. At Talbot, heavy beds of breccia occur, but so far as known they are wholly barren. Mascot is the only locality that has developed blende in economic quantities. New Market, Emberville, and Mossy Creek have produced and are now producing a considerable amount of oxidized zinc. Fairly extensive drilling in these last three localities has, so far, failed to reveal economic blende. Intense brecciation along this line is self-evident. The evidence of faulting is more or less indirect. Both foot and hanging walls are of Knox dolomite and, except for position, indistinguishable. United States geological folios covering this field show overthrust faults, in many localities the fault plane being almost parallel to the dip of the strata. Local identification of a fault at any given point is thus almost impossible. Only by following the strike line of brecciation to points where readily differentiated rocks occur unconformably can the overthrust fault be established. A specific instance on, or near, the line of the Mascot brecciation is west-southwest of Morristown where the Rome formation, normally about 2000 ft. (609.6 m.) below the Knox, is thrust over this latter formation. The fault plane thus represents a vertical lift of 4000 ft. (1219.2 m.), more or less, and the break must thus extend far below the Knox formation.

In Hancock County, south of Sneedville, there are deposits of zinc and lead, both sulphides and oxides, for a strike distance of over 7 miles (11.26 km.) along Comby Ridge. The ridge is cut by deep draws at right angles to the strike. In these draws zinc and lead are found along evident slip (not to say fault) planes, along both bedding and jointing planes. Between Comby Ridge and the Clinch River are several distinct strike faults where the Marysville limestone is brought up against the Conasagua shales. Blende is found along these fault planes also, impregnating the limestone for a considerable distance from these planes. The Marysville limestones here (west of Comby Ridge) rest directly on

the Briceville shales (Carboniferous) establishing a fault with a vertical lift of not less than 3500 to 4500 ft. (1066.8 to 1371.6 m.).

In Union and Claiborn Counties, Tennessee, are numerous fissures running nearly due east and west. In places these fissures are parallel and are separated by about 100 ft. (30.48 m.) (see Fig. 2, cross-section of New Prospect mine). The rock between the fissures is more or less brecciated. In other places, as at Goin, a single smooth-walled fissure from a few inches to more than 3 ft. (0.9 m.) thick is observable. The fissures are occasionally filled with plates of solid zinc carbonate or silicate. In these counties are many examples of these two types, single and double fissures. No one of them lacks more or less mineralization. From personal examination, these fissures extend in an east-northeast line for 25 miles (40.23 km.) and east and west for at least 15 miles (24.14 km.). Not certainly the same fissure, but at least the same line, has been traced for over 75 miles (120.7 km.), from the eastern foot of the Cumberland Mountains through to the foot of the Appalachians. Near New Market in Shenandoah County, Virginia, there are three parallel sets of fissures. One line is picked up about 1 mile (1.6 km.) south of New Market and can be traced for over a mile; the other two are about 5 miles (8.04 km.) to the west. The Shenandoah River seems to flow along the eroded crest of an anticline since the inclosing limestones dip to the southeast and northwest respectively. I personally followed two of these fissures for 4 miles (6.43 km.). I was told that they could be traced continuously for 4 miles additional.

In Virginia, the most important fault occurs on the New River in Wythe County. The New River here, as at New Market, flows along the eroded crest of an anticline. The fault line can be followed more or less continuously from Ivanhoe to near Allisonia, a distance of over 20 miles (32.18 km.). Two large zinc mines have been proved on this line —the Austinville and the Bertha. There are several other localities with favorable zinc showings, but these have not been developed. The Austinville mine is located on a fault striking about N. 45° E. It has a vertical throw of not less than 3000 ft. (914.4 m.) (see Fig. 1). The fault line proper cannot be seen nor can it be exactly located, owing to the mantle of clay. The fact that there is a fault with the given throw is established by the identity of the "ribbon" limestones abutting against the dolomites at the "Stamping Ground" (see Fig. 1) with the ribbon limestones underlying the dolomites at New River at Thorn's ferry. Without details, I merely call attention to fault lines in Pennsylvania, notably at Friedensville; blende-bearing fissures or faults in the white limestones in Sussex County, New Jersey. The Cruz del Aire and Dulces Nombres mines in Nuevo Leon; the Bajan and Encantada zinc-mining districts in Coahuila, Mexico, are districts in which extensive folding with resultant faulting has taken place. The mountains in which these mines occur

are limestone. In the Bajan district the underlying sandstones are exposed and the mines occur a comparatively few feet above the sandstones in the limestones. In the Dulces Nombres district, shales show at the base of the limestone formation. The zinc mines themselves occur at the very summit of the mountains, 10,000 to 14,000 ft. (3048 to 4267 m.), geologically, above the shales.

This much should, however, be added. I examined the zinc deposits in the above-mentioned fields but had time to examine the limestones only in the immediate vicinity of the mines. There might have been eruptives back of the outcrops and on the far slopes of the mountains, but extensive excursions through the Sierra Madre Oriental Mountains in Tamaulipas and Nuevo Leon showed this range to be remarkably free from eruptives. One cannot speak positively on this point without careful local investigation. This much is certain. In many respects the zinc deposits in the above sections of Mexico resemble very closely the fissure deposits of eastern Tennessee and southwest Virginia.

In Wisconsin, Missouri, Kansas, Oklahoma and Arkansas, zinc and lead fields are often cited as examples of large areas, not only notable for the lack of eruptive rocks, but of faulting and fissuring as well. On p. 426 of *Mineral Deposits,* Lindgren gives a typical illustration of "flats" and "pitches" as well as of disseminated lead and zinc in Wisconsin. The illustration is credited to T. C. Chamberlin. The generalized illustration by Prof. Chamberlin shows little or no folding that might account for the "pitches;" while the underlying glass rock and oil rock shows no disturbance whatever. Fig. 9 is a sketch which I made on the ground at the Kennedy mine near Hazel Green, Wis. The axis of the mine runs north-northwest; the section is made at right angles to the axis. The rolls *A* and *B* are from 10 to 15 ft. (3.05 to 4.57 m.) high and in this particular section, at *F–F*, are faults with throws of from 18 in. to 2 ft. (0.46 to 0.61 m). One can pass entirely through the Kennedy mine into the workings of the Winnebago on the north and find the same structure almost continuously wherever the workings have gone down to the glass rock.

In every mine that I examined in Wisconsin, similar structures show rolls in the glass rock with minor faults and slips which show more or less in the overlying galena formation. I quote from a formal report which I made on the Wisconsin zinc field in 1909: "In the underground study of exploited mines one will note that the richest ore occurs in zones of broken rock, locally known as pitches; that this ore goes from the pitches into flats, but only where the rocks are more or less warped and disturbed; that in the glass rock acute rolls 15 ft. high, more or less, are faulted; that these fault planes pass into the galena limestones above and into the underlying rocks."

The axes of these rolls conform to the direction of drainage streams.

By studying the topographical maps of the Wisconsin Geological Survey one will note that the Plattevile limestones, the St. Peters sandstone, and the Prairie du Chien limestones occur in a manner proving that the rolls noted in the glass rock in the mines belong to the same general system of folding. Fig. 10 shows a measured cross-section through the Kennedy mine on the glass-rock horizon. Faulting is very evident in the Arkansas zinc field; it is evident on the surface as well as underground in the Oklahoma field.

In the disseminated lead fields of southeast Missouri, extensive and heavy faulting is readily noted underground and is often plainly to be noted on the surface. The lead deposits are closely associated with these faults.

Minerals Accompanying Fissure Deposits Without Eruptives

At the Mascot mine in Tennessee, while the foot and hanging wall dolomites are, in places, slightly bleached, there is hardly a perceptible change from the general light-colored, crystalline structure of the Knox as a whole. The brecciated fragments seem to be bleached in direct proportion to their size. The larger masses are bleached on their rims. The smaller fragments are sometimes more coarsely crystalline than the rock mass but the fragments are still recognizable as such. The interstitial filling is almost wholly calcite and dolomite. The blende, which is practically the only other mineral, is sometimes attached to the rim of the fragments; more rarely, imbedded in the interstitial crystalline filling.

At the New Prospect mine, the heavier beds at the base of the deposit are broken and warped but not brecciated. There is considerable interstitial secondary dolomite. The metallic sulphides in the order of their abundance are sphalerite, galena and iron pyrites. The lead and zinc occur in fractures in the dolomite and along jointing and bedding planes; also in the main north and south vertical fissures. In these places the lead and zinc minerals are very coarsely crystalline. Iron pyrites occur in small crystals. There are dolomite strata from a few inches to 3 ft. (0.9 m.) in thickness where zinc is 50 per cent. (sphalerite, 74 per cent.) of the weight of the mass. Other strata have no more than 1 per cent. or less of zinc. The blende in these strata is very fine-grained, seemingly replacing the dolomite molecule by molecule. The replacement seems to have eaten into the dolomite mass in spherical form, entering through capillary fractures. The center of the sphere runs high in zinc, fading to zero on the circumference. At the New Prospect neither iron pyrites nor galena appears in the replacements.

At Straight Creek the orebody seems also to lie along a fault plane, but there seems to be but a single fissure at this mine. As at New Prospect, the dolomites carry the orebody, which lies at the base of the Knox and is in contact with the "ribbon" limestones. The Conasagua shales

are close by but not in visible contact with the ore. The entire orebody seems here to be a replacement and the blende is fine-grained and intimately mixed with fine crystals of iron pyrites and galena.

At the Caldwell mine, about 3 miles (4.82 km.) above the New Prospect, on the surface the ore appears exclusively in fissures from ⅛ in. to 20 in. (3.17 to 508 mm.) thick. Here blende, galena, and iron pyrites occur intimately mixed in fine grains and masses, the vein filling occasionally running upward of 90 per cent. combined sulphides. By diamond drills these fissures were followed down to a depth of 1200 ft. (365.76 m.) below the level of the Powell River. As long as the cores showed fissures, the mineralization continued as above. At about 1100 ft. (335.28 m.), one drill core showed 1 ft. (0.3 m.) of nearly solid blende with no galena or iron pyrites. The core showed the ore as a replacement of horizontal strata of calciferous Conasagua shale. Three drill holes showed similar cores in the same kind of rock but there was a difference of about 100 ft. (30.48 m.) in elevation. These facts seemed to point to faulting and displacement along the fissures but no signs of this were to be noted on the surface. The broken line in Fig. 3 shows the contour of the Conasagua shales, as revealed by diamond drills along a line of 45,600 ft. (13,900 m.). The contour may be faulted instead of corrugated by folding, as indicated by the figure.

In the Austinville mine, generalized mineralization is shown in Fig. 4. The mineralization shown in strata I, II, III and IV has been proved to extend vertically for 435 ft. (132.58 m.) by a shaft (see also *a–b–c*, Fig. 1). From the surface to the 235-ft. (71.63-m.) level several strata of oxidized ores have been worked continuously and drifts from the 335-ft. level and ore at the bottom of the 435-ft. shaft line up with these strata on the surface. Mineralization occurs on distinct bedding planes (see Fig. 4, *b–b*); on jointing and fracture planes (*c–c*) and, though not shown in the figure, whole strata seem to have been replaced in brecciated, highly crystalline bodies. The orebody, aside from the gangue, is made up of sphalerite, galena and iron pyrites. The vein filling aside from the metallics is mainly calcite, dolomite crystals and, very rarely, scattering crystals of purple fluorite. The sphalerite and galena occur in the ratio of about 3 to 1, are fairly coarsely crystalline in the main. The iron pyrites occur both as fine-grained crystals with the blende and galena, frequently in great masses entirely replacing the other metals. In these masses lead and zinc runs from zero through traces up to an economic per cent.

As at New Prospect, when the blende and galena occur along bedding, joint, and fissure planes, both minerals are coarse-grained. In general, where the limestone is replaced the mineralization is fine-grained. In places in the mine, a large part of the galena occurs in dense masses of highly crystalline, secondary, dolomite in filaments after the manner

of mushroom spawn in the "brick seed" of the market. In the Forney and Ivanhoe deposits in Virginia and in the Friedensville mines in Pennsylvania, minerals are about as described in the cited cases. The main differences between the minerals of the eastern deposits described and those of the "Valley" States seems to lie in this. In the "Valley" States the iron is mainly marcasite and on crystals of both blende and galena there are often implanted crystals of tetrahedral copper.

Barite occurs more frequently in zinc fissure deposits without eruptives than in fissures where eruptives are present. Near Rhodelia in Union County, Tennessee, there is an outcrop where zinc in the form of blende is 15 per cent. In the gangue, more than 50 per cent. is barite. It is a fairly prominent gangue mineral in the White Pine belt that reaches northeast from White Pine to Mosheim. On the Mascot fissure, beginning near Kiser, barite is found with zinc blende as far south as Sweetwater. About here the barite becomes the dominant mineral to the practical exclusion of zinc.

Barite is not infrequently a gangue mineral in some of the zinc deposits in the Ozark region, Missouri. It has been mined in Wythe County, Virginia, associated with galena and sphalerite, but, so far, no commercial deposit of zinc has more than traces of barite. In the Kelly district, New Mexico, barite is found with zinc and lead in considerable abundance above the Kelly-Graphic fault only.

Minerals Accompanying Fissures in Eruptives

The Butte and Superior and other zinciferous veins in Butte, Mont., seem to carry silver sulphides in addition to the silver in the galena. According to Walter Harvey Weed:[2] "The minerals of the Butte ores are neither numerous nor rare, nor of great variety." Chalcocite, bornite, enargite are the most common copper minerals. Tetrahedrite, pyrite, native gold and silver, marcasite, sphalerite, galena, pyrargyrite, hubnerite, rhodonite, rhodochrosite, silica, crystalline and colloidal, occur more or less abundantly.

In the Silver Pick mine near Mt. Wilson, Colo., a narrow vein carried gold up to 5 oz. or more per ton, silver up to 18 oz. in associated minerals, galena, chalcopyrite, stibnite, tetrahedrite, iron pyrites, and a very black sphalerite. The gangue mineral was mainly quartz. When sphalerite and pyrite began to replace galena and chalcopyrite to any great extent the gold and silver vanished. Zinc in this part of the veins was 20 to 40 per cent. The vein was in a great diorite dike.

In the Alotepec mines in Guatemala, zinciferous fissures in rhyolite dikes carry, in addition to a pure blende, high-grade argentiferous galena

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[2] Geology and Ore Deposits of the Butte District, Montana, *U. S. Geological Survey, Professional Paper* 74 (1912), 72.

and chalcopyrite. No other minerals were observed save the gangue, consisting mainly of calcite.

In Madison, N. H., a large vein in granite carries coarse blende, argentiferous galena, and some iron pyrites. The gangue is mainly epidote, derived apparently from metamorphosed feldspar. No other minerals were observed.

Zinc-bearing Fissures in Limestones and Other Rocks, Accompanied by Eruptives

From an economic standpoint, silver is the dominant economic element in the above class of orebodies. Up to 1901 the presence of zinc sulphide in ores of this class prevented the operation of many of these mines. The silver content of the galena concentrates from these ores sometimes ran as high as 1500 oz. per ton. In many cases sphalerite as well as galena is argentiferous. The presence of silver in these ores is in sharp contrast to the lack of silver in fissures unaccompanied by eruptives. In southeast Missouri the silver content averages about ¾ oz. per ton of pig metal. The galena in the zinc fields of the Joplin region and in Wisconsin has about the same silver tenor. The visible difference between the above classes of deposits and fissures without eruptives is equally striking. Metamorphic minerals, such as wollastonite, zoisite, hornblende, garnet, epidote, and scheelite, may be present. Hematite is an abundant mineral at Hanover and Kelly (in the Graphic mine only), N. M.; Fierro, Nuevo Leon and Las Charcas, San Luis Potosi and also at the Calera mine in Sonora, Mex. Vanadinite, descloizite, wulfenite, and pyromorphite are among the lead minerals; rhodonite and rhodochrosite are two manganiferous minerals; hubnerite is found in Butte, Mont., mines, in mines in the Silverton district and in the vicinity of the Poland mines in Arizona. Tetrahedrite and stibnite are both very common antimony minerals; also many copper and silver sulphides, antimonides, and arsenides. Native gold is found certainly in sphalerite from the Tomboy mines near Telluride and from the Gold Prince and Sunnyside near Silverton, Colo.; it has also been found intergrown with galena and arsenopyrite at many other zinciferous mines in Colorado.

Contact Zinc Deposits

I refer under this heading to contacts with eruptives. In general, where mineralization occurs at or near contacts of sedimentaries with eruptives, minerals described under the preceding head *may* be found, but are not necessarily. As pointed out under "Contact Metamorphism," aside from occasional "baking" effects, neither metalliferous mineralization nor metamorphic minerals are the result of simple contact. Profound fissuring or faulting seems to be absolutely necessary to accomplish this.

Secondary Enrichment of Zinc Deposits

This rather overworked term seems to be well covered by "aggregation" or even by "segregation." Leaving it at its face value so far as zinc deposits are concerned, evidence gathered from a wide range of observation seems to point to subtraction rather than to addition, in many if not in all cases, in accounting for rich deposits of oxidized ores. The Bertha mines in Wythe County, Virginia, are excellent examples of the point in question. The surface of the worked deposit was originally covered by a thick mantle of clay. Mining operations disclosed pinnacles or chimneys of limestone as high as 100 ft. (30.48 m.). Some of these pinnacles were coated with oxidized ores and the "valleys" were occasionally floored with massive deposits from 5 to 40 ft. (1.52 to 12.19 m.) thick. In addition, there were layers of soft and hard "buckfat" carrying from 15 to 17 per cent. of zinc.

The chimneys, while showing more or less blende, were nowhere, en masse, of economic value. In no place was there blende sufficient to account for the probable average of 15 per cent. zinc in the valleys between the pinnacles. The conclusion would seem to be warranted that the rich mass of oxidized ore had been formed by aggregation from the blende-bearing limestones. Yet examination shows that the chimneys and floors are dense, compact dolomites with no evidence of channels, even capillary, through which the solutions might have traveled. This would leave the only source of enrichment to overlying blende-bearing dolomites. But on all sides of the old workings the surrounding dolomites are wholly barren of zinc. In short, there are, at present, no visible channels or sources from which or through which enriching solutions could come. Taking all data now observable into consideration, the conclusion seems to be inevitable that the rich oxidized Bertha deposits were formed *in situ* by the subtraction of the soluble dolomites in which the zinc orebody was originally located in the form of sulphides.

This conclusion seems to be fortified almost to complete demonstration at the Austinville mines about 8 miles (13 km.) to the southwest. These mines were originally worked for oxidized lead exclusively, later for both lead and zinc sulphides. The outcrop of these mines is about 235 ft. (72 m.) above the base level of drainage—the New River (see Fig. 1). Mining operations here disclosed, though in a lesser degree, the same chimney formation as at Bertha. The oxidized ore was of the same character as well. The clays in places were 90 ft. (27.5 m.) thick. Instead, however, of the ores being confined to chimney coatings and floorings, at least six well-defined blankets of oxidized ores were followed, not continuously, for at least 6000 ft. (1830 m.) on strike and were followed down on the dip to the level of the river. At this level the original sulphides appeared to the exclusion of the oxidized ores. In other

words, in repeated instances, oxidized ores were followed down to the original sulphides and these, in turn, have been followed down to the 435-ft. (132.6-m.) level, 200 ft. (61 m.) below the base level of drainage (see Fig. 1). The combined lead and zinc averages about 14 per cent., large masses running as high as 30 per cent. Here there seems to be no possible room for doubt but that the rich oxidized masses of ore, averaging upward of 40 per cent., were formed by subtraction of the dolomite, not by the extraneous addition of zinc and lead.

Another striking example is to be found at the New Prospect mine in Union County, Tennessee. This mine (see Fig. 2) can be represented by a prism about 900 by 50 by 90 ft. (274 by 15 by 27 m.). For 600 ft. (183 m.) the western end of the prism consisted wholly of oxidized ores. Six hundred feet to the east the face of the open cut is about 90 ft. high. The bottom 20 ft. of the mine is shattered dolomite running about 20 per cent. combined zinc and lead. For 30 ft. above this shattered zone the combined metals will average about 10 per cent. Here there seems to be no possible room for doubt that the rich oxidized ores were derived wholly from the decomposition of the mass in place, the result being that the soluble dolomite was, in the main, leached out, leaving the oxidized lead and zinc. Again, this is an example of subtraction, not addition. The same conclusion seems to be inevitable at the Straight Creek, Mascot, in Tennessee; the Hanover, Kelly, and Cleveland in New Mexico; the San Xavier in Arizona; the Leadville and Gilman veins in Colorado; the Tiro General, La Bufa, Cruz del Aire, and other mines in Mexico and the United States. In Guatemala and San Salvador are mines showing the same characteristics. In each of the mines cited, rich oxidized ores have been followed down to correspondingly rich sulphides. The Bertha and Delton mines in Virginia, the Embreeville, New Market, and Mossy Creek mines in Tennessee, show no commercial zinc sulphides below the oxidized surface ores even with fairly extensive drilling.

Some mines in Wisconsin and in the Joplin field (in its most comprehensive meaning) seem to be examples where the subtraction has been carried to the point of exhausting the zinc and lead as well. In these instances, where masses of stained clay evidence the work of meteoric waters, secondary enrichment ought to have followed. On the contrary, however, these waters have removed not only the limestones but most of the zinc as well.

There is a peculiar mine near Santa Eulalia, Chihuahua, Mex. A shaft about 1100 ft. (335 m.) deep was sunk through a rotted rock of undetermined origin; then came about 200 ft. (61 m.) of oxidized ores, then about 200 ft. of zinc and lead and other sulphides, and finally, oxidized ores again. In the zinc mines of Alotepec, Guatemala, there are great dike-like masses of garnet in which zinc and lead run as high as

16 per cent. The galena and blende often come to the surface. In places, however, the blende is decomposed and the cavities are filled more or less completely with zinc oxide. The mine outcrops for about 2½ miles (4 km.) along the Rio de las Minas. The greater part of the ore zone has a hornblende gangue instead of garnet. The hornblende is badly decomposed but carries zinc oxides from 3 to 11 per cent. zinc. When undecomposed hornblende is found, zinc in the form of sulphide is found in the same quantity. In the San Juan de Calera mine in San Salvador, zinc and lead are 39 per cent., mainly as a sulphide. The gangue is garnet and occasionally, as at the Alotepec mines, cavities in the garnet are filled with zinc oxide.

In this same district is El Tejada mine. No blende was observed here but loose sheets of high-grade calamine were seen *in situ*. These sheets were observed in both jointing and bedding planes flanked with rotted, granular masses of white limestone. The rotted planes were from a few inches to 5 ft. (1.52 m.) thick. These were succeeded by blocks of fresh limestone. Observed faces resembled masonry walls, the calamine representing the mortar.

At Hanover, N. M., oxidized ores with rotted hornblende abut in places against solid garnet carrying upward of 5 per cent. zinc as blende. The oxidized ores also occur in jointing and bedding planes.

Without exception, where oxidized ores have been worked underground, the cavities are not entirely filled with clay and ore but it is impossible to calculate the relative proportions of original limestone and zinc. An estimate would lead one to conclude that, were the cavity refilled with the original limestone with the corresponding zinc as blende, it would run about 15 per cent.

In the Bajan district in Mexico, cavities from which zinc carbonate was being mined were 1500 to 3000 ft. (457 to 610 m.) down a slope of 20°.

Depth at Which Zinc and Lead Deposits Have Been Found

In *Bulletin* 606, *U. S. Geological Survey*, p. 202 *et seq.*, Siebenthal says that at Stone City, Kan., zinc ore is reported in a well at a depth of 591 ft. (180.14 m.); at Pittsburg, Kan., at the same depth; in the Joplin district in wells at depths between 300 and 800 ft. (91 and 244 m.); near Rock Island, Ill., at more than 700 ft. (213 m.) and at Peoria at a depth of 699 ft. (213 m.). Siebenthal seems to think that artesian circulation might account for these ores, some of which would be economic if at a shallower depth.

While I can only be sceptical regarding it, changing topography might make such an origin possible, but the ores would then have originated many years ago. I call attention to the following localities where no

reconstruction of ancient topography could possibly account for their origin.

At Austinville, Va., the New River is the base level of drainage and this river flows on the eroded crest of an anticline. As seen in this section (see Fig. 1) the outcropping ribbon limestones on the river are more or less mineralized with zinc and lead sulphides. These rocks dip to the south at an angle of about 45°. The drill hole is located only a few feet above the river level and 1000 ft. (305 m.) south at the foot of the bluffs. At a depth of 750 ft. (228.6 m.) pyrite was cut with traces of zinc and lead. The core shows 10 to 80 per cent. pyrite. At a depth of 1000 ft. another band of pyrite was cut. North and south of the river, ribbon limestone and shales outcrop, dipping respectively north and south. As has already been stated, the block of nearly pure dolomite between these rocks can be accounted for in no other way than by a fault with a throw of something like 3500 ft. (1066.8 m.). If the ore was deposited subsequently to the faulting, and this seems certain, artesian circulation could hardly account for the mineral deposit. Even supposing that the folding and faulting occurred simultaneously with the great Appalachian folds, the crest of the restored anticline would have been at least 4000 ft. (1219 m.) above the present river level and in order to have deposited where they now are, their point of origin must have been far east, at least 50 miles (80.47 km.), and there can be no certainty that at this eastern locus the elevation was above the crest of the New River anticline.

The Mascot mine in Tennessee has the Holston for base level of drainage. The deposit of zinc here goes far below the level of the river. In this mine there can be absolutely no question that the deposit was formed subsequently to the faulting and brecciation. Here it is quite impossible to reconstruct a topography that would allow artesian origin of the ore.

In Claiborn and Union Counties, Tennessee, the Powell and Clinch Rivers are the base level of drainage. The Powell, at least, flows on the eroded crest of an anticline. West of the river, the Knox formation dips rather steeply to the foot of the Cumberland Mountains; east, the dip is gentle as far as Tazewell where the Conasagua shales outcrop. At the Caldwell mine (see Fig. 3) blende was cut in the shales 1100 ft. (335 m.) below the level of the river. Blende was also cut in adjacent holes at depths of 900 and 1000 ft. (274 and 305 m.).

About 15 miles (24 km.) below Austinville, Va., is a partially developed property known as the Forney mine. The New River here flows between bluffs about 250 ft. (76 m.) high. Some zinc and lead has been mined in these bluffs in the bedding planes. These planes can be traced to the level of the river. In the river bottom, shafts have been sunk to bed rock to the river level. In these shafts zinc and lead occur

in what seem to be rich layers between the bedding planes. Along these planes the limestones seem to be more or less brecciated. These mineralized planes have been traced clear across the river. The rocks dip steeply. The river cuts across the strike of the rocks. A drill hole is said to have been put down in the river bed at a low stage of the water and is reported to have shown ore to the bottom.

At Friedensville, Pa., the zinc ore occurs in steeply dipping rocks that have the appearance of anticlines. The surface of the mine is only about 400 ft. (122 m.) above tide.

At Cape Rozier, on Deer Isle, Maine, zinc and lead sulphides are found over 100 ft. (30.48 m.) below sea level. It might be added here that the ore cut in the Caldwell drill holes was 100 ft. below tide.

To sum up: the entire zinc-lead area in eastern Tennessee, southwest Virginia and in Pennsylvania is so broken and faulted that it seemingly precludes the idea of artesian water. Even the reconstructed topography fails to show reasonable grounds for artesian water in the past. Moreover, leaving out of consideration the bearing of previous topography on artesian water, the fact remains that the greater number of known deposits in the localities above mentioned are, in effect and reality, "true fissure veins" with all that this term connotes.

Conclusions

In the body of this report it has been pointed out that in zinc veins accompanied by eruptives the main characteristics are the presence of high silver values in the galena and, in many instances, in the blende, and the presence of comparatively rare minerals—vanadinite, pyromorphite, wulfenite, copper and silver sulphides, antimony, and other minerals of this class—and of metamorphic minerals such as garnet, hornblende, epidote, wollastonite, etc. It was further pointed out that at Charcas, Hanover, the Alotepec and San Salvador mines, the simple presence of eruptive dikes produced none of these effects. In fact, in many instances, notably at Hanover (Fig. 6), a dike cuts through both the blende and the metamorphic garnet. In Fig. 7, also at Hanover, the blende occurs between the limestone and garnet with no dikes whatever.

Another coincidence, not pointed out, in the cases of Alotepec, San Juan de Calera, Charcas, Hanover, and notably at the copper mines in the San Carlos Mountains, Tamaulipas, Mex., the limestones occur mainly as great islands floating in a sea of basal porphyry. In these localities hornblende, garnet, and epidote and other metamorphic minerals appear in great and persistent dike-like masses. It has forcibly occurred to me that such great masses of porphyry would, for ages, retain high temperatures in their interior and that earth movements, such as would result in dikes, would also open channels through which heated, mag-

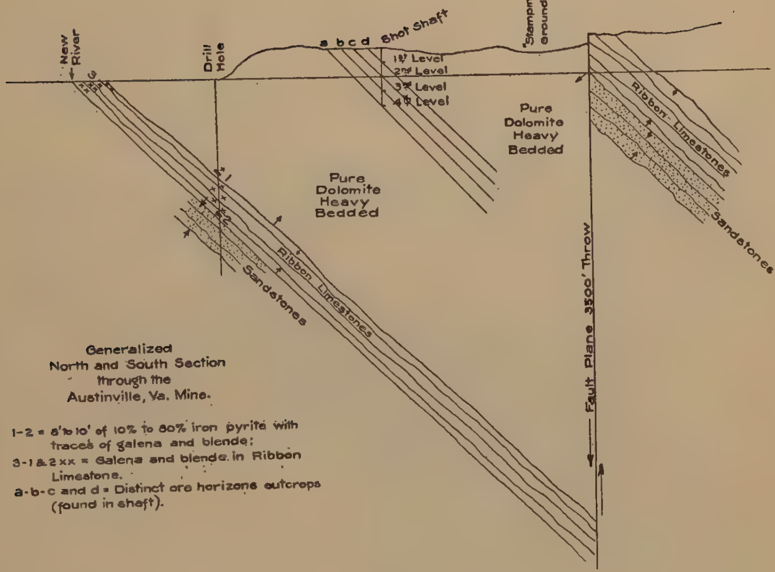

New River
Drill Hole
a b c d Shot Shaft
1st Level
2nd Level
3rd Level
4th Level
"Stamping Ground"
Pure Dolomite Heavy Bedded
Pure Dolomite Heavy Bedded
Ribbon Limestones
Sandstones
Ribbon Limestones
Sandstones
Fault Plane 3500' Throw
Generalized North and South Section through the Austinville, Va. Mine.
1-2 = 8' to 10' of 10% to 80% iron pyrite with traces of galena and blende;
3-1 & 2 xx = Galena and blende in Ribbon Limestone.
a-b-c and d = Distinct ore horizons outcrops (found in shaft).

FIG. 1.

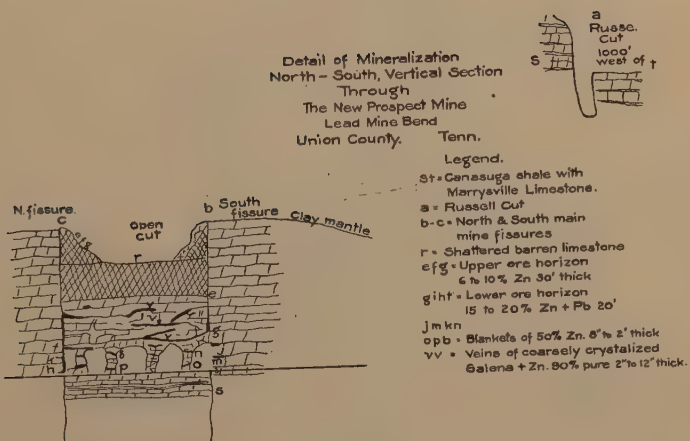
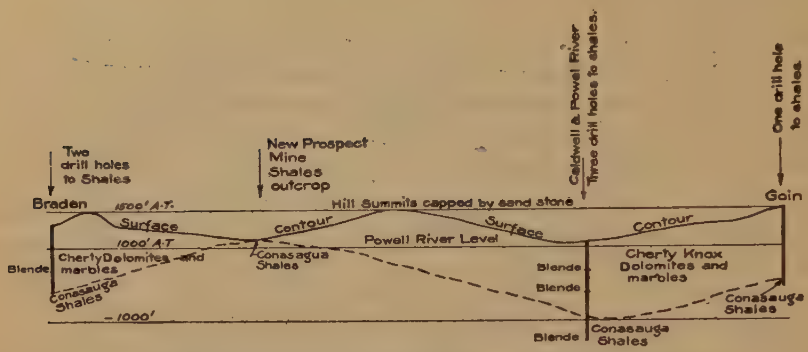

Detail of Mineralization North–South, Vertical Section Through The New Prospect Mine Lead Mine Bend Union County. Tenn.
a Russe. Cut 1000' west of t
Legend.
St = Canasuga shale with Marrysville Limestone.
a = Russell Cut
b-c = North & South main mine fissures
r = Shattered barren limestone
efg = Upper ore horizon 6 to 10% Zn 30' thick
giht = Lower ore horizon 15 to 20% Zn + Pb 20'
jmkn opb = Blankets of 50% Zn. 8" to 2' thick
vv = Veins of coarsely crystalized Galena + Zn. 80% pure 2" to 12" thick.
N. fissure. c
Open cut
b South fissure
Clay mantle

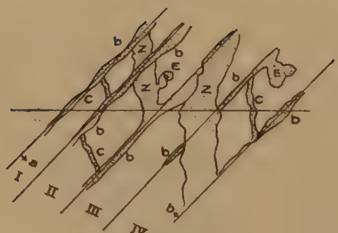
FIG. 2.



Vertical Section from Braden to Eli Goin Prospects Union and Claiborne Counties Tenn. N 65° E. — Showing relation of Conasauga shales to surface as proved by drill holes.

NOTE:- The shales are the ore horizons at New Prospect and Straight Creek Mines.

FIG. 3.



Detail of Mineralization
on
Bedding Planes, especially b-b-b
In cross fractures, C-C-C
In crushed zones, Z-Z-Z.
In etched Cavities, E-E
b-b-b ¼" to 4" thick heavy PbS,
some ZnS secondary gangue.
as above in C-C
In Z-Z and E-E, gangue as above
main mineral ZnS; sometimes
barren, at others, rich.

FIG. 4.

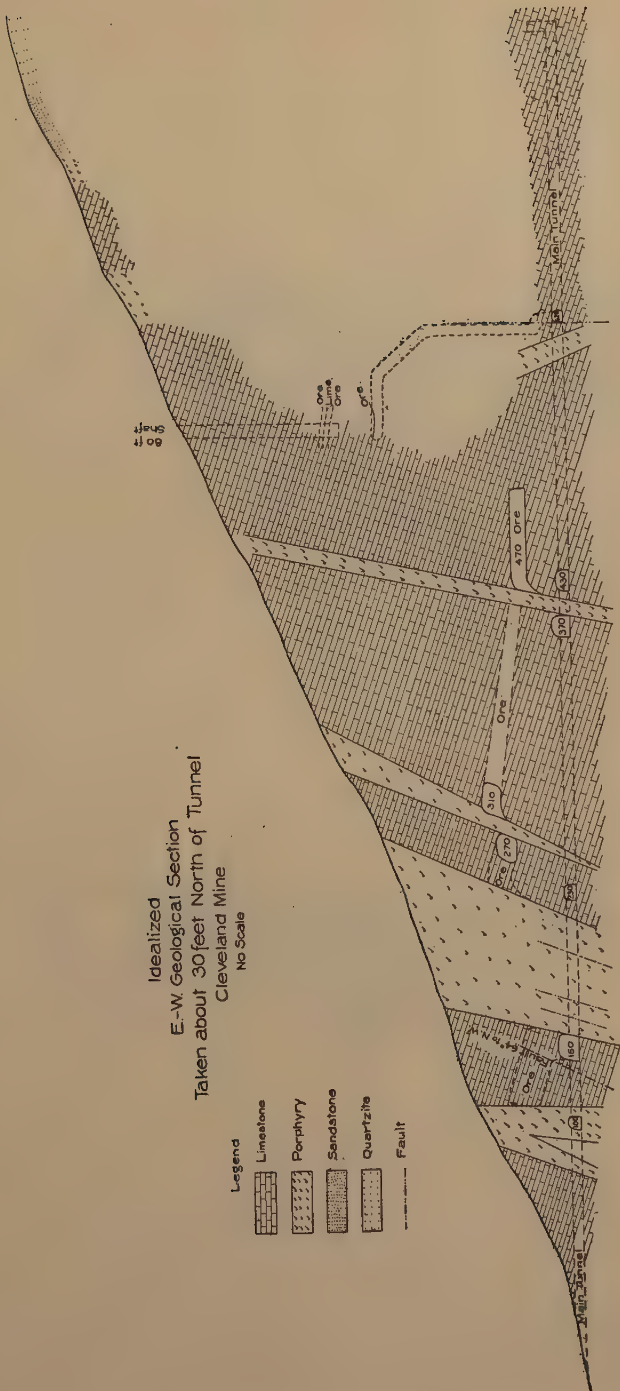
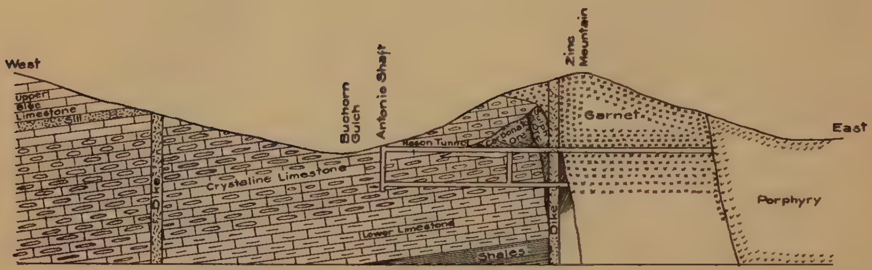

Idealized
E.-W. Geological Section
Taken about 30 feet North of Tunnel
Cleveland Mine
No Scale
Legend
Limestone
Porphyry
Sandstone
Quartzite
Fault
80 ft
Shaft
Ore
Lime
Ore
Ore
Main Tunnel
470 Ore
430
370
Ore
310
270
Ore
160
Ore
Main Tunnel

FIG. 5.



Section through
Nason Tunnel Workings.
The Empire Zinc Co. - Hanover N. M.
Not to Scale.

FIG. 6.

Section
Thunderbolt Mine
The Empire Zinc Co.- Hanover, N. Mex.
No Scale

FIG. 7.

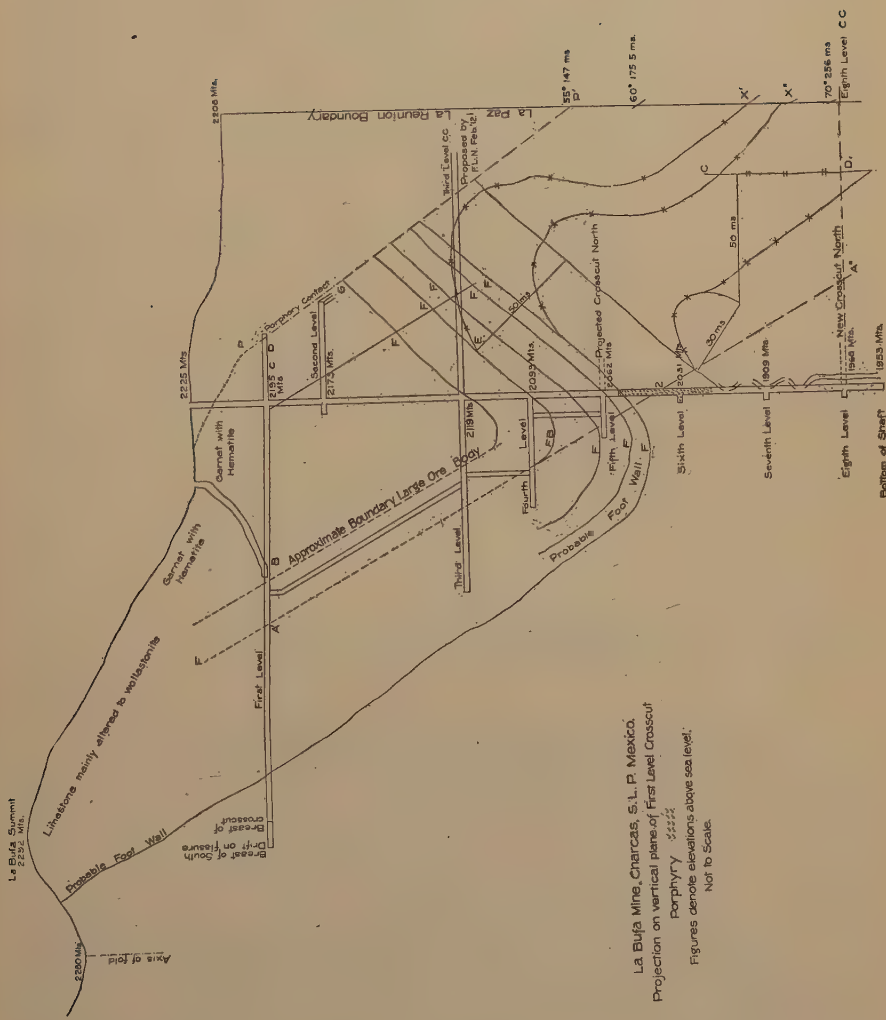

La Bufa Summit 2292 Mts.
Limestone mainly altered to wollastonite
Probable Foot Wall
2280 Mts.
Axis of fold
Breast of South Drift on fissure
Breast of crosscut
First Level
Garnet with Hematite
Garnet with Hematite
2225 Mts
2208 Mts.
La Reunion Boundary
La Paz
Porphyry Contact
Second Level
2175 Mts.
2195 Mts
Approximate Boundary Large Ore Body
Third Level
Third Level C.C.
Proposed by F.L.N. Feb.'12.
2119 Mts.
Fourth Level
2093 Mts.
Fifth Level
2062 Mts
Projected Crosscut North
2031 Mts
Sixth Level
1909 Mts
Seventh Level
New Crosscut North
1853 Mts.
Eighth Level
Bottom of Shaft
Eighth Level C C
55° 147 ms
60° 175 5 ms.
70° 256 ms
50 ms
30 ms
50 ms
Probable Foot Wall
La Bufa Mine, Charcas, S. L. P. Mexico.
Projection on vertical plane of First Level Crosscut
Porphyry
Figures denote elevations above sea level.
Not to Scale.

FIG. 8.

matic waters would rise and manifest their presence by the intense metamorphism noted. It is to be noted that dikes do not always produce open channels and lack of metamorphism accompanying these dikes seems only to lend strength to the assumption that fissuring may or may not produce metamorphism, depending wholly on whether the fissures open channels to highly heated masses lying deep. Whether such heated masses exist or not, in a given locality, is, perhaps, not always demonstrable, but these are at least coincidences. In the localities named, great masses of porphyry do exist and I do not now recall a locality that

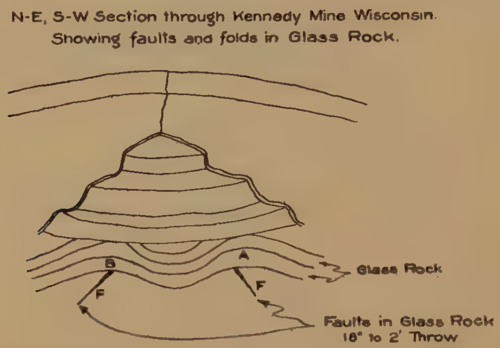


FIG 9.

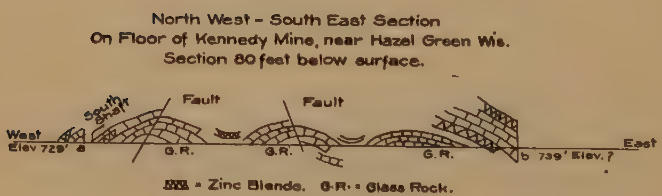


FIG. 10.

would present an exception. However, this suggestion is an inference, and cannot, at present, take rank as a demonstrated conclusion. Whatever rank is assigned to it by others, I regard the facts as of academic rather than of economic importance.

If, however, the other observations which I have recorded find acceptance, they have a most vital economic bearing. These are the universal accompaniment of fissures with zinc-ore deposits whether accompanied by eruptives or not. This fact puts even the zinc deposits of the Valley and the Appalachian States in the class of fissure veins. It follows from this that there is no reason why these fissures should not reach the productive depths of the deepest worked fissures in the United States, at

Butte. Here economic fissures have been and are being worked at a depth of over 3000 ft. (914 m.). It promises to put the great zinc fields of the Mississippi valley into great permanent camps instead of an ephemeral field of shallow diggings with a comparatively small individual volume of ore. The fissure character of all of our zinc deposits I regard as a demonstrated conclusion, not as a plausible inference.

Discussion

H. A. Buehler, Rolla, Mo.—I consider the lead and zinc deposits of the Mississippi Valley to be the result of descending waters. There are many features that we have not determined and yet when you see the general field relations, it seems to me that the cause and effect can be well explained by descending circulation. In the Joplin-Miami district, for instance, the ore deposits are connected with openings or with old cavity fillings which have a direct connection downward from the surface to practically 300 ft. In the Miami camp, in the western part of the field, some of the deepest orebodies are directly connected with water channels; the same condition has been obvious and always present with the shallower ores. I speak of one deposit that I have examined more thoroughly than any other in that district, the Admiralty property, which contains large openings or caves at the deepest levels.

It seems to me that waters which would pass upward through 1800 ft. of dolomite would be saturated with lime and magnesia so far as the carbon dioxide of those waters would dissolve them, and therefore, on reaching upper levels, I do not see how they should suddenly form large cavities by additional solution. That is one of the features that I consider favorable to the descending theory.

The other feature is the black flint that is connected with the ore. If the black color is due to organic material, as it may or may not be, I do not see the source of that organic material in anything but carboniferous shales that are directly above it. I recognize that, to make an interesting discussion of the matter of descending solutions, we ought to have a diagram showing the different formations and the different members of the Boone formation in which the ores occur, and that without it you cannot make a good presentation of the geology of the orebodies. While Dr. Nason believes that the deposits of lead and zinc are the result of waters that have come from an igneous origin, in this district we do not know of any igneous rocks above a depth of 1800 ft., and those constitute the old pre-Cambrian igneous surface that resulted from an erosion interval which lasted at least up to middle-Cambrian time; in other words, the rocks were very old and cold.

There is the argument that emanations from those rocks due to faulting might have sent up waters containing the minerals. In that

case they have had to pass through 1800 ft. of dolomites and sandstones, and I do not believe that we could find water of that character coming from an igneous source, and rising through that extent of formations, without finding a certain amount of metamorphism either in the drill holes or in the area where we find these rocks outcropping in the Ozark hills. We do not find, in any of the older rocks, any metamorphism that would indicate waters from an igneous source; hence, I do not believe that the ascending deep water came from an igneous origin.

J. T. Boyd, Redcliff, Colo. (written discussion*).—If I understand Mr. Nason correctly, he reasons that the zinc deposits of the Valley and of the Appalachian States are true fissures. Therefore, being true fissures, there is no reason why these deposits should not reach the productive depths of the deepest worked fissures in the United States, namely, at Butte.

Now it has always seemed to me that there is such wide difference in the character of fissure veins that they can hardly be compared; that is, although you may have fissures of several classes which extend to great depths, the chance for economic orebodies with depth depends very largely on the class of the fissure. The two vital factors in the formation of orebodies are:

First.—A condition that admits of free circulation of magmatic or other underground waters.

Second.—A condition that permits the free deposition of the salts of the heavier metals from such waters.

Other factors which have a bearing on deposition are:

Pressure and temperature of solutions.

Wall rocks favorable for replacement.

Wall rocks precipitating certain salts.

Brecciated or open condition of rocks through which solutions pass.

Weakness along certain strata or bedding planes.

Mechanical damming of solutions.

Contact of magmatic waters with surface waters.

Therefore, in looking for the continuation of economic orebodies whether of zinc or other metals, you do not necessarily look for the continuation of a fissure or for the continuation of a mass of eruptive rock; you look for the continuation of a certain set of conditions which has made the orebody possible. Let us take some specific illustrations.

The rock formation in the San Juan district, in southwestern Colorado, consists of a number of flows of andesite breccia. These flows are 3000 to 5000 ft. (914 to 1520 m.) total thickness and are capped at points with rhyolite. The rhyolite flows are for the most part eroded. Analy-

* Received Oct. 30, 1917.

ses have shown these breccia flows to be comparatively rich in basic metals. In cooling, these flows developed contraction fissures or simple straight fractures all the way from the surface down to the sedimentaries on which they rest. These fissures were filled with vein matter of which quartz, pyrite, rhodonite, sphalerite, galena, and calcite are the most abundant in the order named.

This filling is evidently the result of direct magmatic segregation from the breccia flows and presents an excellent illustration of that action. Several of these veins were reopened later by faulting, and there was considerable metamorphic action; Mr. Nason mentions one instance, the Sunnyside-Gold Prince.

I have never seen any other district where underground conditions can be estimated so far ahead of development. To begin with, there is little vegetation at this high altitude and the outcrops of these strong fractures can be followed sometimes for several miles. Where the outcrop is narrow, we can expect with certainty the same condition in the vein underground. When the outcrop is iron-stained and widens out, we may predict with assurance the occurrence of wide sulphide stopes extending many hundred feet under these points.

If a tunnel is driven at, say, 11,500 ft. (3505 m.) altitude on a vein which shows fair gold content at that altitude, we are reasonably certain that at 1000 ft. (304 m.) above the tunnel we may look for high-grade ore, as at that elevation the vein passes through a breccia flow where the wall rock has a marked effect on precipitation and where there will be decided lateral enrichment. Ore extension in such fissures can be predicted and even estimated far ahead of development, because we know there is no change in the fundamental conditions which tend to produce that orebody.

Mr. Nason puts the Eagle mine in the class of mines in which the rich oxidized ores were formed more as a process of subtraction than of addition. That is, the decomposition of the original orebody resulted in the leaching out of the soluble minerals. I am not quite sure from this whether he believes the original ores at Eagle were derived from the dolomitic limestones or not. However, this is not our conclusion.

At the Eagle mine, we have first to consider an uptilting of the entire sedimentary strata to an angle of 12° at the time when the granite cores of Holy Cross and Gore ranges were upthrust through the sedimentary series. Probably at the same time a sheet of porphyry was thrust into the limestones which split the sedimentaries on their weakest plane, namely, a shale series at the top of the blue limestone. There may have been some fissuring at this time but we cannot trace the connection with the orebodies. At this period, there was tremendous activity of magmatic waters, which followed parallel lines of weakness into the mountain and replaced the blue limestone directly underneath the porphyry.

These ore-bearing solutions must have originated with the porphyry and must have come from depth, as the size of the orebodies is such as to dispel any idea of lateral secretion.

The original shape of these orebodies must have been flatly lenticular, 100 to 200 ft. (30 to 60 m.) in breadth and 10 to 60 ft. in thickness. There is evidence that aplite dikes were later thrust into the granite underlying the sedimentaries and created a bending stress in the sedimentary series above. These dikes did not reach the limestone, but fissured the quartzite lying between the limestone and the granite, forming deposits of siliceous gold-bearing ores in the bedding planes of the quartzite. At the same time, they caused a brecciation of portions of the orebodies and helped form pockets or rolls in the limestone which aided the collection of mineral along the lines of the primary ore already formed. The fracturing of the original orebody can be plainly seen. I believe that at this time we can recognize the primary ore as it occurred. We have at some points in the mine, especially on the lower levels, a hard glassy mixture of pyrite and ankerite (locally called siderite) carrying sphalerite and galena. This ore is massive and has every appearance of unaltered primary ore in which the pyrite and ankerite are contemporaneous. It has the following composition:

	Per Cent.		Per Cent.
Fe	38	Pb	1
S	28	CaO	3
CO_2	10	Insol	1.5
Zn	2 to 9	Mn	4

The upper part of the mine contains several million tons of non-commercial oxidized iron ore in which there were large bodies of lead carbonate and from 200,000 to 500,000 tons of iron-manganese ore. This iron-manganese ore is locally called Black Iron and large portions of it are plainly pseudomorphic after siderite or ankerite. All of these oxidized ores carry only a trace of zinc. Now if I am right in assuming that the primary ore had nearly a uniform composition, where did the zinc go? The relatively small bodies of zinc carbonate which we have found do not account for its disappearance. Where could it have gone except to enrich the sulphide bodies? This is the reasoning on which I base my idea that a large part of the sulphide body is secondary and that both oxidized and sulphide ores are built up largely by addition and not by subtraction, as Mr. Nason suggests.

The ore occurrence at the Hanover mine is interesting and might be mentioned as an instance of a large orebody of which the extension with depth cannot be definitely predicted. Here we have vertical dikes cutting sedimentaries as well as metamorphic granite. I believe that in such cases the ores are mainly to be found in the more favorable sedimentary rocks only. The fact that dikes or masses of metamorphic rock

go to a great depth does mean that the ores will extend to the same depth but that they will replace only the more favorable strata along the contact.

The occurrence of ore in the Spring Mountain range in Nevada is most unusual. Here we have a district roughly 30 miles long with a great mass of porphyry near the center, near Good Springs. The rest of the district is all sedimentary and no other intrusives can be seen. It has always seemed very probable to me that the porphyry at Good Springs was the mother of all the zinc deposits in that country. All mines except those in the vicinity of Good Springs are plainly the result of infiltrating surface waters following along a shale bed and making workable ore deposits wherever there were favorable rolls or faults which loosened up the surrounding rocks, thus forming pockets where the solutions could deposit their contents. The Potosi mine is a splendid illustration. The intrusives mentioned may have originally spread over a wide area. Later removed by erosion, their leached metallic contents found their resting place as above noted.

The Yellow Pine mine at Good Springs offers a striking contrast. Here we find a mine on the circumference of the porphyry body at its contact with the upturned dolomitic limestone. This is a different story. Here is every reason to expect large replacement bodies along the contact and a deep-seated origin of the solutions which will lead to deep extension of the orebodies. Furthermore, the only chance for deep orebodies in the district is and must be around the porphyry core. Such ore has been found in the Yellow Pine mine, and it is interesting to speculate whether any more deep orebodies will be found in other prospects around the rim.

At Butte, probably the best known examples of fissure veins occur. They are aplite dikes cutting a rock mass of granite, with accompanying fissuring and faulting. We find the same kind of dike cutting the same kind of rock to unlimited depth. We should expect the same kind of vein at depth as at near the surface, after necessary allowances have been made for secondary enrichment, difference in temperature and pressure, etc.

But can these veins formed directly by contraction and fissuring and magmatic segregation in impervious rocks be compared to veins formed by deep-seated external influences in limestone? There is this difference: fissures in sedimentaries have a much greater tendency to close after formation, due to the weight of the rock mass and the yielding nature of some of the strata, which flow out under pressure and fill up any opening. The deposits are naturally erratic and make away from the fissure following lines of weakness along bedding planes of the limestone or precipitating along favorable strata. This tendency with depth is bound to become more pronounced and the ore occurrence still more

uneven. Free circulation of solutions in deep sedimentaries does not seem to be nearly so likely as in a free fissure in eruptives. Would this not also apply to orebodies?

A dike of eruptive cutting sedimentaries has somewhat the same characteristics. You may observe it at some point where its contact with the sedimentary rock is dry as a bone and without trace of metamorphic action. Several hundred feet deeper you may find solutions from the dike freely permeating a receptive bedding plane. I have seen well-marked instances of this at Leadville. On the other hand, fissures by dikes or other influences in plutonic rock masses are more uniform and the magmatic solutions stick closely to their original source.

The importance of fundamental conditions has always impressed me very forcibly. Several times I have had options on prospects and have spent much time and eloquence in trying to persuade some plutocrat to buy them. Unfortunately, there never has seemed to be enough ore in sight to make them attractive, but the conditions for deep-seated orebodies were right. They have become great mines. A prospect with good conditions for ore extension is often a better purchase than a mine with tonnage developed, but where the story has all been told.

As to western fissure veins in eruptive rock, I expect many more large zinc mines to be developed. They may be prophesied at Butte among the many unexplored vein systems. Similar districts, such as Chloride, Arizona, should produce deep mines. Such mines will, of course, require a great deal of faith, patience, and money to find and develop, but when they are needed badly they will be found.

Frank L. Nason (written discussion*).—In his discussion of my paper, Prof. Buehler makes one statement which is directly contrary to my conclusions—"but while Nason believes that the deposits of lead and zinc are the results of waters that have come from rocks of igneous origin" this is precisely what I do not believe. Certainly, I have, for a long time, been forced to the conclusion that deep fissuring or faulting is the only prime source of mineral solutions, lead and zinc included, and that igneous and eruptive rocks are only an occasional phase of faulting and fissuring, not the cause of it; that dikes and sills bear the same relations to such disturbances as do mineral veins; in fact, that these rocks are only a phase of vein deposits. The absence, therefore, of igneous rocks in the zinc fields of the Valley States and in the valley of the Knox dolomites in Pennsylvania, Virginia, and Tennessee has no conclusive bearing on the question farther than to demonstrate the fact that deep deposits of lead and zinc can be formed without the agency of igneous rocks. This is the same as saying, what is evidently true, that faulting and fissuring on a gigantic scale can take place without, so far as is known, the existence

* Received Jan. 12, 1918.

of either dikes or sills. In these States there are certainly zinciferous veins in fissures, but no known eruptives. In Central America, in Mexico, in the United States, and in British Columbia, there are zinciferous veins both with and without eruptives. But this is to be noted; in zinc veins accompanied by eruptives, in most instances which have come under my personal observation, the dikes and sills have been fractured and these fractures are more or less filled with zinc and other metallic sulphides; in others, dikes have actually cut the deposits. Of course, the fractures may have been mineralized by downward flowing waters, thus secondarily, but to assume this to be a fact and to use it as a final argument is just as unwarranted and illogical as to assume that my position is final and conclusive.

Without going into detailed reasons against Prof. Buehler's assumptions that solution cavities in the Miami district could not be formed by the solvent action of waters already deprived of their carbon dioxide by rising through 1800 ft. of dolomite, it appears to me that he assumes either directly or indirectly that carbon dioxide is the only solvent of dolomites or limestones. Moreover, after flowing underground through limestones for 90 miles or more from the Joplin field, will an additional 1800 ft. of limestone make much difference in their solvent powers? Yet in the "Admiralty property" such caverns exist. From whence does the necessary carbon dioxide come? To ascribe the source to the reduction of zinc sulphates to sulphides with the consequent formation of the necessary carbon dioxide would nullify his arguments derived from the rising of mineral-bearing waters through his 1800 ft. of dolomite. Is it an established fact that carbon dioxide in water is the only solvent of limestones under any or all possible conditions?

In regard to the original salts of zinc and other metals now in the sulphide form, we are prone to make the general assumption that these metals were in solution as sulphates and that they have been reduced to sulphides by organic matter. This assumes, practically, that under no condition can zinc sulphide be in solution as a sulphide; it excludes the possibility that metallic zinc could ever be in solution as, for instance, gold, silver, platinum, and other native metals. Supposing zinc to be in solution as a metal, could it not be directly precipitated as a sulphide? It may be true that, with our present limited knowledge of chemistry, we know of no other way, but does it at all follow that there is no other way?

To go back to Prof. Buehler's arguments; assume that at a point of deep origin, zinc was in solution as a sulphate. Such a solution would not, I think, attack lime or magnesian carbonates. Suppose such solutions reached the present horizon of the Miami zinc. There is organic matter in both the limestones and the shales. Might not the following reaction take place: $ZnSO_4 + 2C = ZnS + 2CO_2$? In such case, would not

zinc sulphide be at once precipitated and carbon dioxide set free to begin work on Prof. Buehler's Admiralty cavities?

In this connection, however, in what form would the accompanying lead exist? Zinc, iron, copper, and other metallic sulphates are readily soluble in water; lead sulphate is not. Solution and precipitation problems have not by any means been exhausted by my few examples; but it appears to me that there are so many possibilities that it is highly illogical to select any one explanation and to use this on which to base a positive theory. Nor will it do to base a positive theory on any number, however great, to which exceptions are evident. We are all familiar with the fable of the dying father, the seven sons, and the bundle of sticks; any one of the sticks could be broken when taken singly; together, the strongest son could not break them. But the fallacy of the fable is here: A 1-in. rope of sand has no coherency; neither has a sand dune.

Finally, metamorphism, to which Prof. Buehler refers, appeals to me as being greatly overburdened. Every mining engineer and geologist can testify that metamorphism exists where it should not, and does not exist where it should. The reason for this is perfectly obvious. We do not possess exhaustive knowledge of the causes of metamorphism. Until we reach this goal, arguments based on the existence or non-existence of metamorphism are inconclusive. This statement I regard as equally applicable to all arguments in favor of downflowing as well as of waters rising from great depths; in other words, of arguments in support of either theory. Even specific examples are inconclusive. It appeals to me very strongly that our knowledge of ore deposits is wholly synthetic; it is analytic to a very limited extent and this fact should be kept in mind.

The New Jersey Zinc Co.'s Franklin Laboratory

BY D. JENKINS, FRANKLIN FURNACE, N. J.

(St. Louis Meeting, October, 1917)

THE Franklin Laboratory was designed mainly for the analysis of the products from the two concentrating mills situated at Franklin and Sterling Hill, the most important determinations being the zinc, iron, manganese, lime, moisture, and silica contents of the ores. Within the last 4 years, however, it has been found advisable to examine most of our supplies, and therefore additional space and equipment was allotted for the examination of such substances as oils, greases, soaps, alloys, fuels, paints, explosives, and water.

The building as shown in elevation (Fig. 1) and plan (Fig. 2) consists of a main or routine laboratory, an experimental laboratory, a water

FIG. 1.—WEST ELEVATION.

analysis room, and the other accessories such as stock room, library, sampling and drying rooms, and office. The main laboratory is further divided into a balance room and zinc analysis room.

As the greater part of our determinations are volumetric, good light is a very important factor and with this in view the interior walls of the building were constructed of white enameled brick laid in Keene cement. These also have the additional advantage that they give a clean and attractive appearance to the room. The windows are large, being 76 by 44 in. and even on very dark days the light is good throughout the whole laboratory. Our zinc titration requires a constant light and as, on many days, especially in winter, the light is variable, we installed two Artificial

Daylight lamps and have found that they meet all our requirements. They also make it possible to carry out titrations at night if necessary.

On referring again to Fig. 2, one can obtain an idea of the arrangement of the desks, sinks, cases, and hoods. All of these have been designed with the idea of expediting the work as much as possible with the least confusion. The hoods shown at the center and running the whole length of the room were designed by R. M. Catlin, and as they are somewhat novel a few words in explanation of their construction may be apropos. They consist of a series of small compartments, there being in this case 50. Each compartment is about 27 in. long, 8 in. wide and 8 in. high and is

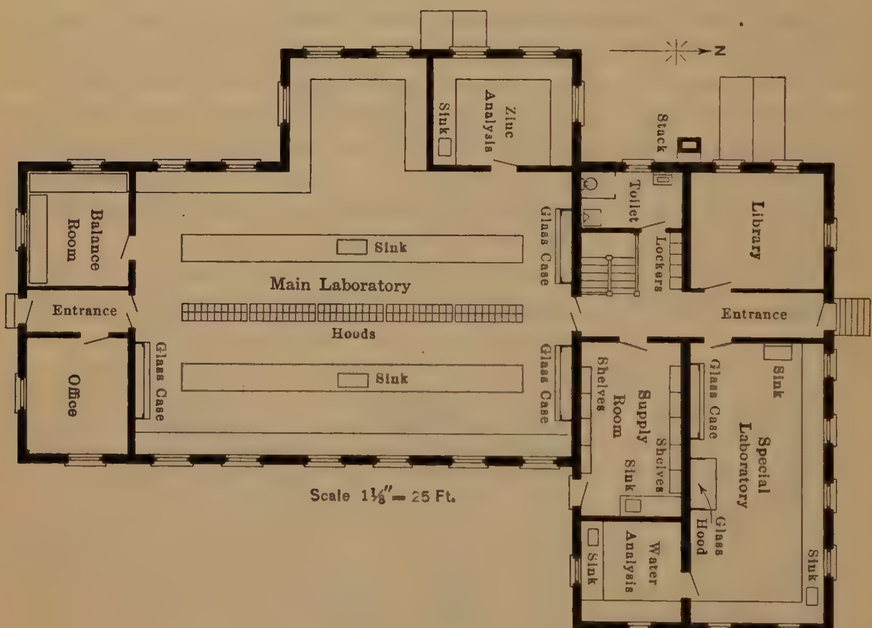


FIG. 2.—FLOOR PLAN.

made of Alberene stone. Each hood accommodates two sand baths, and all the gases arising from evaporation are taken off at the center of the hood and conducted outside of the building by means of a slight vacuum produced by an air aspirator placed in the mouth of a 6-in. Orangeburg fiber pipe, which leads from the hoods. One aspirator exhausts the gases from 10 compartments. Each compartment is supplied with two sliding doors which fit into grooves cut in the sides of the hood. By means of a notch cut in these grooves, the doors may be partly opened if necessary. Heat is furnished by means of a series of Bunsen burners placed under each sand bath. A better idea of the construction of the hoods may be obtained from the attached photograph and drawings.

In our old laboratory we had experienced great difficulty in keeping our wooden floor in good condition, so in our new building we treated the wood with aniline black and up to the present time the floors have remained in excellent condition despite the fact that in many places they have been subjected unintentionally to the action of concentrated acids.

Adjoining the main laboratory is a room set apart for zinc titration only. This arrangement has many advantages, the chief being that the atmosphere of the main room is not contaminated with ammonia fumes, as all of our zinc titrations are made in ammoniacal solutions. The ventilation of the zinc analysis room is taken care of by means of a small fan.

The balance room also adjoins the main laboratory. The slabs on which the balances rest are supported by concrete piers which do not con-

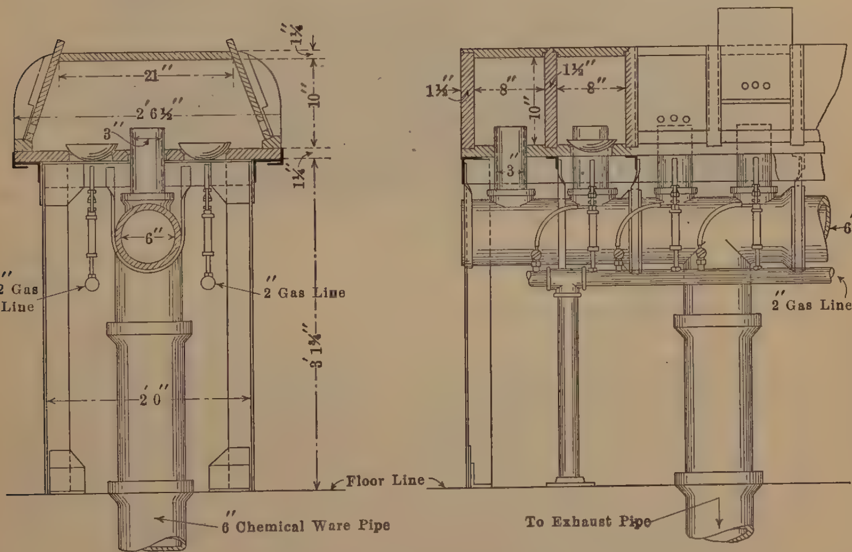


FIG. 3.—ALBERENE STONE HOODS.

nect with the building, and this removes annoyance due to jars. The room was built large enough to accommodate the filing case for the samples, which adds to the convenience of the work because the samples are readily accessible in case results are to be checked.

The experimental laboratory is provided with a hood somewhat similar to those in use in the Bureau of Standards. It is constructed of Alberene stone and wire glass. The top and sides being of glass makes the hood very light, and those who have worked in the old-fashioned wooden hoods can readily appreciate this advantage. The hood is downdraft, being exhausted by an aspirator similar to the one operating the hoods in the main laboratory.

The experimental laboratory is equipped to make any analysis that

may be required of the supplies furnished for the plant. Gas is used for heating purposes, in general, but electric hot plates or multiple-unit fur-

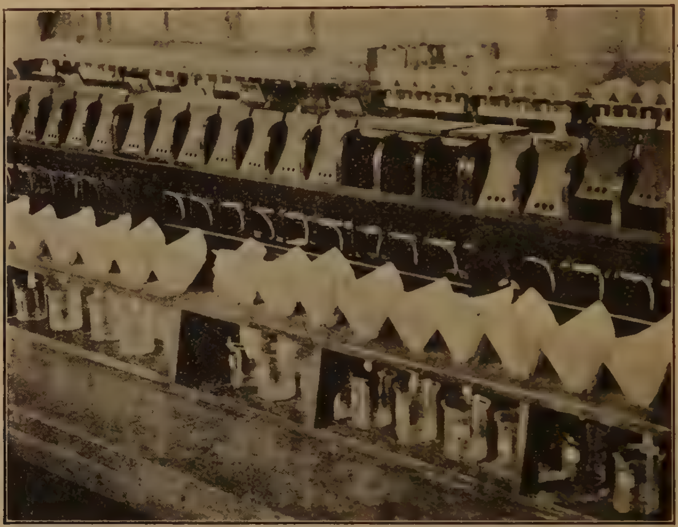


FIG. 4.

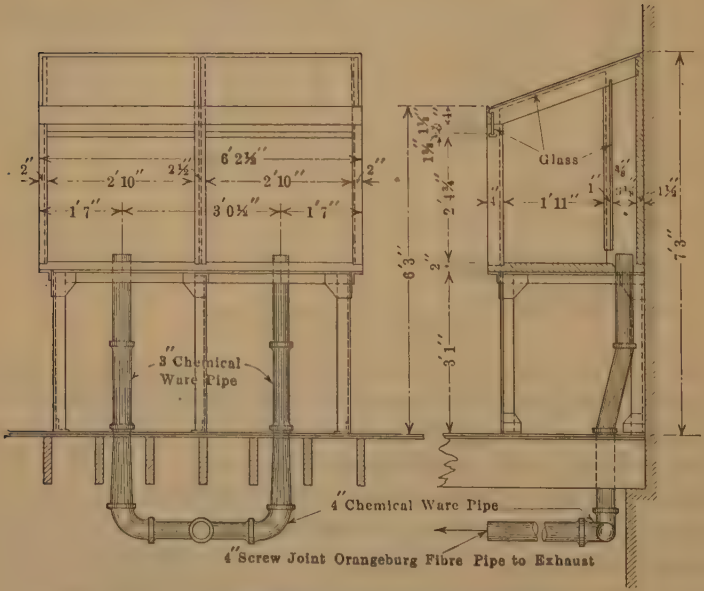


FIG. 5.—ALBERENE STONE AND GLASS HOODS.

naces are employed where it is possible to use them. For drying purposes the Freas electric ovens have been installed, and for electrolytic work a Veit apparatus has been provided.

As the supervision of the town water supply is entrusted to the laboratory, it was found advisable to fit a room for carrying out both the chemical and bacteriological examination of the water. Our equipment is of the usual nature required for such work, consisting of a sterilizer, incubator, refrigerator, and the other accessories.

This covers, in general, the principal features of our building, but before closing I would like to call attention to the size and location of our stock room, which is so situated that it is central to both the main and the experimental laboratories.

The library, also, is located so that it is easily accessible from both laboratories and is so furnished that the men can calculate and keep their records in this room and not have them distributed, as is the usual custom, throughout the whole building.

The rooms for drying and grinding of the samples are in the basement and are equipped with hot plates, bucking board, and grinder. Only the small samples are handled in these rooms, as the sampling and grinding of the large samples is carried out at the sample houses situated near the different mills. The basement also affords ample room for storage and packing.

A Barnstead still of about 5 gal. capacity per hour furnishes us with distilled water which is distributed to all parts of the building by means of block-tin pipes.

An adequate wash room is provided for the men, and individual lockers are furnished.

Although the laboratory was designed primarily for the analysis of zinc ores, we have found that it is capable of meeting any of the many problems that arise in a plant of this size.

Palmerton Zinc Refractories

BY C. P. FISKE,* PALMERTON, PA.

(St. Louis Meeting, October, 1917)

THE pottery of the New Jersey Zinc Co. (of Pa.) is equipped to make three classes of refractories; namely, spelter vessels, spelter condensers, and high-grade fire-brick. The most important of these are the spelter vessels, on account of the extreme severity of the service which they must perform, necessitating great care in the selection of materials and in the whole process of manufacture. Next to the spelter vessels in importance come the condensers, as large numbers are used in the manufacture of spelter, and improvements in quality result in material decreases in working cost. While the fire-brick which form the laboratory, or high-temperature portion of the spelter furnaces, are placed at the foot of the list, they are nevertheless of much importance, since high quality will decrease the frequency of the always expensive shutdowns for repairs.

To promote simplicity, the manufacture of these three products will be described separately in this paper, except in so far as materials and the crushing and handling system common to all are concerned.

Materials

Bond Clay.—The bond clay used entirely at the Palmerton pottery is a St. Louis clay furnished by the Grand View Fire Clay Co. It is stored in a system of covered bins, and no attempt is made to weather it in the accepted sense of the term "weathering." Some years ago our clay was weathered before pugging, but we did not find the benefit sufficient to offset the additional handling and the loss of clay incurred. Furthermore, we believe that the rotting of the pugged mix (described later) accomplishes the same results to better advantage. That there is some action taking place in the clay while stored in the covered bins, however, is evidenced by an increase in volume which at times has caused damage to the retaining walls of the bins. We have at present a stock of bond clay weathering in the open, and when this is put into service an opportunity will be afforded for making comparisons with the clay as usually handled.

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Chemical analyses are seldom made, for they have been found of little or no value as an indication of the service the clay may perform. Physical tests are frequently made, and include refractoriness at 1630° C., tensile strength and shrinkage when dried at 110° C., and tensile strength, shrinkage and porosity when burned at 1300° C. These tests are of value in determining what may be expected from the material, although at best they serve mainly to eliminate materials which the tests show would be manifestly unsuitable for vessel manufacture. The final test lies in the vessels themselves.

Grog.—The grog used is of several kinds. For some years the principal source of our grog has been calcined flint clay from the Clearfield district of Pennsylvania. We have used in the past brickbats which have seen service in spelter or other furnaces at the Palmerton plants, but in the long run we have found the use of the calcined flint to be more economical and generally satisfactory. Brickbats must be thoroughly cleaned to remove all slag and other materials that would be deleterious to the vessel life, thus not only adding expense but requiring careful and constant supervision to obtain best results. Even so, brickbats are cheaper than flint clay, but the longer life of vessels made from the latter makes it a more economical material.

During the past year or more, experiments have been made leading toward the utilization of broken saggers from concerns manufacturing pottery ware, and present indications are that they will be fairly serviceable.

All of our grog materials are physically tested at intervals, particularly the new material, in much the same manner as are our bond clays. They are mixed with a standard bond clay in the proportions used in vessel manufacturing; the tensile strength and shrinkage at 110° C. are determined for the resulting mix as well as the tensile strength, shrinkage, and porosity after burning at 1300° C. The refractoriness of the grog at 1630° C. is determined separately. Here again, we have found chemical analyses of little or no value, and they are consequently very infrequent.

Coke.—For the past year or more, pulverized coke has been a constituent of our spelter vessels. We are using chipped petroleum coke having the following typical analysis:

	Per Cent.
H_2O	0.47
Volatile	8.62
Carbon	89.00
Ash	1.83
Sulphur	1.14

The coke as received is crushed in a roll crusher to about ½-inch size and finer, and is then pulverized in a Fuller-Lehigh mill and delivered to the pottery for use.

Tests

In conducting our tests, the materials are mixed by hand, approximating the machine mixing to as great a degree as possible, tamped into a "figure eight" mold such as is used for cement testing, dried at least 1 day at atmospheric temperature, and 1 day at 110° C. One-half of the briquettes so made are then tested after the drying, and the balance are burned about 24 hr. at 1200–1300° C. A set of briquettes made from the standard materials is always carried through the tests with the new materials, so that variations in time of drying, temperature of burning, etc., are not of great importance. What we aim to do is to obtain materials or processes that give a mix superior to that in use, or to obtain equal results at less expense. This being the case, no definite limits are set for any of the properties sought, except that they must be at least equal to those of the standard mix.

Tables 1 and 2 give typical data obtained from tests of some new clays.

TABLE 1.—*Tests of Straight Clay*

	A	B	C	Standard Grand View
Cone test at 1630° C.	Fused*	Fused very badly.	Very good.	Good
On 20 mesh after 10 days rotting in water, per cent	28.4	55.4	2.7	1.7
Water used to mix, per cent	17.1	....	16.6	18.7
Strength at 110° C., 24 hr. rotting, pounds	90.5	....	60.0	83.0
Strength at 110° C., 1 week rotting, pounds	70.0	....	72.0	83.0
Strength at 110° C., 2 weeks rotting, pounds	79.0	....	91.0	81.0
Strength at 110° C., 3 weeks rotting, pounds	56.0	....	76.0	84.0
Strength at 110° C., 4 weeks rotting, pounds	....	....	62.0	75.0
Strength at 1230° C., 1 week rotting, pounds	439.0	....	534.0†	528.5§
Shrinkage at 110° C., per cent	5.2	....	4.95	4.12
Shrinkage at 1230° C., per cent	9.60	....	8.61	8.94
Porosity at 1230° C., per cent	....	....	4.80	4.55

*Refractory at 1600°C.
† Two did not break at 600 lb.
§ One did not break at 600 lb.

Tests were not carried out on sample B, as it fused badly and rotted poorly, so that we did not consider it of value for retort purposes.

Table 2 gives the data obtained from the test of the above clays (except B) when mixed with standard-size Harbison Walker flint grog. Grand View clay with the above grog, is used for comparison.

TABLE 2

	A	C	Standard Mix
Water used to mix, per cent	14.0	13.0	14.0
Strength at 110° C., 24 hr. rotting, pounds	63.0	38.0	72.0
Strength at 1230° C., 24 hr. rotting, pounds	409.0	463.5	483.0
Shrinkage at 110° C., 24 hr. rotting, per cent	1.5	1.69	2.8
Shrinkage at 1230° C., 24 hr. rotting, per cent	3.9	3.82	3.1
Porosity at 1230° C., 24 hr. rotting, per cent	9.0	7.75	6.8

From the above results, clay "A" was condemned, while clay "C" was considered worthy of a trial in muffles to determine its real value.

The purpose of the individual tests is largely evident, as each is indicative in some way of what may be expected of the material when in service. The tensile strength is perhaps the most important, as it is an indication of the toughness of the vessel, and therefore of its ability to stand handling both before and after burning, as well as to withstand the strains and bumps due to cleaning out the residuum. The purpose of the porosity test is clear, but in view of our manner of making the test the term is perhaps a misnomer. We allow the burned briquettes to stand in water over night, or longer, and determine the increase in weight due to the water absorbed. It is thus more a measure of absorption, but we find it fairly reliable, nevertheless. Attempts have been made in the past to determine the porosity by ascertaining the rate of air leakage through a given mass, but this method was cumbersome and proved less reliable than our present one.

Tests of this nature are not conclusive, of course, but by standardization and comparison they can be made very useful, and will often prevent expensive failures which might otherwise occur in a trial of new materials.

Crushing and Screening System

All of the above materials, except the coke, are crushed and screened in the same system. They are brought from their respective bins by wheelbarrows and dumped into a Sturtevant No. 2 open-door rotary fine crusher delivering into an elevator pit. The elevator raises the material to the top of the building, where it goes to the screen. This is shown in the accompanying flow sheet, Fig. 1.

The screen is of the type known as the "Newaygo Separator." The material passing through it is fed to a conveyor belt, whereby it is distributed to the proper storage bins, of which there are four. These bins are placed above the feed mechanism so that the material falls by gravity to the feeds.

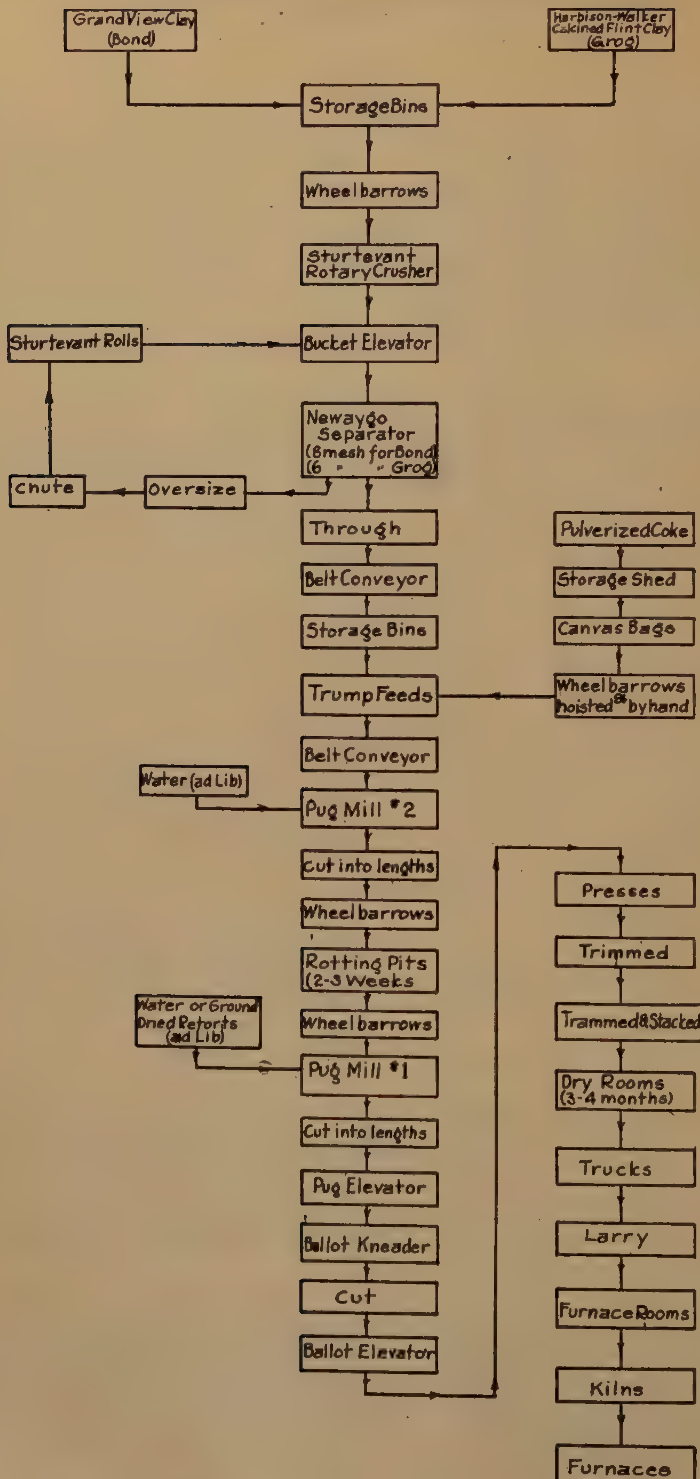


Fig. 1.—Pottery flow sheet. Retorts and muffles.

The oversize descends through a chute to a pair of small Sturtevant rolls, and then back to the elevator boot.

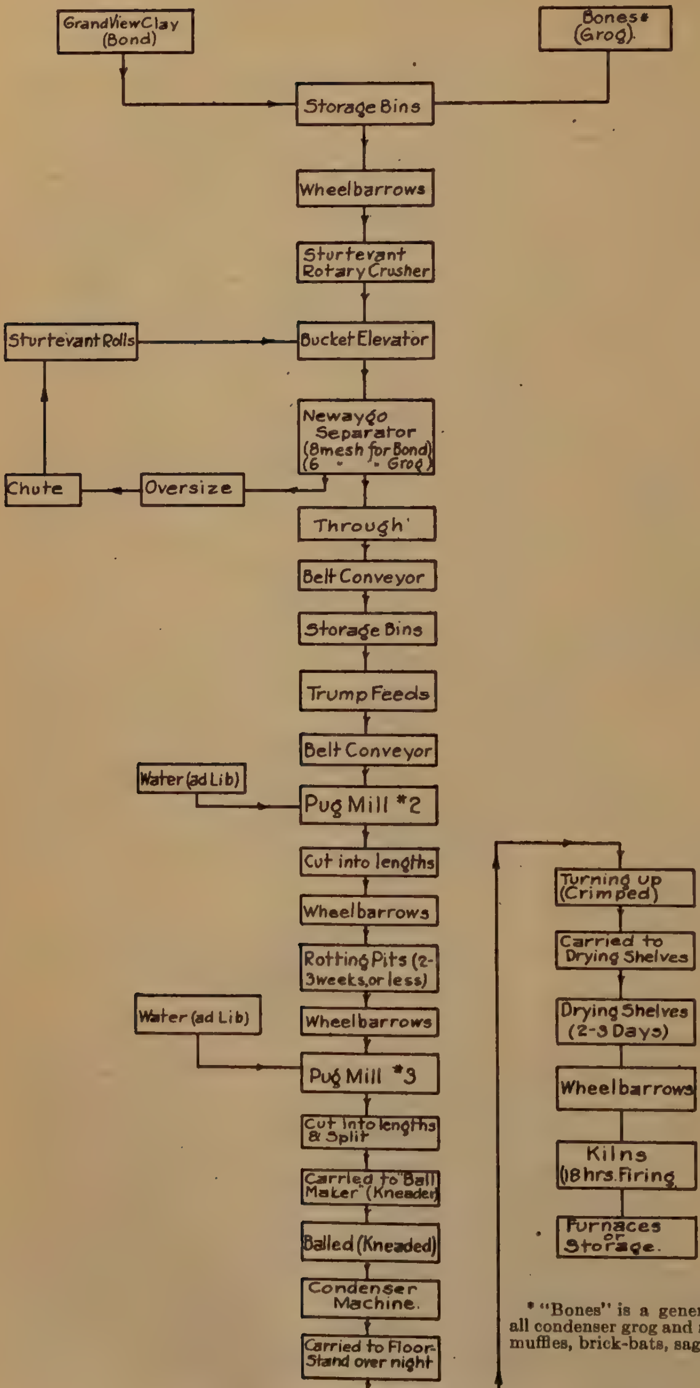
The crusher room is kept free from dust by means of pipes in which a strong suction is maintained at the crusher, rolls, and screen.

We are at present using a 6-mesh screen for grog and an 8-mesh screen for the clay. The use of a single screen for the grog is more or less of a makeshift, as the Newaygo separator was originally installed to deliver three sizes of crushed material quantitatively. In a chapter by William B. Fuller in *Concrete, Plain and Reinforced,* by Taylor and Thompson, are given the results obtained in improving the tensile strength and density of concrete mixtures by scientific sizing of the aggregate. The theory of this sizing is described at some length by Mr. Fuller, and he advocates the use of a parabolic formula in which the diameters of the particles are the abscissas and the percentages are the ordinates. Reasoning that the sizing of grog might play just as important a part in the quality of the spelter vessels as sizing the aggregate does in the quality of concrete, experiments were made some years ago in which the grog was sized on the laboratory scale according to Mr. Fuller's formula. The results were surprisingly successful.

Using identically the same bond clay and grog, except that the latter was in one case used just as it came from the crusher and in the other case was sized in the laboratory, increases in tensile strength as high as 50 per cent. were obtained. There was also a noticeable decrease in porosity. The laboratory tests were performed with the aid of many small screens, and consequently more nearly theoretical conditions were obtained than could be duplicated on a commercial scale. A commercial combination of the various sizes was evolved approximating theoretical conditions, which consists of:

25 per cent.	Through 10 mesh on 16 mesh.
25 per cent.	Through 16 mesh on 30 mesh.
50 per cent.	Through 30 mesh.

The Newaygo separator was installed to accomplish this result, but operating difficulties due to moisture variations and difficulty in obtaining sufficient fine material caused all but one screen to be removed after trial. At present, by judicious mixture of wet and dry materials delivered to the crusher, the pottery foreman is able to obtain a grog of surprisingly close approximation to the desired sizing. Daily screen analyses are made to check the results, but they are not always satisfactorily close. In view of the very encouraging results obtained in the laboratory tests, the writer is convinced that if the sizing can be made still closer to the theoretical, better vessels will be obtained. With this in mind he is giving the separator a further trial and hopes to solve the difficulties encountered.



* "Bones" is a general term applied to all condenser grog and may include broken muffles, brick-bats, saggers, etc.

Fig. 2.—Pottery flow sheet. Condensers.

Feed Room.—All material except coke is fed by gravity to trump feeds from the storage bins mentioned above. The pulverized coke is brought in from the storage shed in wheelbarrow-loads of small canvas bags, is raised by hand to the platform adjoining the trump feeds, and emptied by hand into the feed employed for this purpose. This work is disagreeable, and a housed-in elevator has been designed to eliminate the manual labor. The work on the elevator is at present held up, however, pending trials of vessels containing no coke.

The addition of coke to the spelter vessels was first tried out in 1916, having in mind its use in other plants, particularly abroad. A careful microscopic and other examinations of vessels burned in the firing kiln, however, fails to disclose a trace of unburned coke and it is difficult to see why it can be of any service to the vessels when it is not present. Theoretically the burning out of the coke would leave the vessels porous, which is not desirable. One reason advanced for the use of coke is its lubricating action in the presses. Two thousand vessels containing no coke have been recently made up, however, and after a continuous experience for nearly 2 years with vessels containing coke the two press crews making the vessels, as well as the writer, were unable to detect any difference in their behavior in the presses. The use of coke was adopted on data concerning which recent developments have caused some doubt, and the present tests with vessels containing no coke will, we hope, settle this question more definitely.

The five trump feeds (one for each storage bin and one for the coke) are placed directly under the sloping bottoms of the bins, which form part of the roof on the feed room. The trump feed consists of an inverted truncated revolving cone, below which a circular plate revolves in the opposite direction. The effect of this motion is to spread out the material upon the plate. An adjustable slicer placed at one point of the plate cuts off a fixed amount of the material from the plate, which drops to the conveyor belt below. The belt conveyor and the feeds are separately motor-driven, and the feeds are so arranged that any one may be set in motion. At intervals along the belt, suction pipes are arranged to prevent dusting.

The conveyor brings the material to the feed end of pug mill No. 2 where the preliminary pugging of the material with water is carried out. Only the feed end of this pug mill is in the feed room, the mill projecting through a partition, so that the discharge end is on the ground floor of the pottery proper (see Fig. 3).

The object of pugging, of course, is to secure an intimate admixture of the bond and grog clays with the requisite amount of water. The material is discharged through an opening at the side, as a compact square cake with rounded corners. The pug mill is driven by toothed gearing from a motor.

The feed end of the mill is protected by a ½-in. screen to prevent any coarse foreign material from falling into the mill. The material conveyed by the belt from the trump feeds drops into the feed hopper of the mill, and at the same time water is introduced from a pipe. The amount of

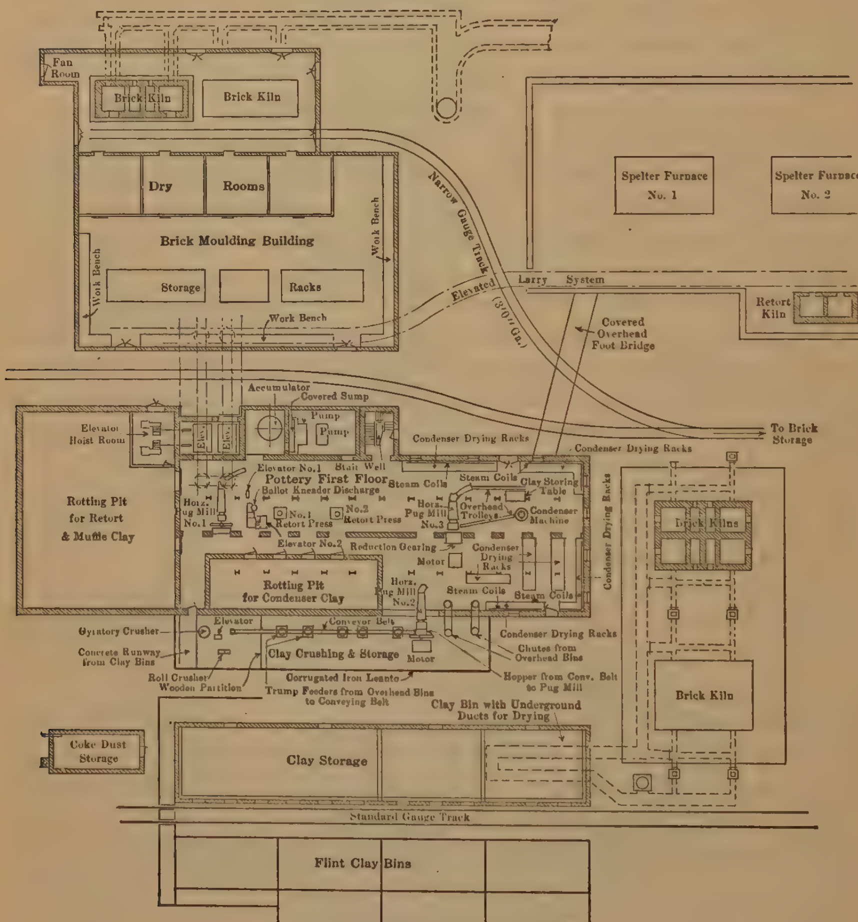


FIG. 3.—LAY-OUT OF PLANT, INCLUDING FIRST FLOOR OF POTTERY.

water used is entirely regulated by the "feel" of the pugged material. Thus, condenser mix is made up so wet as to be distinctly sticky to the touch, while muffle mix contains less water, so that it is just plastic.

An auxiliary spout and chute arranged near this pug mill is used for introducing muffles that have broken in drying and are to be repugged.

The mixes at present in use are as follows:

Spelter Vessels

	Pounds per Minute	
Grand View clay...........	145	
H. W. flint clay............	150	
Pulverized petroleum coke...	19	(water 10–11 per cent.)
	314	

Condensers

	Pounds per Minute	
Grand View clay...........	150	
Grog....................	150	
	300	(water 14–15 per cent.)

The material discharged from the mill is forced out on a wooden bench, about 5 ft. long, which serves to support it until it can be cut. The surface of this bench is kept well oiled to prevent the pugged material from sticking to it.

The material is cut off into pieces 8 to 8½ in. long, weighing about 60 lb. The cutting is done by means of a piece of No. 18 piano wire held taut across the open end of a U-shaped frame made of 1-in. steel bar. This frame is about 14 in. on a side, and after the pug is cut the cutter is allowed to rest on top of the pugged material, thus furnishing a mark for cutting off the next piece. From 13 to 15 of these pugs are loaded into a wheelbarrow, and piled in the "rotting pits."

Rotting Pits.—There are two rotting pits, as shown in Fig. 3. They are large brick chambers with no windows or other means of ventilation, where the pugged material is piled and allowed to remain from 3 to 4 weeks. Burlaps are thrown over the finished piles to minimize evaporation, and at times during extremely dry weather they are moistened, and steam is introduced into the pits. The capacity of the pits is as follows: Pit No. 1, 750,000 lb.; Pit No. 2, 500,000 lb.

The exact nature of the action taking place in the rotting piles is not known, but it is commonly agreed that rotting for the proper length of time materially increases the strength and life of the finished product. It is supposed to be due to bacterial action, and if this is the case temperature conditions are probably important. We are not certain of the optimum temperature conditions, but are considering the installation of a system for controlling temperature and humidity so that more exact knowledge of this subject can be obtained.

Laboratory tests have shown that rotting has a very decided influence upon the strength of St. Louis clay dried at 110° C. after various periods of rotting. The following table is somewhat typical of this action, the tensile strength being given in pounds per square inch:

Tensile Strength Dried at 110° *C.*

	Pounds
24 hours rotting	67.5
1 week rotting	83.0
2 weeks rotting	93.0
3 weeks rotting	67.0
4 weeks rotting	39.0

These results indicate that it is just as important not to rot the mix too long as it is to rot it long enough. With the present capacity of our pottery in relation to the demands of the spelter furnaces we are not at present troubled with the former condition. Not all rotting tests show this same effect, however, and this matter of rotting is a subject that merits further consideration and experimentation.

Up to the time of removal of pugged wads from the rotting pits, the materials for the manufacture of spelter vessels, condensers, and fire-bricks are handled in the same manner, as outlined. From this point on their treatment is different, and will be outlined separately.

Spelter Vessels

Up to this point the term "spelter vessels" has been used to cover both "muffles" and "retorts" as variously used in different localities. At Palmerton both terms are used, "retorts" covering vessels of circular cross-section and "muffles" covering vessels of approximately elliptical cross-section. There is nothing inherent in the terms which justifies this differentiation, but their use in this manner minimizes confusion and facilitates discussion, record keeping, etc. Retorts and muffles are manufactured in exactly the same manner except that the die in the press is of different form.

The rotting mix is removed from the pits by wheelbarrows, and wheeled to pug mill No. 1 for repugging. During the rotting, the outside of the pugs sometimes dries considerably and one object of the second pugging is to again secure a uniform mixture. The main object of the second pugging is to take advantage of the further disintegration of the clay during the rotting, by remixing. The mill used for repugging is similar to the one already described, and is fed by hand. One man attends to the feed while the other cuts off the pugs and places them on the adjoining pug elevator. This man also attends to the discharge of the nearby ballot kneader. The feedman adds a little water, or some crushed dried broken retorts, according as the mix appears to be too dry or too wet. The pugs are cut off about 8 in. long and are placed on the pug elevator to be conveyed to the floor above.

Continuous operation of this pug mill is not required, as the ballots can be turned out faster than they can be pressed into retorts. From time to time, therefore, the press operator signals from the floor above

and the men operating pug mill No. 1 shut off the mill and devote themselves to bringing in a fresh supply of material from the rotting pits to be repugged.

Ballot Kneader.—The feed opening of the ballot kneader is on a level with the press floor, and into this the pugs fall by gravity from the pug conveyor. The ballot kneader is practically a large pug mill placed vertically. The discharge, which is axial, is on the floor below, near pug mill No. 1, and the man attending to the discharge of this mill also cuts off the ballots and conveys them to the ballot elevator.

The ballots are 15 to 17 in. in length, are 14 in. in diameter and weigh over 200 lb. After cutting off, they are rolled along an adjoining wooden bench to a small wooden turntable fixed in the bench, where they are turned through 90° and placed in the holder over the ballot conveyor.

Ballot Conveyor.—This serves to carry the ballots to the tops of the presses on the floor above. When the conveyor is in operation, a ballot carrier passes through the cage about every half minute.

Presses.—There are two Dorr muffle presses, one being generally used for muffles and one for retorts. They are operated by hydraulic pressure at approximately 2000 lb. per square inch, the pressure being furnished by two Worthington duplex pumps and a heavy accumulator. The presses are mounted on heavy brick and concrete piers.

The die for producing muffles makes a vessel of the following dimensions:

Height inside	9.0 in.
Height outside	11.0 in.
Width inside	7.0 in.
Width outside	9.0 in.

The die for producing retorts makes a vessel of the following dimensions:

Diameter inside	7.625 in.
Diameter outside	9.5 in.

Both muffles and retorts are cut to a length of 61.5 in. after removal from the presses.

The cross-section of the muffles consists of two semicircles having a diameter equal to the width of the muffle, these diameters forming the long sides of a rectangle whose short dimension is the difference between the height and width of the muffle.

Table 3 shows the change in size of vessels during the processes of drying and burning.

The operations involved in pressing the vessels are as follows: The ballots are rolled from the discharge "cage" of the ballot conveyor, along a wooden table arranged along the wall parallel to the presses and a short distance behind them. A small inclined hinged shelf leads from this

table to the mouth of each press and permits the ballots to be easily placed in the presses. Only one press is run at a time, and four men are required to operate it and remove the finished vessels, as follows: press operator, press helper, trimmer, trammer.

TABLE 3.—*Average Measurement of* 60 *Vessels* (*Inside Dimensions*)

	From the Dry Room	Red Hot from Kiln	Cold from Kiln
Muffles:	Inches	Inches	Inches
Length	57.57	57.35	57.23
Height	8.62	8.59	8.56
Width	6.91	6.84	6.84
Retorts:			
Length	57.56	57.31	57.18
Diameter	7.34	7.24	7.24

The press operator arranges the ballots on the table behind him, introduces the ballot into the press, and operates the two valve handles, which project through the floor and control the motion of the press plungers. After the vessel has been formed, he cuts it free from the press at the bottom by a cutter similar to those already described as used for cutting off pugs. He also assists in putting the muffle into the wooden muffle case.

The press helper swings on and off the die and cover, and arranges the counter-weighted circular plate against which the vessel is pushed out. He removes the clay remaining in the die, which is thrown back into the ballot kneader. He also assists the trimmer in carrying the muffle in its wooden case to the tram.

The finished vessels are received and carried about in wooden cases in order to avoid injury. These cases are made of just the proper length to hold a vessel and are shaped so as to fit the lower half of it. They are provided with leather handles for ease in carrying and are greased with press oil on the inside so as to prevent the vessel sticking to them. A removable extension piece is attached to the end of these cases when the muffles are placed in them, so as to support a muffle about 6 in. longer than the length desired when finally trimmed.

The trimmer carries one of these cases, with the extension piece attached, to the press. The muffle is placed in this and carried by the trimmer and press helper to the tram car. The latter is capable of holding three cases. On the car, the muffle is pushed gently along until its butt end rests against the end of the holder, the extension piece is removed from the other end of the holder, and the muffle is then trimmed off with a knife by the trimmer, flush with the end of the holder. In this way the

proper length of muffle is secured. After trimming, the end of the muffle is brushed with press oil by the trimmer.

The trammer wheels away the muffles in their cases to the dry rooms and places them. He then returns with the empty cases, and wheels away another load.

This press gang works on a task basis, and makes 320 vessels in 8 hr. or less.

Labor Summary.—It will be seen from this outline that six men are required from rotting pits to finished dry rooms. Four men are required at pug mill No. 2, but as this mill pugs about four times as much per day as does No. 1—that is, it runs only one day in four on muffle mix—a total of seven men is required from crushed material bins to finished dry rooms.

The present pottery is an evolution from a much smaller original installation, made necessary by a somewhat rapid increase in the size of the spelter plant. In enlarging the pottery, as much of the original equipment was used as was possible, but it was rearranged along lines calculated to reduce the required labor to a minimum. In an entirely new plant designed for the present capacity, further improvements could probably be made, but we feel that under the existing conditions a great deal has been accomplished.

Dry Rooms

The layout of the rooms for drying and storing the muffles and retorts is shown in Figs. 4 and 5. The smaller rooms are about 18 by 40 ft., and hold 700 vessels, while the larger rooms are about 25 by 35 ft. and hold 1000 vessels.

The dry room corridors on the different floors are connected by two elevators, one of these being used to transport the muffles from the press to the dry rooms, the other for carrying the muffles from the dry room to the larry for transportation to the furnaces. Tracks are placed in the corridors in front of the dry rooms for ease in trucking away the dried muffles, and a turntable in front of the entrance to each room permits connection with a track extending the length of the room. In the corridor are placed also the recording portion of recording thermometers with which each dry room is provided, also the valves for control of steam to the steam coils, the ventilation control, and the electric light switch.

The dry rooms are about 7 ft. in height, with hollow tile walls and an open wooden lattice floor. Below this floor run steam coils, of which there are two for each room. From the ceiling of the dry room is suspended an asbestos-covered sheet-iron flue, containing adjustable openings. This is connected with a stack, and serves to carry off the moisture. The rooms are closed by sliding fire doors.

The muffles are placed in the room by the trammer, beginning at the back. They are set upright on their butts, and placed close to each other.

Our drying period is 4 months. When a room is filled the door is closed, and a record showing the date the room was filled and the number of vessels put in is placed on the door. In summer, the vessels remain in the room from 2 to 3 weeks without steam, but in winter enough steam is turned on at once to bring the room up to summer heat. We feel confident that for best results a vessel should never be allowed to become chilled, hence the difference in practice during the different seasons. Only one coil is turned on at first, and a note made of the fact on the card on the door. About a month later the second coil is turned on, and the record made. This gradual drying minimizes warping and breakage during drying, and has a generally beneficial effect on the service of the

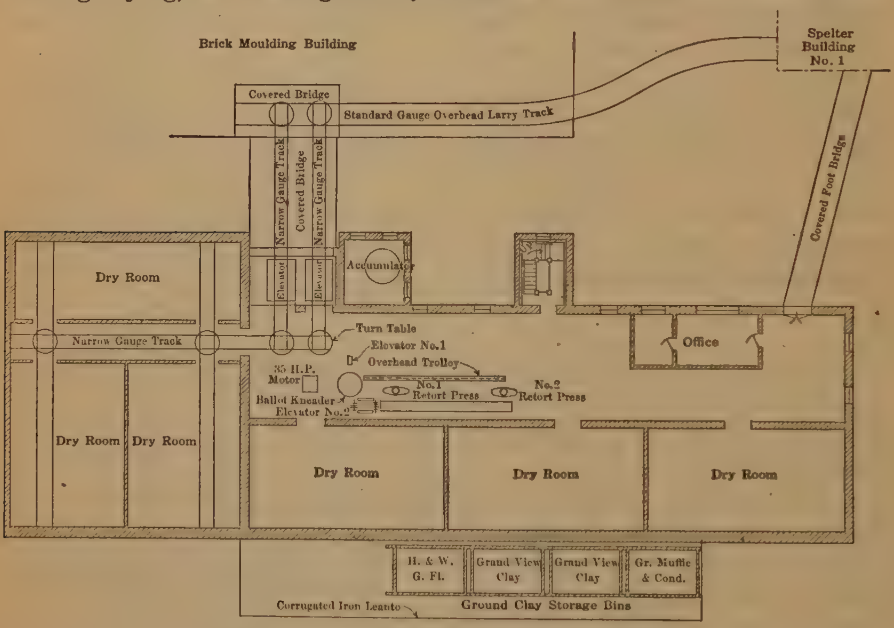


FIG. 4.—PLAN OF SECOND FLOOR OF POTTERY.

vessels. The final temperature of the rooms will range from 120 to 125° F. When taken out after drying, the muffles are sometimes considerably warped, but this does not interfere with their use provided it is not excessive. The muffles placed next to the walls of the dry room show a strong tendency to crack and break, due in all probability to the unequal temperature around them. Various attempts have been made to overcome this by lining the walls of the room, but have not yet been carried far enough. Our practice at present is to fill the rooms in rotation as far as possible, thus taking advantage of the warmth from adjacent rooms.

The vessels when dry are loaded by the truckers into iron boxes with high sides. These are mounted on removable trucks, and are so arranged

that they can be picked up by a larry and carried to the furnace buildings. The boxes will hold from 14 to 18 vessels each and the muffles are prevented from injuring each other by having ropes wrapped around them. They are burned in firing kilns located on the spelter furnace room floors (Fig. 3), and taken from there directly to the furnaces while red hot.

Spelter Condensers

The condenser mix is removed from the rotting pits in the same manner as outlined for muffles, and is repugged in pug mill No. 3. This mill is similar to the pug mills already described, but instead of being driven directly by tooth gearing from a motor, it is driven by an induction motor

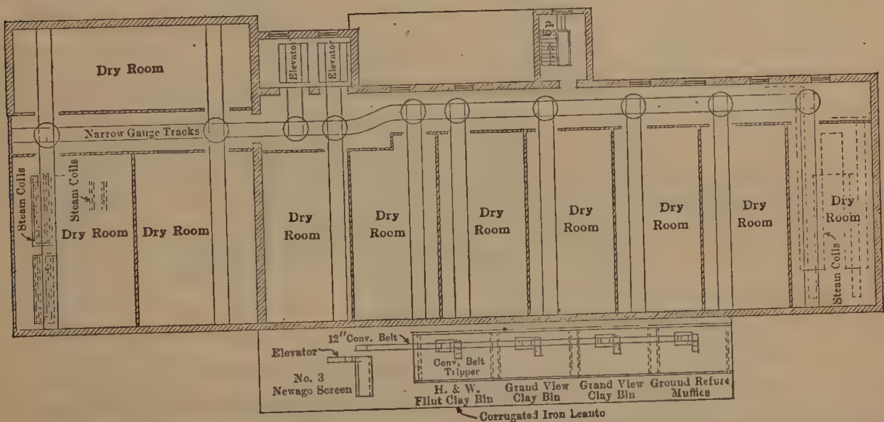


FIG. 5.—PLAN OF THIRD FLOOR OF POTTERY. (DRY ROOMS ON FOURTH AND FIFTH FLOORS ARE SAME AS THOSE ON THE THIRD FLOOR.)

geared down by pulleys and belts. A wire is fastened across the middle of the discharge opening, so as to automatically cut the pug material in two. One man attends to this mill, bringing the material from the rotting pits, cutting off the pugs, and carrying them to the ball maker's table, placed near the condenser machine. The operation of the pug mill is, therefore, not continuous. The pugs are cut about 6 in. long.

The ball maker, standing at his table near the condenser machine, takes a half pug, kneads it into a mass much resembling in shape a large loaf of bread, and drops it into the condenser mold in the machine, at the same time operating the counter placed on the machine.

The condenser machine is a standard machine, altered somewhat to decrease maintenance and improve its operation.

Three men are required for the operation of the machine; the ball maker, whose duties have already been described, the condenser machine operator, and the carrier. The motions of the machine operator are as follows:

1. Puts empty mold into machine.
2. Revolves the table.
3. Greases new mold with press oil.
4. Trims finished condenser with knife, flush with the top of the mold, and throws trimmings into a basket.

NOTE.—This basket is suspended from an overhead rail, and from time to time the pug-mill operator pushes it to the mill, where the material is repugged.

5. Removes the waste material pushed out at the bottom of the mold by the cone, on the condenser then being formed, and throws the material into the basket.
6. Takes trimmed condenser and mold from the table and places them on rack beside him.
7. Takes empty mold from the rack and places it on the machine.

The operations are then repeated. The operation is very rapid, over five condensers being turned out per minute, or about 200 per hour under usual operating conditions. All of this work is on piece rate.

The carrier removes the finished condensers with their molds from the rack beside the machine operator, and empties the molds, standing the condensers small end up on the floor. Here they stand and dry for a day, as they are entirely too soft to be handled directly after formation.

The dimensions of the condensers as they leave the machine are as follows: Height, 17 in.; small end, inside diameter, 2¾ in.; outside diameter, 4¼ in.; large end, inside diameter, 6 in.; outside diameter, 7½ in.

Turning Up

As formed on the machine the condensers are all circular. The large end is therefore turned up or crimped so as to fit snugly into the mouth of either a circular retort or elliptic muffle. The tools used for this purpose consist of sheet-iron forms of the desired size and shape about 6 in. high, mounted on wooden blocks. After drying on the floor over night, the soft large end of the condenser is pushed into the desired circular or elliptical form. This process also causes a distinct bellying of the central part of the condenser.

Drying.—After crimping, the condensers are placed large end up on nearby latticed wooden shelves, where they dry from 2 to 3 days. These shelves have steam pipes passing around the front of them so as to furnish heat for drying. After drying, the condensers are loaded in wheelbarrows and taken to the kilns.

Burning.—The kiln building adjoins the pottery and clay storage bins on the east. The building contains, in all, eight small downdraft kilns in which the condensers and brick are burned (see Fig. 3). The kilns are fired with the finer sizes of anthracite coal, a small motor-driven blower furnishing the draft.

The condensers are piled into the kiln on end, with the large and small ends alternately placed down so as to fill up the space as completely as possible. A kiln will hold from 900 to 1000 condensers.

After the kiln is filled, the door is bricked up, and the fire started, the firing being kept up for a day. As it takes some time for the kiln to heat up, however, the condensers are not "at heat" for more than about 18 hr. The kilns are brought up to a bright red heat, about 900° to 1000° C. No attempt is made to keep within these figures; on the contrary, the kilns are fired as hard as possible with the fuel in use, and tests with pyrometers and Seger cones have shown that our average temperatures are within these limits.

As two of the kilns are used for fire-brick, the remaining six allow one kiln to be filled a day, not including Sundays.

After burning, the condensers are removed in wheelbarrows (about 30 making a load) and are taken either to the condenser storage or are loaded by hand into the iron boxes with high sides (the same as used for the retorts and muffles) and taken by a larry to the furnace rooms. Loading and emptying the kilns are done on piece rates, only unbroken condensers being counted.

During the firing, some of the condensers may crack, in which case they are rejected. The condensers from the top of the kiln have a more or less pronounced brown color, due to the deposition of ash on them, as the flame strikes directly against these condensers. Below, the condensers are nearly white, while those at the bottom of the kiln are bright pink in color. We find that the condensers from the middle of the kiln appear to give somewhat better service than the rest, but this difference in firing can hardly be eliminated.

The use of the high-grade St. Louis clay for making condensers dates back only 2 or 3 years, cheaper local clays having been previously used. The better service given by the St. Louis clay, together with the saving in handling due to the better condition of the clay as received, enabled us to prove that the more expensive material was in the end the cheaper.

Manufacture of Brick

For the manufacture of brick, the same rotting mix as is used for retorts and muffles is employed. It is first repugged, adding more water so as to make a much more pasty mass than is used for vessels. The repugged material is then sent to the brick-making shed, which is situated to the north of the pottery. The process employed is similar, whatever kind of bricks are to be made. A wooden mold of the shape required, but without top or bottom, is placed on a flat greased board. The pugs are then cut up into small pieces with a wire. The brick maker, standing over the mold, takes pieces of convenient size and throws them forcibly into the mold so as to cover the bottom of it. The clay is then tamped down with a wooden tamper, and the process is repeated until the mold is full. The excess clay is then cut from the top, by pulling a wire

through guides fixed to the top of the mold, and the top of the brick is carefully smoothed down with a wooden float. The mold is then removed, and the sides and bottom of the brick are smoothed in the same way. The bricks are then dried on shelves in the brick shed, or in drying rooms which are also in the shed, and are finally fired.

As indicated earlier in this paper, the brick made are used principally in the spelter-furnace laboratory. The main requirements for this service are maximum refractoriness and minimum coefficient of expansion. As these qualities must be possessed to a high degree by the materials entering into spelter vessels, these materials are used also for making the fire-brick. Furthermore, the simplicity obtained by this elimination is a factor which makes for a decrease in cost, minimizing operating and accounting errors, and improving operating conditions generally.

Conclusion

In conclusion, a few words as to the writer's idea of the qualities we should aim to obtain in spelter vessels may not come amiss. Theoretically, the perfect vessel should have a maximum heat conductivity, refractoriness, mechanical strength, and toughness, and a minimum (or no) porosity and coefficient of expansion. At the same time, it should be sufficiently strong, when dried, to allow handling without undue breakage, and should be chemically inert to the various slags which the gangue in the different types of zinc ores produce. Materials which possess all these desirable qualities are, unfortunately, still to be discovered.

There are many beds of clay in this country whose refractoriness is amply high. The writer has examined and tested many such clays which have exceeded the St. Louis clay in refractoriness, usually to find their shrinkage too high or their toughness too low, or both. Although the combination of St. Louis clay and calcined flint gives us the best results of any yet tried, there is still much to be desired. The porosity of this mix, when burned, averages about 6.5 per cent. This figure is found by determining the amount of water the mass will take up in 12 to 15 hr., and where water will penetrate in this time zinc vapors and slag will penetrate during service. This is obviously detrimental to vessel life and spelter practice.

The purpose of calcined grog in the mix is mainly to lessen the shrinkage of the mass as a whole, by introducing a material which has a much lower shrinkage than that of the bond clay. This in itself produces an undesirable effect, for the bond clay still performs its normal shrinkage during burning, and as it must shrink around the grog particles having a much lower shrinkage, minute cracks are opened up which account largely for the porosity found. A one-material vessel, then, should be more satisfactory, or if cost considerations made two or more materials

desirable, they should all have the same shrinkage. As a matter of fact, this latter condition is hardly obtainable, as the drying of the bond clay produces shrinkage which in all probability is partially responsible for this porosity.

We have at times made vessels containing refractories manufactured under various trade names, but so far without sufficient success to warrant the additional cost of the materials.

One feature that is probably largely responsible for a certain slowness in developing improvements in vessel manufacture is the difficulty in obtaining conclusive tests. A considerable period of time must necessarily elapse between the first steps in the production of an experimental batch and its final trial in the furnaces, and there are so many factors that may influence the results that it is often difficult to decide just what has been accomplished. Operating difficulties may affect the period of rotting and drying, labor conditions at the spelter furnaces may change, ores may change more or less in their character, and so on almost without limit, as those who have tried probably know. Furthermore, such experiments must be handled with caution, for ill-advised innovations are likely to prove very costly.

The time required for the manufacture of spelter vessels is a further handicap to the pottery man in determining sources of trouble under ordinary operating conditions. A batch of muffles may at times fail after an unusually short life, or may exhibit certain unusual features which are undesirable, but a search for the cause is usually fruitless. A few months ago some of our muffles showed a peculiar blistering on the outside after firing in the kilns, and there were as many opinions given regarding the cause as there were people qualified to express an opinion. A careful search was made to obtain some clue as to the cause, but without avail, and in a short time the trouble disappeared, leaving us as much in the dark as ever.

If the perfect material is discovered, pottery men will not be slow in making use of it. In the meantime, efforts must be directed toward a judicious selection of the materials obtainable, balancing their good qualities against those which would be detrimental to retort service, together with improvements in their combination and treatment.

Discussion

H. Ries, Ithaca, N. Y.—The part of Mr. Fiske's paper that interests me especially is that portion dealing with the raw materials, and the tests that were applied to the raw materials in order to determine their value in advance, if possible, because that, I believe, is a step in the right direction.

For a number of years I have done my best to persuade different

consumers of clay to develop standard methods for testing raw materials. We have in this country a large variety of clays which can be used for a great many different purposes, but in many cases the consumers either have no standard methods for determining the value of those clays in advance, or if they have they do not publish them; consequently it often puts the clay miner at a disadvantage. If, for example, manufacturers of different types of refractory materials would state publicly just what physical tests their raw materials have to meet, it would, I believe, help them to find more easily the materials that they are looking for, and the clay producer would also be benefited. It would also encourage him to have his clays tested in advance, so that samples could be accompanied by certificates of physical tests that would be of convenience to the manufacturer.

At the present time, in this country there are certain types of clay which seem to be rather scarce. There apparently are not many deposits of clay that are exactly suited for the manufacture of zinc refractories. So, too, there are not many deposits of clay that are suited for the manufacture of glass-pots, or the blocks for glass tank furnaces, or clays that can be used for graphite crucibles.

Now, in many cases, no standard series of tests has been developed. I am also gratified to see that Mr. Fiske considers the tensile strength tests of value. Some years ago the American Ceramic Society appointed a committee to draw up a series of recommendations covering standard tests. These recommendations were especially for the use of State geologists and the U. S. Geological Survey. Among the tests that were considered was the tensile strength test, which Mr. Fiske has mentioned in his paper. Unfortunately, a majority of the committee turned it down, and the test was therefore not officially approved, but two of us, who formed the minority, believed in it. Since then the other members of the Committee have come to see that that is an important test to make and Mr. Fiske has emphasized it in his paper.

I would like to ask Mr. Fiske whether he has determined the tensile strength of these clays when burned at different temperatures, and whether he considers that the tensile strength test can be used at all to indicate the resistance of the clay to cracking in drying? Personally, I have not found that the tensile strength test indicates the clay's resistance to withstand cracking when drying. In other words, a piece of clay molded into a briquette may not crack in air drying, whereas the same clay formed into a full-sized brick may crack badly unless dried very slowly. Sometimes clay that shows a very low tensile strength will stand even rapid drying without cracking. On the other hand, clays that show a rather high tensile strength are sometimes what the brick maker calls tender; they crack very easily if dried too rapidly. I am also glad to see that Mr. Fiske thinks that the absorption test is just as

satisfactory to make as the porosity test and I think he will agree with me that the latter is very troublesome to make. As a matter of fact, if one determines both the absorption and porosity curve, they run practically parallel, so the absorption curve gives an indication of as much value as the porosity curve.

C. P. Fiske.—We have tried the tensile strength of the various clays only at temperatures between 1200° and 1300° C., because we are not interested in any other temperatures, and our tests are made primarily to determine how these clays are going to work in muffles. In regard to the tensile test, I am surprised that Mr. Ries does not find that the test gives evidence as to cracking. His experience is contrary to ours, for we have found the tensile test a very good indication as to whether the clay is going to crack or not. I might mention an exception to that; some years ago we had a mixture of two New Jersey clays that gave an excellent tensile strength test. The muffles, however, always cracked, beginning at the mouth, usually cracking back 6 or 8 in. in the first 2 or 3 hours, and we seldom got one that would give us reasonable service. That one case does agree with his experience, but it is the only exception that I can recall.

F. E. Pierce, New York, N. Y.—Mr. Fiske speaks of using the trump feed for measuring the quantities of materials that are used in making the various mixes. We have paid two visits during this meeting to plants where pottery operations were going on, one to the Laclede-Christy plant where we went through the dry-room and saw the mixing of materials mainly for the manufacture of brick; another one to the East St. Louis plant of the American Zinc, Lead and Smelting Co., manufacturing spelter furnace retorts and condensers. In the first one, I think everyone was impressed with the large number of laborers required to handle the materials.

I happened to be connected with The New Jersy Zinc Co. at the time the pottery described by Mr. Fiske was altered, and I was at Palmerton at the time the trump feeds were installed. There was considerable difficulty at first in getting them adjusted and I left before they were finally made to work properly. I am very much pleased to hear that they are satisfactory.

The reading of Mr. Fiske's paper indicates that a great deal of labor is done away with in handling materials from the time they are received at the pottery up to the final vessel or brick, and apparently very satisfactory results have been obtained in getting exact measurements. It seems to me that all factories are vitally interested in matters of this kind where economies in labor are introduced, and especially at this time when there is not enough labor to go around. A little later on, when the spelter business will be on a very close competitive basis, it will probably

be most essential for all companies to introduce as many labor-saving devices as can possibly be provided.

Retorts are a material item in the working cost of zinc ores. Before the war they were estimated as costing somewhere between 75 c. and $1. I understand that today they cost between $1.50 and $2.

It would be interesting to hear from Mr. Fiske just what troubles were experienced in adjusting and checking up the trump feeds in the continuous operation of the plant.

C. P. Fiske.—The original trouble with the trump feeds was that the revolving cone above the plate was of such a diameter with relation to the diameter of the plate, and such a height above it, that the material took its angle of rest at the edge of the plate. The trouble has been remedied by placing a band so that the angle of rest of the material is well inside the edge of the plate, and none is discharged but what is scraped off by the slicer. I might say, however, that the relative dryness of the material is a factor which must be watched to get the best results.

A Few Notes on the Future Work of the Petroleum Geologist in the Mid-Continent Oil Fields

BY DORSEY HAGER, *TULSA, OKLA.

(St. Louis Meeting, October, 1917)

THE possibilities of finding new oil pools in Oklahoma and Kansas are far from promising.

In 1916, the only new pools of importance were the Franchot pool near Bixby, the Garber, and the Billings pools in Oklahoma, and the Virgil and Utopia fields in Greenwood County, Kansas.

In the first 8 months of 1917, a small pool of unknown extent was opened south of the Stone Bluff pool in Oklahoma, and a pool near Casey, Paunee County, Oklahoma. In Eastern Kansas, interest has been caused by the sinking of new wells near the old Iola pool, but nothing of magnitude has yet been found. The outlook for new pools is far from encouraging, especially as geological surveys have covered all of Oklahoma and Kansas. There are a few areas where geological surveys throw little or no light on the subject, as exposures are poor, but the chances of finding new pools in such areas are slight indeed, if one can judge from the past history of such areas.

So far the geologists have based their opinions largely upon the presence or the absence of geologic structure, such as anticlines, domes, and terraces, but some pools will most certainly be found where no surface structure exists, but where local changes in the sand body favor oil accumulations. However, the possibilities of new pools off structure are small, as fully 90 per cent. of the oil pools are on definite structure.

In the past, it is estimated that one test in 150 "wild cats" or new tests has brought in a new field. The writer feels safe in asserting that the proportion will decrease still further, so that but one out of 500 new tests may bring in a pay well in a new area.

From now on the principal field of work in Oklahoma and in Kansas will consist in developing the present pools. The years 1915, 1916 and 1917 saw the high peak of Oklahoma and Kansas oil production. The Eldorado and the Augusta oil fields present the largest possibilities for continued production in Kansas. These two pools have been scarcely more than one-third developed, and have greater potentialities than any of the present Oklahoma pools. In Oklahoma, pools of magnitude should

* Petroleum Geologist and Engineer.

be developed at Garber, Garfield County, and at Billings, Noble County, as the domes are large, and development has just commenced. The Osage-Hominy dome in Osage County will perhaps be a large producer, considering the area of undrilled acreage. The Morrison pool has possibilities, but to date is more a gas area than an oil producer. The new Fox pool north of Healdton indicates an erratic occurrence of oil and gas, and a large pool may be developed, but as yet it is too early to judge possibilities.

An increasingly important factor in Oklahoma and Kansas has been the finding of deeper productive sands. A horizon lying below the Pitkin limestone (the Upper Mississippi limestone) and the Boone Chart (or the Lower Mississippi) has been known as a pay horizon in Eastern Central Oklahoma for a number of years. This horizon has recently been found productive in the old Glenn pool area, and also near Sedan, Kans. The finding of this deeper productive horizon opens a large field of deeper drilling below the Bartlesville, Tucker, and the Burgess horizons. It would not be at all surprising to see the Cleveland and the Cushing pools also strike the deeper horizon. Besides the deeper horizon, there are numerous places where the Tucker and the Burgess sands are as yet untested, and these sands will now be tested. A deeper sand should also be found at Healdton in southern Oklahoma.

It is becoming recognized by the oil operators themselves, though not yet by the general public, that the Mid-Continent oil fields are entering the final period of intensive internal development. The period of expansion has passed and now several millions of acres of oil leases will be drilled and eliminated. The costly rentals will be stopped, and the oil fields will settle down to an investment basis. The day of the speculator and the gambler are past, and the close, cost-keeping operator who saves at every turn will now come into his own. The spectacular fortunes will be ancient history, but large sums of money will unquestionably be made from producing oil.

This period calls for more specialized work by the petroleum engineer. It also calls for a closer application of engineering methods. In the past 4 years the time of the geologists has been devoted largely to scouting and to mapping new untested areas. Their success in that line is attested by the fact that 70 per cent. of all the pools opened in the Mid-Continent fields in the past 4 years have been opened on geological advice.

Oil operators who have employed geologists have come to learn that engineering methods can eliminate a large percentage of risk. Under ordinary haphazard drilling methods, one test in 150, or 0.66 per cent., has opened a new pool. The geologist has made that risk one in three, or 33.3 per cent. The operator who plays geology has a 50 times better chance of striking oil, than he who does not. Oil operators appreciate what the geologist or petroleum engineer can do for them in new

untested areas, but they do not yet realize the tremendous saving that can be made in developing the pools more intensively. Many thousands of dollars can be saved by avoiding needless "dry" holes on the edge of oil pools. With the information that an engineer can often assemble and classify, much needless drilling can be saved. Nearly all engineers know of instances where companies drill in synclines and down the dip after it has been definitely proven by one or two tests that such areas were locally unfavorable. The engineer, too, can be of service in overcoming water and gas troubles, which trouble the operators in pools like Cushing, and at Blackwell, Okla.

Closer coöperation between engineer and operator is essential. The petroleum geologist has won his recognition, but that is not enough. He must make himself an invaluable factor not only in untested areas but in efficiently developing the proven areas. The detailed mapping and close study of sand conditions, as shown by the well records and the logs, can guide an operator in deepening his wells to lower sands and in extending his property.

The intelligent use of petroleum geology and engineering methods will mean a great saving to the oil industry, but as yet only a few of the operators realize that a geologist is of any service in intensive development. It is our duty not only to ourselves as geologists and engineers, but to the Nation and to the petroleum industry, to bring the average oil man to a complete understanding of the value of applied petroleum geology. We can do this by personal talks, by written papers, and best of all by showing results in our professional work.

In briefly summarizing the questions touched upon above, the following points are to be emphasized, namely:

1. That the possibility of extending the present producing areas in Oklahoma and Kansas is very slight.

2. That the geologist has reached his limit in outlining new untested areas.

3. That the principal work of the oil men will be in developing and extending the areas already proven productive.

4. That there is a possibility of finding deeper pay sands in many areas.

5. That the period of intensive development has now arrived.

6. That the petroleum engineer should now be utilized more fully than before.

7. That the oil operators do not fully understand the uses of applied geology and petroleum engineering, and

8. That every engineer should endeavor to bring the oil operator to a realization of the value of geology and of engineering methods in intensive development of oil properties.

Geologic Structure in the Cushing Oil and Gas Field, Oklahoma*

BY CARL H. BEAL,† WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

Introduction

During the latter part of 1915 and the first half of 1916, the writer held the position of geologist in connection with the conservation work instituted by the U. S. Bureau of Mines, on oil and gas land belonging to the Indians of the Five Civilized Tribes in Oklahoma. The duty of the geologist was to correlate formations containing oil, gas, and water, and to determine the relations of these substances in the different formations so that the conservation agents might carry on their work most intelligently. On account of the rapid development of the Cushing field, which lies in the western part of the Creek Nation, much work was done there by the inspection force.

In the course of the geologic studies a large amount of information was collected, which, on analysis, has disclosed some interesting facts worthy of publication. The report, of which the following remarks are an abstract, has been prepared and submitted to the U. S. Geological Survey for publication. This report is now (June, 1917) in the press and will doubtless be issued in a few weeks.

Methods Used

In addition to the structure maps showing the folding of the Pawhuska limestone, which crops out in several places in the Cushing field, separate structure maps of the three most important productive sands—the Layton (Fig. 2), Wheeler, and Bartlesville—have been prepared. These structure maps were prepared by obtaining the logs of nearly all the wells

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* Abstract of *Bulletin* No. 658 to be issued shortly by the U. S. Geological Survey in coöperation with the U. S. Bureau of Mines.

Published to afford opportunity for discussion at the St. Louis meeting in October. In this brief abstract only the more important data and conclusions are given. Published by permission of the Director of the U. S. Bureau of Mines and the Director of the U. S. Geological Survey.

† Petroleum Technologist, U. S. Bureau of Mines.

drilled in the Cushing field, by subtracting the surface elevations from the depth of each sand given in the log of that well, to obtain the elevation above or the depth below sea level of that sand. Inasmuch as the accuracy of structure maps depends upon the accuracy of the elevations and of the logs, the maps prepared cannot, of course, be considered absolutely accurate, but in general they were sufficiently accurate to serve as an excellent working basis in the analysis of the information collected.

In an attempt to prepare a convergence map showing the increase in the thickness between the Layton and Bartlesville sands, the intervals between these two formations, as indicated by the available logs, were plotted on a map, but as the local irregularities were so great that it was impossible to construct a convergence map, the map which accompanies the more detailed report shows only certain groups of the same or similar intervals. For instance, all wells showing an interval of less than 1050 ft. (320 m.) were separated from those in which the interval was between 1050 and 1100 ft., which in turn were separated from the areas where the interval was between 1100 and 1150 ft., and so on.

In studying the oil, gas, and water bodies in each sand, maps showing their relations were prepared by plotting the content of each sand as recorded in the logs collected. In studying the water conditions, the depths below sea level at which water was encountered in certain wells were plotted on a map. This map showed that the depths were very diverse, a fact that led to a detailed study, by the use of contours on the surface of the edge water, of the actual position of the water backing up the oil and gas.

Summary

The work done in the Cushing field has consisted in the study and analysis of the information collected in the course of the inspection of the oil and gas land and the following principal facts have been disclosed as a result:

1. The folding of the formations in the Cushing field usually becomes greater with increase of depth, and there are many marked differences in structure among the Layton, Wheeler, and Bartlesville sands, and the surface beds.

2. The interval between the Layton and the Bartlesville sands is generally greater around the edges of the anticlines than on their crests.

3. The distribution of the bodies of oil, gas and water, indicates that the source of the oil lay west or northwest of the Cushing field.

4. In general, the oil area in an elongated dome where folding is simple extends farther down on the long axis of the anticline or dome than it does on the steeper sides; or, in other words, the area beneath which a sand contains water only extends higher on the steep flanks of an elongated dome than at the ends where the beds are more gently tilted.

5. The water surfaces on which the oil and gas rest in the different sands are not level, but are inclined away from the center of the anticlinal folds. This inclination does not conform with but is usually less than the dip of the bed in which it occurs.

Development

The Cushing field, the most productive light-oil field in the world, embraces nearly 35 square miles (90.6 sq. km.) of productive territory, in which have been drilled about 2500 wells ranging in depth from 1200 to nearly 3000 ft. (365 to 914 m.) (Fig. 1). Oil was first discovered in March, 1912, when C. B. Shaffer and others drilled well No. 1, on the Annie Jones (F. M. Wheeler) farm in the NW. ¼ Sec. 31, T. 18 N., R. 7 E., about 1 mile north of the present site of Drumright. This well was drilled to the Wheeler sand and for more than a year and a half this sand produced the entire oil output of the field. In December, 1913, however, the Prairie Oil and Gas Co. completed the first well to the Bartlesville sand in Sec. 3, T. 17 N., R. 7 E. Development has been extremely rapid, especially since the discovery of oil and gas in the Bartlesville sand, and at one time the daily production of the field reached more than 300,000 bbl. of oil. The total marketed output of all the oil sands in the Cushing field has been more than 200,000,000 bbl.

Stratigraphy

This investigation has been only in the nature of a study of underground conditions with a view to determining the relation of the contents of the more important oil- and gas-producing sands, and very little stratigraphic work was done. The rocks of the area, according to Buttram,[1] are exclusively sedimentary and, except the "terrace" sands and the alluvial deposits, are all of late Pennsylvanian age.

In this part of Oklahoma the rocks dip in general to the west, the base of the Pennsylvanian series cropping out in northeastern Oklahoma in the western foothills of the Ozark Mountains. The Pennsylvanian formations at the surface become successively younger as the Cushing field is approached and are hidden from view about 12 miles west of the field by the overlying Permian series. The exact line between the Pennsylvanian and the Permian series is uncertain, but in accordance with the provisional current usage of the Geological Survey it has been drawn at the base of the Cottonwood limestone, which is about 50 ft. above the Neva limestone. It should perhaps be drawn as low as the Neva limestone, or possibly, as some writers think, at the base of the Elmdale,

[1] Frank Buttram: The Cushing Oil and Gas Field, Oklahoma. *Oklahoma Geological Survey, Bulletin* No. 18 (1914).

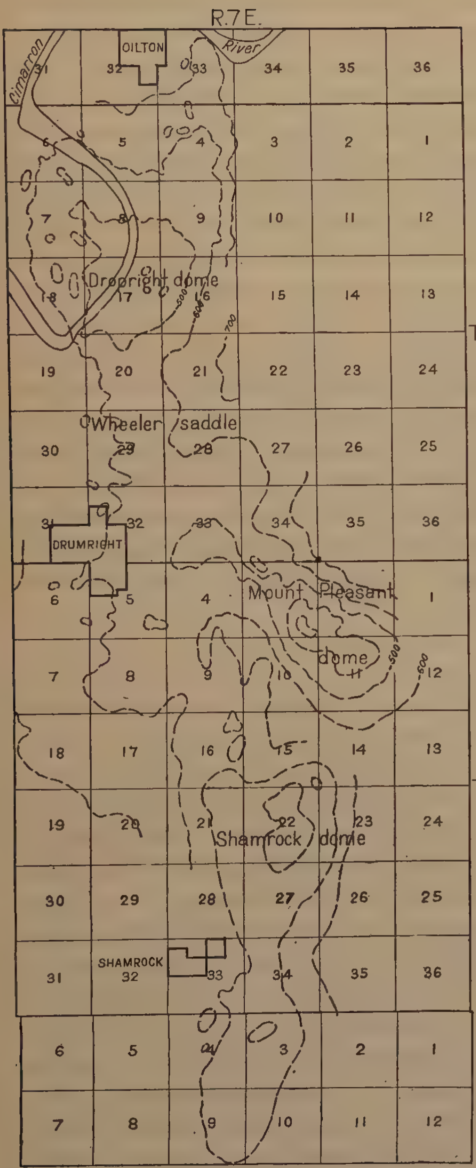

FIG. 1.

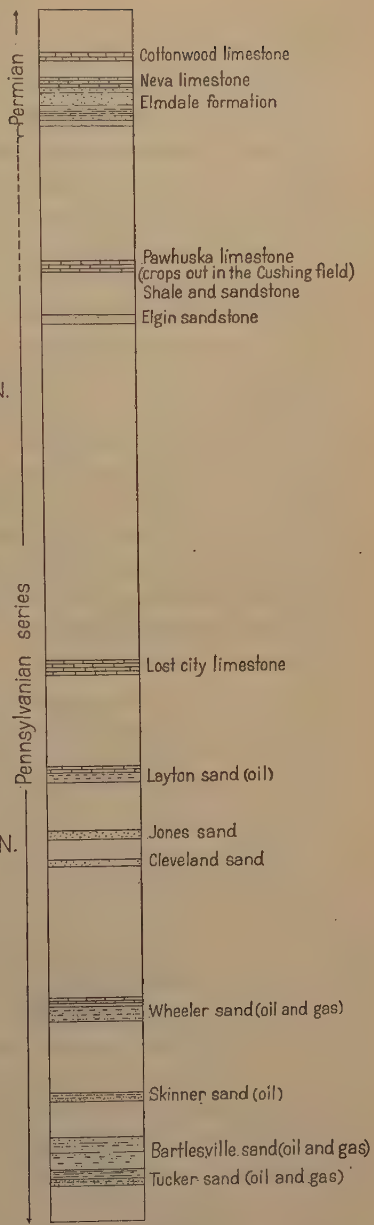

FIG. 2.

FIG. 1.—SKETCH MAP OF CUSHING FIELD SHOWING GENERALIZED STRUCTURE OF LAYTON SAND AND LOCATION OF PRINCIPAL STRUCTURAL FEATURES. CONTOUR INTERVAL IS 100 FT.

FIG. 2.—COMPOSITE COLUMNAR SECTION SHOWING RELATIVE VERTICAL POSITIONS OF PRODUCTIVE OIL SANDS IN THE CUSHING FIELD AND SOME OF THE BETTER KNOWN FORMATIONS IN KANSAS AND OKLAHOMA.

which embraces 130 ft. of sediments below the Neva limestone of the Kansas section (Fig. 2). This limestone forms an escarpment just west of the town of Cushing, Oklahoma, 12 miles west of the oil field.

The most important outcropping formation in the Cushing field is a prominent limestone, which according to Buttram is probably equivalent to the Pawhuska limestone of northern Oklahoma. Later investigations made by the U. S. Geological Survey in regions northeast of the Cushing field indicate a necessity for some nomenclatural revisions, which, however, are not yet completed. It is even somewhat probable that the limestone called by Buttram the Pawhuska, and used in this report as a key horizon, may not be the same as that conspicuously exposed near Pawhuska.

In the Cushing field the drill penetrates a series of alternating sandstones, shales, and limestones. The limestones apparently become less prominent and the sandstones more prominent toward the south end of the field near Shamrock. According to Buttram,[2] the Pawhuska limestone near the center of the field in the vicinity of Drumright lies about 2340 ft. (713 m.) above the Wheeler sand. The Bartlesville sand in the Cushing field is supposed to be equivalent to the Bartlesville sand of the fields of northeastern Oklahoma, and lies near the base of the Pennsylvanian series.

Description of the Productive Sands

In the Cushing field, oil is being produced from six different sands—the Layton, Jones, Wheeler, Skinner, Bartlesville, and Tucker.

The Layton sand is found at depths ranging from about 1200 to more than 1500 ft. (365 to 457 m.), the depth depending on the locality. It is productive of oil principally in the northern part of the Cushing field, in the district south and east of Drumright, and in an area a few miles south of Shamrock. Generally it underlies a hard limestone, 10 to 20 ft. thick, called by the drillers the Layton lime, in contrast to the Layton sand, which is a soft sandstone not fully saturated with oil. The top of the sand is at many places barren and the "pay" sometimes lies in streaks, a condition probably due to differences in porosity and to intraformational barriers caused by cross-bedding. The maximum thickness of the sand reported is about 100 ft., and the average of many reports is about 50 ft. At a few isolated points no Layton sand has been found. The sand is coarse-grained, porous, and comparatively soft, and is fairly uniform in texture and porosity. In the Cushing field about 14 square miles of this sand produced oil, and 12 square miles when first drilled into carried much gas.

The Jones sand lies about 200 ft. below the Layton sand and pro-

[2] Frank Buttram: The Cushing Oil and Gas Field, Oklahoma. *Oklahoma Geological Survey, Bulletin* No. 18 (1914), 43.

duces oil in commercial quantities only in a small area on the south side of the dome in the north part of the field, although it contains a little oil and gas at many other localities. The sand at a few places is as much as 50 ft. thick, although the average thickness lies between 15 and 35 ft.

About 100 ft. below the Jones sand is the Cleveland sand, from which, so far as the writer knows, oil has never been produced in commercial quantities, although, like the Jones sand, it contains some oil and gas at many localities. The Cleveland sand is thinner than the Jones sand and is not reported in some logs. Its thickness ranges from a few feet to 30 ft.

From 600 to 900 ft. (183 to 274 m.) below the Layton sand is the next commercially productive formation, known as the Wheeler sand, named from the Wheeler farm a short distance northeast of Drumright, where it was penetrated by the first well drilled in the Cushing field. This sand is one of the most uniform in the field in thickness. It includes the overlying "Wheeler lime," from which it is separated by a shale "break," and ranges in thickness from 50 to about 100 ft. The lower sandy member is correlated by the drillers with the "Oswego lime" of northeastern Oklahoma and southeastern Kansas. It is a coarse-grained, brownish limestone that includes porous or sandy layers which contain the oil. The part below the shale "break" is more porous than the part above it and comprises about half the formation. At some places the limestone above the shale "break" carries gas in commercial quantities and at others, on the sides of the folds, it carries water. The shale "break" between the two members ranges in thickness from 5 to 25 ft. In the Cushing field about 11 square miles of the Wheeler sand produced oil and about 21 square miles produced gas, exclusively. The Wheeler sand, like the Layton and Bartlesville sands, is spotted.

The Skinner sand lies 250 to 400 ft. below the Wheeler sand and locally produces oil in the northern part of the Cushing field and at a few places near the center of T. 17 N., R. 7 E., southeast of Drumright.

From 350 to 550 ft. (106 to 167 m.) below the Wheeler sand lies the Bartlesville sand, the most productive oil sand in the Cushing field. This sand ranges in thickness from a few inches to about 200 ft. and in porosity from compact brown shale to lenses of porous brown coarse-grained sandstone. In different wells it varies greatly in thickness, texture, and porosity and in its content of oil, gas, or water. In some wells in the north part of the field it attains a thickness of over 200 ft., and one part of the sand may be "dry," another part may carry great volumes of gas under tremendous pressure, still another part may furnish great quantities of oil and others may be filled with salt water. These so-called "streaks" are probably due to differences in porosity and apparently occur in no regular order, salt water under great pressure being found in some wells above excellent oil "pay," below which more water may be

found and below that more "pay." Notwithstanding these facts the sand has been immensely productive, though its yield has not been so great as might be expected from a body of sand so thick, for the real "pay" may form only a small percentage of the total thickness of the sand. The total oil and gas area in the Bartlesville sand is about 20 square miles, of which but 2 square miles carry gas only.

The Tucker sand lies from a few feet to about 200 ft. below the Bartlesville sand, and is thought by some to be a part of that sand. The principal area in which oil is produced from the Tucker sand lies near Drumright and is not large. This sand is uniform in porosity, medium-grained, and blue or bluish-green, and although its average thickness is perhaps less than that of the Bartlesville, not enough wells have been drilled through it to determine this question. Many of the logs show that the Tucker sand is separated from the Bartlesville sand by a thin bed of green shale, which is recognized by the drillers as a "marker."

Structure of the Cushing Field

The dominant structural feature in the Cushing field is a broad north-south anticlinal fold along whose axis there are domes and along whose sides there are many subsidiary folds and irregularities. The structure contours of the three oil sands are very irregular and differ locally from the contours of the Pawhuska limestone, although the general structure axes are practically coincident. Each sand in the field exhibits small irregularities that apparently bear no definite vertical relation to each other. A detailed description of the folding of the various sands contoured and of the surface beds may be found in the forthcoming bulletin on this subject.[3]

Through the construction of the contour maps showing the folding of the three oil sands, it has been possible to determine in detail the differences in folding among these beds and the surface beds. The amount of folding in different directions from the crests of various domes and anticlines is given in Table 1.

In practically every case where the dip has been measured, folding becomes greater with depth. The greatest difference in folding usually occurs between the surface beds and the Layton sand.

The possible causes for the irregularities observed may be any one, or a combination of two or more, of the following:

1. The difference in resistance to compression of the hard and soft beds of which the formations in the Cushing field are composed.

2. The lenticular form of the Bartlesville sand.

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[3] Carl H. Beal: Geologic Structure in the Cushing Oil and Gas Field, Oklahoma, and Its Relation to the Oil, Gas, and Water. *U. S. Geological Survey, Bulletin* No. 658 (1917).

3. One or more unconformities between the surface beds and the Bartlesville sand.
4. Folding during deposition.
5. Cross-folding.

TABLE 1.—*Elevation in Feet of Highest Contour on the Surface Beds, the Layton Sand, the Wheeler Sand, and the Bartlesville Sand, on the Crests of Four Folds in the Cushing Field; also the Dip of These Beds in Different Directions from the Crests of the Folds*

(+, above sea level; −, below sea level)

	Dropright Dome				Shamrock Dome		
	Crest	2½ miles N. E. of crest along anticline	1½ miles west of crest	¾ mile east of crest	Crest	1½ miles west of crest	¾ mile east of crest
Surface beds (Pawhuska limestone)	+1,050	100	150	125	+1,125	135	60
Layton sand	−400	225	175	*250	−325	325	250
Wheeler sand	−1,075	225	200	*350	−1,150	175	200
Bartlesville sand	−1,525	175	200	200	−1,550	325	250

	Mount Pleasant Dome					Anticline in Northern Part of T. 16 N., R. 7 E	
	Crest	2½ miles west of crest	4 miles west of crest	¾ mile southwest of crest	¾ mile northeast of crest	Crest	½ mile east of crest
Surface beds (Pawhuska limestone)	+1,100	150	225	75	75	+1,050	50
Layton sand	−275	300	425	325	325	−400	75
Wheeler sand	−1,100	350	475	325	400	−1,325	50
Bartlesville sand	−1,450	350	†475	325	400	−1,700	100

* Part of vertical distance estimated.
† Part of horizontal and vertical distances estimated.

The Interval Between the Layton and Bartlesville Sands

The well in which is found the smallest interval between the Layton and Bartlesville sands lies on the west side of the Dropright dome in the northern part of the field; the interval here is 945 ft. (288 m.). The largest interval observed is 1366 ft. (416 m.) and lies in the southern part of the field. The gradations between these limits is irregular locally, but the increase from north to south is very noticeable in general. The map prepared showing the locations of areas where wells had certain intervals shows plainly that a definite relation exists between the folds and the thickness of the beds between these two sands, the interval being generally less on the crests of the folds. This relation would naturally be

expected if the Bartlesville sand were more steeply folded than the Layton sand, and to arrive at a conclusion as to the cause of the differences in interval, it has been necessary to study the causes for the difference in folding of the various oil sands. The possible causes for the differences in folding have been enumerated above. It must not be thought, however, that the general increase in the Layton-Bartlesville interval from north to south can also be attributed to differences in folding. This increase is undoubtedly due to the conditions under which the formations between the Layton and Bartlesville sands were deposited.

The Direction of Migration of Oil and Gas

As a result of the study of the relation of the oil, gas, and water, it was observed that the more important oil areas in each sand lay on the west side of the Cushing field, and gas in many cases had been forced to the east side of the structure, sometimes as far as the water line.

The hydraulic theory of the accumulation of oil and gas is based on the assumption that the hydrocarbons were concentrated by bodies of water that moved through the sand, and although the determination of the direction from which the Cushing field derived its oil and gas was not the primary object of the investigation, it was evident from the distribution of the oil and gas bodies that the hydrocarbons in this region probably migrated from the west or northwest.

The Relation of Oil and Gas to Structure

The maps prepared show that the areas containing only gas conform closely to the higher parts of the anticlines and domes, and the oil areas lie farther down on the sides of the same structures. The areas overlap in each sand and have narrow irregular strips in common, where both oil and gas were originally found.

In connection with this phase of the subject, a study was made of the initial productions of the oil wells. The initial productions of a large number of wells drilled to the three sands studied were plotted on the map showing the distribution of oil and gas, and lines drawn to outline the areas where wells had certain initial productions. Representative areas of each sand show that most of the places of greater initial productions were on domes or anticlines, or that they correspond, in part at least, to areas on the sides of the folds, where both oil and gas are found in the same sand. For instance, some of the more important oil areas in the Layton and Wheeler sands lie down on the west side of the anticlines whose higher parts are completely filled with gas, and, in one noteworthy instance, the most prolific area of oil in the Layton sand conforms closely to the area wherein both oil and gas occur in this sand. On one side of

the area of high oil production, wells with small initial productions were found without any marked amount of gas, whereas on the other side gas occurs without oil.

Peculiarities in the Distribution of Oil and Gas

A very interesting, and economically important fact, has been determined from a study of the distribution of the oil, gas, and water bodies in connection with the anticlines and domes in the Cushing field. In an elongated dome where folding is simple, the oil area in a sand apparently extends farther down on the long axis of the anticline or dome than it does on the steeper sides. Practically every important dome in the Cushing field, which has not been complicated by folding along its sides, shows this feature in each of the three oil and gas sands studied. The fact is shown particularly well in the Layton sand on the Dropright dome (Fig. 1) which lies in the north part of the Cushing field; on the steep west side water completely occupies the sand at a depth of 500 ft. (152 m.) below sea level, whereas on the gently dipping north side the oil extends down as far as 650 ft. (198 m.) below sea level and also completely occupies the Wheeler saddle on the south side of the Dropright dome, the lowest point of which is about 575 ft. below sea level. Another interesting occurrence is in the Wheeler sand on the Mount Pleasant dome, which lies east of the center of the field. This dome has a northwest-southeast axis and the lowest gas on the steep southwest and northeast sides is about 75 ft. higher structurally than the lowest gas on the southeastward plunging axis. No oil occurs in this sand on the Mount Pleasant dome.

Much detailed work might be profitably done, both experimentally and in the field, in connection with this interesting subject which obviously is of economic importance. If it can be determined that on domes and anticlines, folded to a certain degree of intensity, the oil is universally found lower on the long axis than on the steeper sides, this fact will be of great aid to geologists in the work of selecting lands for leasing, locating test wells, estimating the oil content, and in considering the mode of accumulation of petroleum and natural gas.

Inclination of Water Surfaces

The expression "water surfaces," as here used, denotes the level or inclined plane between the oil and gas in a sand and the edge water upon which the oil and gas rest. This water surface must not be confused with the surface of the ground water, which in an undulatory region is not level, and which is defined as a surface below which the rocks are saturated with water. The water surface here referred to is a surface that is confined to one porous stratum, and has been referred to previously by other writers as the "water level."

It was decided at the outset of the inspection work in the Cushing field to collect all the available information about edge water, and if possible to reach some conclusions as to its rate and method of encroachment and its original and present level in each sand on all sides of the Cushing field. Accompanying the bulletin prepared for publication by the U. S. Geological Survey, are maps showing the contours on the water surfaces in the three different sands. The contours do not show accurately the detail of the water surfaces, and the only object in drawing them was to show that the plane of separation between the oil and the water in a given sand is not level, but dips at a gentle angle away from the center of the structure.

The most complete information has been obtained on the Layton water surfaces in the northern part of the field where the maximum dip of the water surface in the Layton sand amounts to about 100 ft. in a mile. On the west side of the Wheeler saddle which lies north of the center of the Cushing field, the contours show a western dip on the water surface of about 90 ft. in a mile. In this locality similar dips were determined on the water surface in the Wheeler sand. In the southern part of the field, in the more recent development, the water surfaces show dips of about the same amount. On account of the complexity of the composition of the Bartlesville sand the information in connection with edge water in that sand is of much less value, and the detailed studies have been confined almost entirely to the Layton and Wheeler sands.

In searching for the reason for the inclination of the water surfaces, the first point necessary to settle was, whether or not the inclination existed originally, prior to the development of the field, or was wholly the result of the extraction of oil and gas, and also, if it existed originally before the field was exploited, whether it was rendered greater by the extraction of oil and gas. If the inclination existed before the field was developed it might have a significant connection with the mode of accumulation of the oil and gas; if it was acquired since development began, a study of its causes might contribute to our knowledge of the drainage of oil sands; if it existed prior to the development and was made greater by the extraction of oil and gas, its study might afford valuable conclusions both as to the accumulation of oil and gas and as to the drainage of the sands. The methods used in determining these problems will not be discussed in this report because the discussion has been given in *U. S. Geological Survey Bulletin* No. 658, referred to above. The conclusion arrived at was that the inclination of the water surfaces in the Layton and Wheeler sands existed prior to development, and was rendered greater by the rapid extraction of oil and gas. In some places the subsequent inclination was as great or greater than the dip of the oil sands, from which it was concluded that some of the wells near the centers of the more productive areas which lie on the crests of the domes and anticlines would begin to

show water before the wells on the flanks of these folds, and after the wells on the crest had gone completely to water there might still be a small commercial deposit of oil remaining on the sides of the folds.

The Necessity of Similar Work

The results obtained from this study, which was necessarily hasty and carried on without the benefit of all available information, are thought to have been well worth the effort of collecting and analyzing the information. The writer believes that much more detailed work might be done in this and in other oil fields, with very far-reaching and conclusive results, and has submitted the results of this study with the hope of stimulating similar and more detailed investigations. Undoubtedly our knowledge of the accumulation of oil and gas and the changing conditions under which they exist during development, and during the productive life of a field, is slight, and much work of this sort must be done before sufficient facts are available for the formulation of any tenable theories to account for many of the at present unexplainable phenomena encountered in oil fields.

Granite in Kansas Wells

BY PARK WRIGHT, ST. LOUIS, MO.

(St. Louis Meeting, October, 1917)

THE fact that granite has been encountered by the drill by those in search of oil and gas in Kansas is becoming more and more a matter of interest, not only to the oil producer but to everyone directly and indirectly connected with the oil business. It is, therefore, time that someone should offer an explanation for its presence and form a definite opinion, based on all the data available, in regard to the effect it will have on the future prospects of the area affected as probable oil- and gas-producing territory. At present there seems to be no consensus of opinion. To the writer's knowledge, only one attempt has been made to explain; by Erasmus Haworth, State Geologist of Kansas, in a paper entitled *Bulletin Number Two on Crystalline Rocks in Kansas.*

The purpose of the present paper is to offer a theory for the presence of the granite, and to draw conclusions which will aid in determining the prospective oil and gas value of the area in question.

In taking up this question it will be well to consider first the tests in which granite has been reported, with a general history of each. The accompanying map (Fig. 1) shows the location of all wells described.

The first well in which granite was reported was that drilled just south of Zeandale in section 27, township 10 south, range 9 east, by a local company which had taken up a block of leases in that vicinity. At a depth of 958 ft. (291.99 m.) granite was encountered. Drilling was continued to a depth of 1093 ft. (333 m.) in the same formation, at which depth the hole was abandoned. Another well was started 1 mile east of the first well and the same rock was found at a depth of 945 ft. (288 m.). Drilling was continued to a depth of 1200 ft. (365.76 m.) with the bit in the same formation. Samples of the drill cuttings have been examined and they are undoubtedly crushed granite.

The next well in which granite was found was that drilled by DeLaat and Shepard, independent wildcat operators, in section 34, township 19 south, range 7 east, near the town of Elmdale. Granite was struck at 1707 ft. (520 m.) and the hole was abandoned in the same formation at 2525 ft. (769 m.). Mr. DeLaat watched the drilling of this well very closely and he has described to me in some detail the formation immedi-

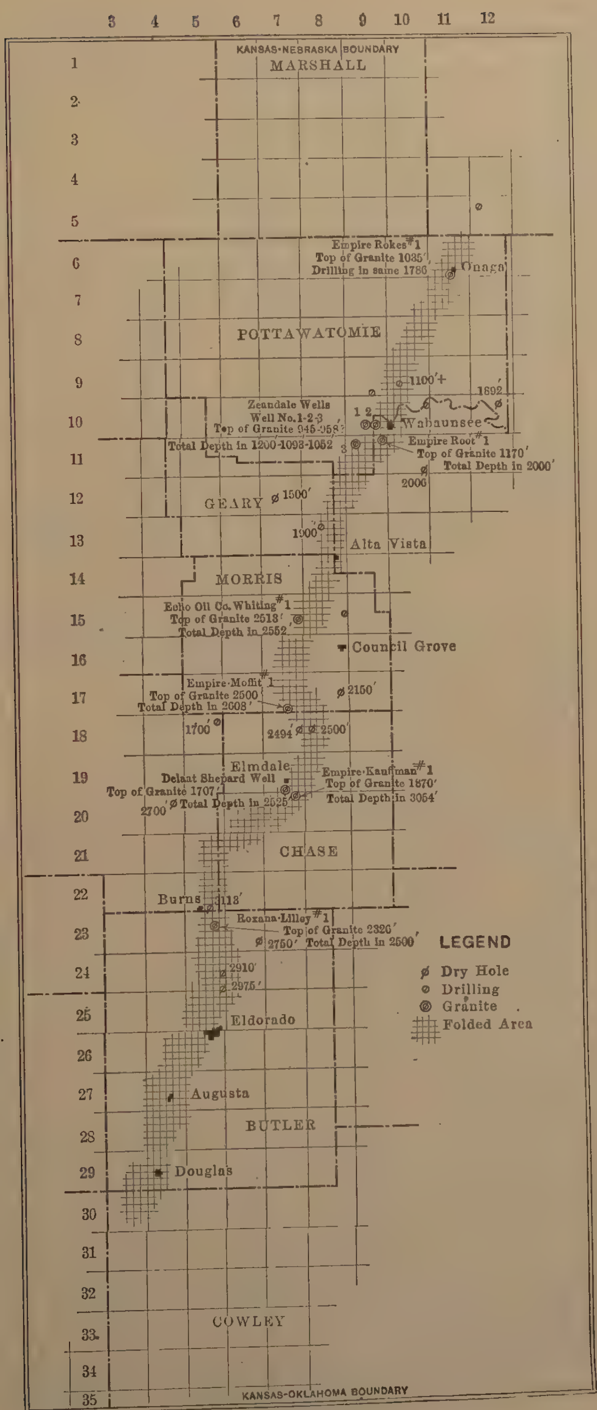
KANSAS-NEBRASKA BOUNDARY
MARSHALL
Empire Rokes #1
Top of Granite 1035'
Drilling in same 1786
Onaga
POTTAWATOMIE
1100'+
1692
Zeandale Wells
Well No. 1-2-3
Top of Granite 945-958'
Total Depth in 1200-1093-1052
Wabaunsee
Empire Root #1
Top of Granite 1170'
Total Depth in 2000'
2000
GEARY
1500'
1900
Alta Vista
MORRIS
Echo Oil Co. Whiting #1
Top of Granite 2518'
Total Depth in 2552
Council Grove
Empire Moffit #1
Top of Granite 2500
Total Depth in 2608'
2150'
1700
2494
2500'
Elmdale
Empire Kauffman #1
Top of Granite 1870'
Total Depth in 3054'
Delaat Shepard Well
Top of Granite 1707'
2700'
Total Depth in 2525
CHASE
Burns
2113'
Roxana Lilley #1
Top of Granite 2326'
2750'
Total Depth in 2500
2910'
2975'
Eldorado
Augusta
BUTLER
Douglas
COWLEY
KANSAS-OKLAHOMA BOUNDARY
LEGEND
Dry Hole
Drilling
Granite
Folded Area

FIG. 1.

ately overlying the granite and the nature of the granite. They had about 30 ft. (9.15 m.) of cave just overlying the granite, which was composed of red mud and smooth worn slivers of limestone, the whole in an unconsolidated condition. There were no signs of contact metamorphism. The surface of the granite was badly decomposed, showing signs of weathering but unaltered below the surface.

Several wildcat wells were later started by the Empire Gas & Fuel Co., encouraged by their great success in the El Dorado and Augusta Pools. The test in section 2, township 20 south, range 7 east, was the next reported to be drilling in granite. It was encountered at 1870 ft. (570 m.) and drilling was continued to a depth of 3054 ft. (930.86 m.) where the hole was abandoned. A sample of the cuttings has been examined, and it is undoubtedly the same as that in the other wells.

The same company's test in section 1, township 11 south, range 9 east, got the granite at 1170 ft. (356.6 m.) and the well was abandoned at 2000 ft. (609.6 m.) in the same formation. Near Onaga in section 34, township 6 south, range 11 east, the Empire Gas and Fuel Co. well was drilled into granite at 1035 ft. (315 m.) and abandoned at 1734 ft. (528 m.).

In section 34, township 17 south, range 7 east, the Empire well drilled into granite at 2500 ft. (762 m.) and the hole was abandoned at 2608 ft. (794.9 m.) in the same formation.

The Echo Oil and Gas Co. in its test in section 24, township 15 south, range 7 east, reached the granite at 2513 ft. (765.96 m.). The hole was abandoned while drilling in the same at 2552 ft. (777.85 m.).

The Roxana test in section 14, township 23 south, range 5 east, just north of the El Dorado pool drilled into granite at 2326 ft. (708.96 m.) and the hole was abandoned at 2500 ft. (762 m.).

From the above data, it will be noted that there is no regularity in the depth of the granite and in no place has it been drilled through, although one test was drilled 1184 ft. (360.88 m.) into it. At Elmdale, shallow gas was found and in the Roxanna test near Burns a small showing of oil was found in the 2200-ft. (670.56-m.) sand. Outside of these two tests, no showings have been encountered above the granite in any of the wells drilled to it.

A very significant point is that all of the above-mentioned wells are located on or very close to the axis of an anticline and in no place has granite been found in wells drilled on either side of this continuous line of folding extending in a northeast-southwest direction across the State. Contrary to this, in several cases deep wells have been drilled favorably located on this folding which did not encounter granite. This would make it seem that if there is any direct relationship between the granite and the folding it cannot be based on the fact that it has never been found any place except on the structure without explaining the failure of wells

to reach the granite which are favorably located with reference to the structure. It is therefore necessary to account for the results of all the wells drilled in the area under consideration in order that an explanation of the presence of the granite and an estimation of its probable extent will hold good.

There are three possible modes of occurrence of the granite. We will assume that it is an igneous intrusion subsequent to the deposition of the overlying beds, a sedimentary formation composed of granitic fragments, and an ancient ridge present prior to the deposition of the overlying beds. For each assumption, the corresponding surface expressions we would expect to find will be briefly outlined so that we may eliminate those that do not satisfy the conditions as they actually exist.

If we assume an intrusion subsequent to the deposition of the overlying rocks, the chance of finding surface exposures of the granite in places where the molten rock has pushed through would be highly probable. The surface beds would show very complex folding accompanied by a larger number of faults and a general shattering of the formations due to the fact that there would be great tensile stresses produced by the pushing upward of this great mass of molten rock. The effect of contact metamorphism of the overlying beds would be evident from the drill cuttings. The surface of the granite would very probably be much finer due to the more rapid cooling of the molten mass coming into contact with the stratified rocks.

If we assume that the granite is a clastic formation, as Mr. Haworth has done in his report, classifying it as a "modified form of hard sandstone with unusually larger amounts of feldspar present, as in the Phillips County stone, cemented into this hard rock by some of nature's processes," it must be equivalent to a similar sedimentation which would be encountered in other wells over a large area which have reached the same horizon, if deposited during Pennsylvanian times. We would expect to find a more or less constant interval from certain known surface formations, especially in relatively short distances. However, if this formation is arkose, laid down prior to deposition of the Carboniferous, the above condition of constant interval would not necessarily be expected on account of the probabilities of an erosional unconformity. In either case the drill cuttings should show signs of decomposition and water-worn pebbles.

If we assume that the granite is from an ancient mountain range present prior to the deposition of the overlying beds, we would expect to find the granite body relatively extensive and continuous.

The chance of finding surface exposures would not be so great as in the case of an intrusion. The overlying beds would very probably be folded along its entire extent, forming a continuous line of structure made up of rather simple folds accompanied by little, if any, faulting. In-

stead of metamorphism at the contact of the granite with the overlying beds, we might expect an unconsolidated formation made up of the ingredients of the overlying beds, together with a very small percentage of granitic material due to the abrasion of the ends of the beds against the face of the granite, if folding had taken place. The surface of the granite would show signs of decomposition and weathering. Samples of the main body below the surface would show fresh, sharp angular cuttings.

Having considered the various possible modes of occurrence, the actual conditions shown by the surface rocks and those brought to light by the drill will now be presented. There is a continuous line of folding extending across the State in a northeast-southwest direction from township 6 south, range 11 east, to township 31 south, range 3 east, an air-line distance of approximately 153 miles (246.23 km.) It is very probable that this structure extends north into Nebraska, but the writer has no proof of this, not having examined that territory. The fact that the granite in all cases to date has been found only along this continuous line of structure certainly must have some significance, and if so, this continuity must be due to some cause out of the ordinary. The shape of the folding from Cottonwood Falls north is entirely different from that found in any other place not affected by the presence of granite at shallow depths. The rocks to the east of the fold dip very gently at the rate of 15 to 20 ft. to the mile, into a very flat syncline which has practically no depth in comparison with the height of the anticline. From this flat syncline they rise abruptly at the rate of about 200 ft. to the mile, to the crest of the anticline. After passing over the crest they gradually dip off to the west again at the rate of about 25 ft. to the mile. This shape of structure is characteristic of all folds where granite has been found except in the Roxana test on the Burns dome, where it is not so pronounced and which is probably formed under slightly different conditions to be explained later. More than three-fourths of this structure has been mapped in detail and not a single fault has been found.

The overlying beds show no signs of contact metamorphism. All the formations are found regular and unchanged until a depth is reached just above the granite, where a bad cave is encountered. The amount of this cave varies in thickness. Samples of this cavy formation show an unconsolidated mass of broken pieces and long smooth slivers of limestone intermixed with shale. After passing through this the drill comes into contact with the granite, which, in most of the samples examined by the writer, shows signs of decomposition and weathering. After the surface is drilled through, the cuttings show a very fresh unaltered granite. In two wells a break occurred in the granite that has been called slate. Concerning this the writer is very doubtful. The maximum depth the drill has penetrated this granite is 1184 ft. (360.88 m.) and so far it has not been drilled through. The top has been encountered at various

depths below the Winfield limestone from 1427 ft. (434.95 m.) at Onaga to 2728 ft. (731 m.) at Council Grove, a range of 1300 ft. (396 m.). At no place has the granite been found exposed at the surface.

It is plainly evident from the above data that the only logical assumption to be chosen from the three probable assumptions presented in regard to the mode of occurrence of the granite, is the old range theory. There is more evidence to support the stratified rock theory than the intrusion theory, but hardly enough to balance the evidence against it. Proceeding with the assumption that a granite ridge was present prior to the deposition of the overlying beds, it is very important to ascertain just what relation the folding bears to the granite ridge, so that we may form some opinion concerning the probable extent of the granite from the surface. As stated above, in all places where granite has been found, the rocks above have been folded in such a manner that the shape of the anticline and syncline formed is so different from the ordinary structure, that one is led to believe that it is nothing more than the surface expression of the granite below. If, then, we can account for this extraordinary folding and its relation to the granite, it will be possible to estimate the probable extent of the granite within the reach of the drill by determining the extent of the "granite structure." In order to do this it will be necessary to work out a satisfactory theory for the various stages of development of an anticline localized by a granite ridge present prior to the deposition of the overlying beds.

In taking up this problem we will consider five stages of development. The first stage may represent the granite ridge present prior to the deposition of the overlying beds. During the second stage, deposition takes place and the beds are in the process of consolidation. During the third stage, a slight arch is formed in the beds, due to the differential consolidation of the beds under the increased load of the overlying strata on either side of the granite ridge. During the fourth stage, this arch gradually becomes more pronounced with the first effects of the lateral pressure from the east caused by the readjustment of the beds after the Ozark uplift, which seems to be the cause of all the folding along the westward dipping monocline over the greater part of Kansas. During the fifth and present stage, added pressure has caused the ends of the beds in contact with the east face of the granite and those extending over the top, to slip along the face, leaving the products of the abrasion, composed of smooth slivers of limestone mixed with the shale, along the contact between the overlying beds and the granite. This condition is reasonable enough when one considers the fact that the strata against the side of the barrier which offers resistance to the lateral pressure will be most affected and will show the result of this resistance in the way suggested. The surface expression will be a sharp east dip. On the other hand, the west flank will be practically unaffected because the difference in resistance to the lateral thrust

above the granite would be so great that there would be a tendency for beds to gently "roll over" the crest and dip off gently to the west. This jamming of the beds against the granite buttress will undoubtedly be accompanied by a compression and flowage of the intervening shales which might have otherwise caused an overturned fold. The lack of faulting is probably due to the easing up of the early thrusting by the initial dip caused by the differential consolidation of the beds. The final result of these forces acting in this way will be the formation of a fold with a broad syncline, a strong east dip and a very gentle west dip just beyond the crest, which is the condition existing in the folds where the granite has been found.

From the same line of thought it is reasonable to suppose that, within certain limits, the high places along the granite ridge will have some direct effect on the general shape and height of the structure of the overlying rocks. We would therefore expect to find the granite at the shallowest depths on the highest parts of the domes. This principle, of course, is of no value in estimating the probable depth in a proposed well. It merely serves to determine the relative chances a well has of encountering granite, given its location on the structure.

It is, therefore, quite possible to fail to get the granite in two wells along the axis of an anticline, and encounter it at a shallower depth on the top of the dome, as in the case of the Roxana test on the Burns dome. One well to the north was drilled to 3113 ft. (948.84 m.) and another just south to 2910 ft. (887 m.). Both wells failed to reach the granite, yet the Roxana well drilled into it at a depth of 2326 ft. (709 m.). This seems to show that the granite under the Burns dome comes to a high peak, causing a corresponding high place in the structure.

An examination of the shape of the folding from the south line of township 24 south, range 5 east, southwest to the end, shows none of the characteristics of the folding where granite has been found at shallow depths. It is impossible to say whether the granite extends south under this structure at deep depths, because in either case it would not be expressed in the surface structure. Moreover, this long line of folding would not be cut off sharply at the end of the granite ridge but would extend beyond the end some distance, and gradually die out when it reached a point where the forces tending to continue the folding along the line localized by the granite were insufficient. The effect of this long line of folding would have a tendency to localize folds even beyond this, which probably accounts for the Blackwell folding in Oklahoma.

With these conditions existing, it is evident that the value of the granite structure for oil and gas is materially diminished because the granite has replaced large portions of the strata which ordinarily would be available as reservoirs from which oil and gas might be produced. Inasmuch as it is impossible to tell in advance of the drill the extent of this

replacement, an element of uncertainty enters which increases the unfavorable chances and therefore decreases the value of the prospect. However, in the face of all these disadvantages, all this structure should not be condemned as unfavorable. The accumulating advantages offered by the anticline may be diminished to some extent but not destroyed. It is quite possible to find productive areas along this granite structure where the conditions are favorable, *i.e.*, where the granite is 3000 ft. (914.4 m.) below the surface for 8 to 10 miles along the axis of the fold. Even though such a place does not exist, it is likely that there are some extensive stretches of sand at shallower depths along the fold, which have not been replaced by the granite and which have been enriched with oil and gas because of the accumulating advantages offered by the anticline.

It is therefore clear that the granite reduces the value of the structure in direct proportion to the areal extent and the depth below the surface of the oil and gas sands, which the granite replaces. It is also reasonable to suppose that the replacement of these large bodies of sand will tend to act as a disadvantage toward the action of accumulation. But, on the other hand, the chance of finding favorable conditions where oil has accumulated is good enough to warrant further development. The El Dorado and Augusta fields, although they may not be deeply underlain by this granite ridge, are examples of accumulation along this same line of folding where the conditions are ideal. It is possible that similar conditions will be found to the north, but in no place is it likely that conditions just as favorable will be found.

Review of Present Knowledge Regarding the Petroleum Resources of South America

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(St. Louis Meeting, October, 1917)

TABLE OF CONTENTS

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INTRODUCTION

Scope of Discussion

There has hitherto been no systematic effort to make public the available information on petroleum in South America and the object of this paper, therefore, is broadly to discuss the subject as it pertains to South America as a whole, endeavoring to collect in readable form what is known regarding individual fields. This article is not intended exhaustively to discuss individual fields, or to solve the many complicated problems regarding petroleum and natural gas in the various republics, since those necessitate years of study and work by petroleum geologists, engineers, financiers, and drillers. Neither can the reader expect absolute accuracy in all statements, since this is impossible at the present time. The writer has attempted to be correct in all cases, but in a review involving so many countries and extending over so many phases of the subject, it is to be expected, of course, that some errors will have crept in.

Literature and Acknowledgments

Little of a reliable nature has been published on petroleum in the South American countries, leaving the general public in great ignorance of

conditions. Recently, however, an admirable paper has appeared by Ralph Arnold,[1] who has summarized the state of development in the respective countries and his ideas as to the future of the industry. Sir Boverton Redwood, in the various editions of his *Treatise on Petroleum*[15] has discussed South American nations in a general way, but not even he indulges in the details that will be of interest in the various countries. Many articles have appeared in technical and scientific journals, in the *Pan-American Bulletin* and in the *Mineral Resources of the United States Geological Survey,* which have kept the world informed of any special activity or increase of production. An excellent account of the Peruvian fields has been given by R. A. Deustua,[6] to whom the thanks of the writer are due for looking over and correcting the pages relating to that country. The writer is also indebted to F. M. Anderson for his perusal and correction of pages relating to Colombia.

Geographic Distribution of Petroleum in South America

Properly speaking, there are only two oil districts in South America, that east of the Andes and that west of them; but in order to describe the distribution of petroleum there it is desirable to classify the fields and known favorable districts into similar groups to those in North America, and the following may conveniently be used:

1. Pacific district.
2. Orinoco district.
3. Caribbean district.
4. Central Andean district.
5. South Atlantic district.
6. Western Argentina district.

1. The Pacific district is a continuation of the California district in North America and includes the many fields and prospective fields in the belt, generally less than 100 miles (160 km.) wide, extending interruptedly from Alaska southward between the Rocky Mountains and Andes and the Pacific Ocean into Tierra del Fuego. Occasionally some fields east of the extreme western range may perhaps be included for convenience in this group.

2. The Orinoco district will include all fields of the Orinoco Valley and Delta, or in general any that may be discovered between the Cordillera de Merida on the northwest and the Sierra de Pacaraima on the southeast.

3. The Caribbean district will be used to include all fields on the

[1] The superscript numbers refer to the corresponding numbers in the reference list at the end of the paper.

Caribbean coast and north of the Cordillera de Merida and Eastern Cordillera in Venezuela and Colombia and east of the Western Cordillera.

4. The Central Andean district includes the fields occupying the plateaus and east Andean slopes in Peru and Bolivia, with possible extensions into Ecuador and Brazil.

5. The South Atlantic district contains the Comodoro Rivadavia fields and other prospective fields on the Atlantic Coast from the Rio de la Plata south to Tierra del Fuego.

6. The Western Argentina district includes the fields lying along the eastern border of the Andes and possibly in some intermontane areas from Bolivia southward to an unknown distance.

These districts may merge in places, occasional new districts be discovered, and the classification perhaps change from time to time, but it forms a convenient grouping for consideration of present knowledge; and since it is based on geographic conditions, it may be changed by additions or further subdivision. It is particularly likely that one or more districts will be added by the discovery of petroleum in Brazil, and that developments on the Pacific Coast may result in some subdivision.

Geographical Comparison with North America

In order to understand the conditions affecting oil distribution in South America, it is necessary to study the geology and geography of the continent in its relation to that of North America. In both continents mountain ranges and other regions of igneous and metamorphic rocks separate regions in which rocks exist of identical age with those which contain oil in North America and Europe.

Most essential to our studies are two belts, as follows:

1. A broad area known technically as the Gulf Coastal Plain, which starts in Southern United States, follows the Gulf of Mexico through Mexico and Central America into Colombia, and bends eastward through Venezuela, and which includes the valuable oil fields of the Caribbean district in these countries. The formations range from Cretaceous to late Tertiary in age, and the oil exists in sands and limestones. The details of geologic structure, or attitude of the rocks, are different in the different countries, but in many ways the general geographic and geologic relations are similar.

2. The second belt is a much narrower one, lying between the Rocky Mountains and Andes and the Pacific Ocean, extending somewhat brokenly from the coast of British Colombia southward through Washington, California, Mexico and Central America, Colombia and Ecuador into Peru. This belt is called for convenience the Pacific Coast district, already mentioned. The rocks commonly are of similar late geologic age

in most of the countries, but the structural relations are different. In order to understand all the complicated phases of the question, therefore, the geological engineer must be familiar with the occurrence of oil in Texas, California, and Mexico, and he must apply his knowledge of North America to the countries of South America according to the particular type of structure involved.

Somewhat separate from the Caribbean district is the Orinoco district, and still further separated is the Andean district covering wide areas on and along the east flanks of the Andes southward from Colombia and Venezuela nearly to Tierra del Fuego.

These two last-mentioned districts may be likened geographically to the Hudson Bay district and the Mid-Continent district in North America, and as an encouraging aspect of the Andean district we may remark that the Mid-Continent district has proved the richest in North America.

The fifth geological belt of importance in relation to oil is the South Atlantic district, which, with the points of the compass reversed, corresponds with the northwest Canada district in North America.

Geologic Inferences Regarding South America

The geological conditions of South America have been little studied from the standpoint of oil occurrence, though some geologists have visited Venezuela and Colombia for private companies, the results not being available for publication. The details of the fields therefore must be left to future exploration. We know that practically all the great oil-bearing rock series of the world exist in South America; and since important oil seepages also exist, we may expect the development of enormous fields. Strata of Carboniferous, Cretaceous, and Tertiary ages, which yield oil in North America and Europe, may be expected to do so likewise in South America.

Quality of South American Oil

While some fields of light paraffine oil exist in South America, the great preponderance appears to be of asphaltic or mixed base, which may be fortunate, since this is more valuable as fuel oil, which will be much needed in the coming decade. The quality of petroleum must be considered from various aspects, as, for instance, specific gravity, paraffine or asphalt base, results of fractional distillation, calorific value, etc. For comparative purposes, Table 1 is introduced, giving the specific gravities and calorific values of representative petroleums the world over:

TABLE 1.—*Specific Gravities and Calorific Values of Representative Petroleums*

Field	Specific Gravity	Baumé	Calorific Value (B.t.u.)
Texas (Sour Lake field)	0.964	15.0	
Louisiana	0.939	19.0	19,300
Texas (Spindletop field)	0.920	22.5	19,574
California (Coalinga field)	0.915	23.5	17,500
Wyoming (Salt Creek)	0.909	24.0	
Russia (Baku field)	0.884	28.5	20,600
Galicia	0.870	31.0	18,000
Burma	0.869	31.0	19,250
Oklahoma (Bartlesville)	0.859	33.0	
Canada (Petrolea field)	0.858	33.0	
Roumania (Bushtenari field residue)	0.854	34.0	19,600
Peru (Negritos field)	0.850	35.0	19,445
California (Pico Cañon)	0.850	34.5	18,675
West Virginia	0.841	36.5	18,400
Ohio (Lima field)	0.839	37.0	
Roumania (Campina field)	0.836	37.5	19,900
Pennsylvania (Bradford field)	0.819	43.0	
Pennsylvania (Washington field)	0.800	45.0	
Italy (Parma field)	0.786	48.5	18,200

This is, of course, only a very brief list, but serves as a basis for comparison with oils that may be found in South America. Unfortunately, we have no available record of the calorific value of South American petroleums, aside from Negritos petroleum from Peru.

South America's Position in the World's Production

All five continents of the world contain petroleum, and are, in the order of importance of the present developments, as follows:

1. North America.
2. Europe.
3. Asia.
4. South America.
5. Africa.

The East and West Indies are also producers. Every indication exists that South America will probably rise, in the course of a few years, to second place in this list; although in the opinion of the writer, the final order of importance of the continents by total production, when fully developed, is likely to be as follows:

1. North America.
2. Asia.
3. South America.
4. Europe.
5. Africa.

It is possible that North America will be exceeded in production by Asia or South America, but this is not deemed likely.

Remarks on Political Conditions in South America

One of the greatest difficulties or hindrances in development of South American oil fields has been the unstable nature of certain governments, and frequently the more or less unfriendly attitude of these governments toward outside capital. It should be realized that the improved stability of the governments will result in an influx of capital from North America, which will be of great benefit to the future welfare of the various republics. To attain the foremost position they must encourage the influx of capital; and while providing for the payment of small royalties, must not exact taxes or royalties that will discourage or delay exploitation.

PERU

1. Distribution of Peruvian Fields

Arnold[1] has grouped the Peruvian fields into two general districts as follows: (A) that of the Andes, which contains the

1. Titicaca field;

and (B) the Pacific Coast district, which includes the following:

2. Zorritos field.
3. Lobitos field.
4. Negritos field.

In accordance with our own practice, we might more properly speak of the Zorritos, Lobitos, and Negritos fields as "pools;" since they all lie in the Department of Piura, but this distinction is unimportant for practical purposes. Arnold[1] quotes C. M. Hunter[10] as giving (in a personal letter) the total area of oil territory in Peru as over 5000 sq. miles, of which 100 sq. miles have oil possibilities, and 200 sq. miles are proved. In 1915, there were 524 wells in the whole country, of which 90 had been sunk the previous year.

Petroleum is also found in the districts of Pirin, Chimbote, Jauja, Huancavelica, and Ica; in fact, it is said to occur at frequent intervals from Tumbez all the way to Lake Titicaca. In 1915, the following petroleum companies were operating in the country:

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[1] References are to bibliography at end of paper.

Bayovar Oil Development Co., Ltd., owning 558 pertenencias (claims) near Payta, but not having started operations.

Establecimiento Industrial de Petroleo de Zorritos, owning fields of 17 sq. miles at Zorritos, 27 wells producing. The output in 1913 was 3,500,000 gal. It has a refinery at Zorritos.

Lagunitas Oil Co., Ltd., owning 8 sq. miles, and situated 11 miles from Talara, under lease for 35 years from the London & Pacific Petroleum Co. The 1915 output exceeded 2500 tons a month.

Lobitos Oilfields, Ltd., owning 11,000 acres between Lobitos and La Cruz; also owning stores and three tank steamers. The production has been as follows:

	Tons
1911	52,172
1912	78,272
1913	74,314

London & Pacific Petroleum Co., owning extensive leases in the Province of Payta, partly subleased to the Lagunitas Oil Co. It also owns a fleet of tank steamers and has a refinery at Talara.

Titicaca Petroleum Co., operating in the Titicaca field.

West Coast Fuel Co., Ltd., owning storage tanks, pipe lines, etc., at Payta and Callao.

A. Central (Andean) District

Oil is found at several points in the Andes, as in the Huallanca region, in Cerro de Pasco, and in the Provinces of Jauja and Parinacochas. The principal known deposits are near Lake Titicaca.

1. *Titicaca Field.*—This field is situated in the district of Puno, 300 miles (482 km.) from the Coast, 8 miles (12 km.) distant from Lake Titicaca, high in the Andes, and near the Bolivian frontier, extending from Cuzco to the boundary of Bolivia. The deposits are found in the Provinces of Canas, Lampa, Azangaro, and Huancane. Exploration was most extensive in the region of Pirin in the district of Pusi, Province of Huancane, about 5 km. northwest of the town of Pusi. The work has been done by the Titicaca Petroleum Co., a United States corporation which discovered oil in 1906, and which in 1912 had sunk 10 wells reported producing an average of 50 bbl. per day, the oil being paraffine base. The Pirin and Corapata localities are only a short distance from Juliaca Station of the Southern Railway near the Bay of Escallani on Lake Titicaca. At Corapata, a company composed of Peruvian and Chilean capitalists has carried on exploration work. The Titicaca field now appears to be abandoned.

B. Pacific (Coastal) District

The coastal belt extends southward through a desert waste of 2000 miles (3218 km.), from the Ecuadorean frontier to Chile. The developed fields, however, extend only from the town of Tumbez, south of the Gulf of Guayaquil, for 180 miles (289 km.) to and beyond Paita, being bounded on the east by spurs of the Andes Mountains, and on the west believed to include the Islands of Lobos. The field is said by Hunter to be about 30 miles (48 km.) wide and occupies the Province of Tumbez and the northern part of the Province of Piura, but English experts believe it will prove to be 150 miles (241 km.) wide east and west.

The fields are situated between the hills of Illescas, south of the port of Paita in the Department of Piura, and the town of Tumbez, in the north of the province of the same name bordering on Ecuador. The eastern boundary is considered to be the chain of foothills of La Brea or Amotape, constituting outliers of the Andes.

The coastal fields have little vegetation. Rain seldom falls, but the temperature is said to be ideal. All water used is condensed sea water. Wells range from 250 to over 3000 ft. (76 to 914 m.) in depth.

The best geographic description of the Pacific district of Peru is given by Deustua,[6] who says that there are two great regions: the eastern, comprising the La Brea chain of mountains and other spurs of the western Cordillera of the Andes; and the western, which is smooth, and lies along the coast, and includes a series of plateaus. The mountain region forms the eastern boundary of the coastal plains and of the oil-bearing formation of the Provinces of Tumbez and Paita. The plateau to the north of Paita has an elevation of from 50 to 150 m. (164 to 492 ft.), but, further south, it almost declines to sea-level.

2. *Zorritos Field.*—The Zorritos field is the oldest and the most northern in Peru. It is situated only about 24 miles (38 km.) south of Tumbez, south of the Gulf of Guayaquil. The producing territory extends along the coast for about 4 miles (6 km.) most of the wells being drilled at the water's edge and some in the edge of the ocean. The Zorritos field is owned by Faustino G. Piaggio & Co., of Callao, whose property in 1910 covered 2,160,000 sq. m., not all situated in this field, however.

The greatest depth known to have been reached in the Zorritos field is 3020 ft. (920.5 m.) in Peroles ravine; but most wells are between 600 and 2000 ft. (182 and 609 m.) deep. The productions run in some cases as high as 500 to 600 bbl. per day; one-third of all the wells drilled have been failures.

3. *Lobitos Field.*—Next south is the Lobitos field in the Department of Piura, about 60 miles (96 km.) north of Paita. The proved area of this field is about 25 sq. miles (64 sq. km.). It is owned by the Lobitos

[6] References are to bibliography at end of paper.

Oil-fields, Ltd. This is the district of second largest production in Peru. The cost of wells 3000 ft. (914 m.) deep averages about $10 per foot as against $1.50 or $2 per foot for wells under 1500 ft. The deepest well in the field was sunk to a depth of 3435 ft. without success, requiring 18 months to drill. All the wells are over 2000 ft. deep. The shallower ones in this field have been short-lived, but the deeper sand favors greater expectations. In 1915 a new pool was reported opened at Punta Restin, 12 miles north of Lobitos.

4. *Negritos or Talara Field.*—The southernmost and richest of the developed coastal areas is the Negritos field, situated 40 miles (64 km.) north of Paita in the Hacienda La Mina Brea and Parinas. The area of this field is about 650 sq. miles (1683 sq. km.). Negritos is the center of drilling operations, and Talara, 16 miles distant, where the refinery and wharves are situated, has an excellent harbor which is connected with the field by a 6-in. pipe line and 16 miles of narrow-gage railway. At a point 11 miles east of this field is an asphalt seepage called "La Brea." In the Negritos field the first company to operate was the London & Pacific Petroleum Co., the holdings of which covered 650 sq. miles. The average depth of the wells is 2500 or 3000 ft., and the most important oil deposits are below 1500 ft.

2. Surface Indications

The surface indications in Peru consist of:

1. Asphaltic and petroleum-saturated limestones, clays, etc.
2. Association of bitumens with ore deposits.
3. Bituminous dikes.
4. Oil seepages.
5. Submarine petroleum springs.

1. *Asphaltic and Petroleum-Saturaied Limestones, Clays, etc.*

Redwood[15] reports petroleum-saturated and asphaltic limestones at the following localities:

(*a*) Huacota, Huallanca region, near boundary between Ancacho and Huanaco.

(*b*) Huayta-huaru.

(*c*) Colpa.

(*d*) North of Balza and near Quemia, Cocabamba district, Luya Province, this occurrence being somewhat doubtful.

(*e*) Chuquibambilla, in the district of Ayaviri, Province of Lampa (Puno).

(*f*) In the vicinity of Nazca and El Portachuelo Grande, in the central zone.

Asphaltites and asphaltic clays are used extensively as fuel in Peru and the ashes are recovered in many cases for their vanadium content. The asphaltites exist mostly in the central Cordillera region, in Canta, Minasragra and along the Mantaro River.

2. *Association of Bitumens with Ore Deposits*

Redwood mentions elaterite, associated with cinnabar, at Chonta Mines, but this is supposed only to be an indication of the presence of oil in former geological times.

3. *Bituminous Dikes*

Asphaltic dikes are mentioned by Redwood at Huari and Soaro in Tarma Province, Department of Junin, a long distance from any known oil fields. They mean only that oil once occurred in the region.

4. *Oil Seepages*

According to a report of Consul George H. Pickerell, dated Para, Brazil, July 15, 1916:

> "An American citizen who has just arrived in Para after completing a transcontinental trip from Lima, Peru, states that while he was in Iquitos reports were current of the discovery of oil on the Huallaga River, about 2 days' journey from a small town in Peru known as Yurimaguas. Samples of this oil have been secured and found to be of good quality, with the result that various investigations concerning the value of the discovery are under way."

Traces of oil are also reported at several points at Cerro de Pasco and in a belt southward as far as Huancayo, Department of Junin. Seepages are reported as follows:

(*a*) Argascara Ravine at Mito, Jauja Province, Department of Junin.

(*b*) Sascamarca Ravine, Department of Huancavelica.

(*c*) La Brea, Chumpi district, Parinocochas Province (oil spring of some dimensions).

(*d*) Gullies ("quebradas") of Calangache and Ichupalla, northeast of Pirin in the Province of Huancane, Department of Puno.

In the Titicaca field, according to Deustua,[6] "the oil issues in the form of small boiling springs, accompanied by brackish water, having a strong odor of petroleum and sulphurous acid, and depositing around the spring a series of precipitates of yellowish-white sulphur." In the Zorritos field there are filtrations in certain cliffs ("barrancas"), as well as evidences of oil in surface sands. Petroleum is reported also in Condorocana Mountain, Province of Angaraes; in Calaveras Mountains, Province of Camana; and in Chumpi, Province of Parinacochas; but up to date these reports are not confirmed and the character of the indications is in doubt.

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[6] References are to bibliography at end of paper.

3. Geological Conditions

While certain generalizations can perhaps be made, the various fields must, as a rule, be considered individually. Peru is crossed from end to end by several great mountain ranges, connected in places by central peaks, and forming high plateaus. The western and eastern Cordilleras are of different ages, the eastern being the more ancient; the western range or continental divide being of more recent upheaval. We thus see that, while the Peruvian fields are commonly grouped as the northern and southern districts, the distinction geologically lies between the fields west of the mountains and those to the east, both groups being subdivided by cross ranges and other belts of unfavorable rock formations.

Pacific (Coastal) District

Age and Character of Oil-bearing Rocks.—The oil in northern Peru is reported as found in sandstones of probable Eocene age interstratified with clay, shale, etc. Deustua describes[6] the sequence of formations in the Pacific district of Peru as follows:

"Commencing in the mountain region of La Brea, we have: (1) A series of crystalline rocks, dioritic mostly, which may be regarded as the basal formation of the chain; (2) a mass of sediments composed almost entirely of shales metamorphosed into phyllades and phylladic slates, and enormously folded and faulted . . . these constitute the western flanks of La Brea; (3) a series of thick beds of sandstones, highly indurated and unconformable to (2), and which also form a part of the flanks of La Brea; (4) a series of alternating beds of sandstones and clays or shales, unconformable to (3), and which extend from the base of La Brea Mountain to the sea-shore . . . the lower beds of this series enclose the oil-bearing horizons; and, finally (5) a series of horizontal deposits, unconformable to (4), forming the plateau of the "tablazo," and composed of clays of different colors, alternating with a few beds of loose and little-hardened sandstones . . . both rocks are capped by thick beds of conglomerates.

"The metamorphism of the phyllades (3) is due to the intrusive rock (1), and these rocks form the principal base of the northern oil-bearing region, and the eastern limit of the oldest sediments. The rocks (3) are more modern than (2), but older than (4), which form a series of anticlinal and synclinal flexures, terminating in a wide anticline along the coast line. The beds of the upper sandstones are grey in color and coarse in grain, and are intercalated with red and yellow clays rich in fossils, while those of the lower oil-bearing sandstones are dark in color and coarse in grain, and are intercalated with thick clayey beds which are greenish in color, non-fossiliferous, and more compact than the upper argillaceous beds.

"The plateau deposits show three well-marked fossiliferous beds containing various species of gasteropods and lamellibranchs. The conglomeratic cap is formed of pebbles, breccia remains and coral reefs, strongly cemented. Fossils are found on the surface, the species of which are identical with those now living in the sea, near the shore. It is obvious that the sedimentation must have taken place in shallow waters. Both the elevation of the coastal plain and the thickness of the upper horizontal beds increase from south to north. At Paita the thickness is 75 ft., and the elevation 175 ft.; at Colan the thickness is 114 ft., and the elevation 229 ft.; in the

region of Cabo Blanco the thickness is 200 ft., and the elevation 450 ft. On the other hand, to the south of the Illesca Mountains the coastal plain descends gradually almost to sea-level (hence called *'desierto'* or *'desplobado'* to distinguish it from the high formation)."

Geologic Structure.—Describing the structure of these fields, Deustua continues:

"The oil fields of Negritos and Lobitos, to the zone of Cabo Blanco, lie on the eastern flanks of a long and wide anticlinal formation, the strike of which is, more or less, northwest to southeast, and the axis almost parallel to the line of the sea-shore. Probably this anticlinal formation is prolonged toward the interior, forming a series of anticlinal and synclinal folds right up to the western flanks of La Brea or Amotape. The principal axis trends further inland as one advances northward toward Los Organos and Mancora. The formation presents irregularities, due, principally, to secondary folds which affect the symmetry of its flanks, and to local faults which also alter its typical form. This anticlinal formation has favored the accumulation of petroleum, for the oil impregnates porous sandstones, and is surrounded and covered by impervious beds of clay or shale. But, as already stated, the oil-bearing sandstones are only of limited and local area."

"Borings have proved that the oil-bearing deposits are generally lenticular in form, their extension varying from a few square meters up to 50 or 100 cu. m. They may be grouped into three great horizons: (1) From 45 to 1000 ft. in depth, poor in filtrations; (2) from 1000 to 1800 ft.; and (3) from 1800 to 3000 ft. There is probably a fourth horizon, recognized in La Brea and Breita, but quite inaccessible to the drills used on the coastal band."

The rocks are much broken up, but the formation is generally monoclinal, with beds dipping eastward. Some anticlines have been located, but the production is more generally from monoclinal structure where local saturation is increased by both dip and strike faults.

According to Deustua,[6] the Zorritos fields are generally associated with a wide and elongated anticline, which extends along the coast. The oil-bearing rocks are largely sandstones which are sufficiently porous to render "shooting" unnecessary.

The same authority tells us[5] that the sea-coast in the Negritos fields corresponds closely with

"the eastern flanks of a wide and long anticlinal structure. The dip of the sandstone and clay at the south end of the region is southeast, advancing northward the dip becomes east, and, north of Negritos, northeast. The dome is not perfect, as the eastern flanks present a number of secondary folds and local faults, which alter somewhat the theoretical distribution of the oil horizons. But the bore holes made along the flanks have yielded the best results. Along the anticlinal axis or ridge itself they have been poor, while far to the east of the flanks there is the danger of encountering the corresponding synclinal, where the probability of meeting with oil is remote.

"From a depth of from 45 to 1000 ft. as many as seven oil-bearing formations have been met with, but, as a rule, these are poor, the most important being those below 1500 ft. in depth."

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[6] References are to bibliography at end of paper.

Central (Andean) District

Deustua tells us[6] that the formation in the Titicaca and other fields of the Andean district in southern Peru is composed in general

"of the alternating beds of limestone, clay, and sandstone, the whole zone resting on a long, well-marked anticlinal flexure, striking northwest to southeast. Exudations of oil occur in a series of greenish, porous sandstones, many of which are saturated with a pitch-like substance or residue. Elsewhere the oil issues in the form of small boiling springs, accompanied by brackish water, having a strong odor of petroleum and sulphurous acid, and depositing around the spring a series of precipitates of yellowish-white sulphur. The whole of the zone would appear to be well fissured and fractured, and eaten into by the circulation of thermal waters, represented by the springs already cited. The fissuring is especially observable in the limestone, which is highly folded, and, in places, almost vertical. The above has no doubt favored the accumulation of petroleum, which, according to some authorities, impregnates the limestone, and, according to others, the sandstone."

In the Pirin region in the district of Pusi, where the principal deposits exist

"the formation consists of a series of thick beds of limestone, alternating with argillaceous sands or clays and of sandstone, alternating with shale and conglomerate, all of which are strongly folded and dislocated. Upon these rest, unconformably, calcareous beds of lacustrine origin. There exists a wide and long anticline, trending approximately N. 20° W. to S. 20° E. On the flanks of the anticline are secondary folds, dislocations, and local faults, which change the general strike to a certain extent. The anticline appears to be a continuation of that of the Province of Canas, and both may be regarded as forming part of one large oil basin."

"According to Steinmann, the petroleum comes from the Lower Cretaceous sandstone, but, according to Duenas, from the Mesozoic limestone."

The walls in the Titicaca field struck the sand at 250 to 800 ft. (76 to 243 m.) from the surface.

4. Quality

Peruvian petroleum does not show the characteristic amber color of that of Pennsylvania, but nevertheless ranks high. It is rich in gasoline and kerosene, yields excellent lubricants, and has been shipped extensively to San Francisco, where it is sold to the Standard Oil Co. of California. The petroleum of northern Peru is generally light greenish in color, with a yellowish tinge. Traces of sulphur are sometimes found, but are generally absent. The oil ranges from 32° to 48° Bé. and averages 15 per cent. of gasoline, 35 per cent. of kerosene and 50 per cent. residuum. It generally has an asphaltic base, though the asphalt content is small.

According to Arnold[1] the petroleum in the Zorritos field is of asphalt base, and ranges in gravity from 37° to 43° Bé. The petroleum in the Lobitos field is similar. That in the Negritos field is brown and ranges from 35° to 42° Bé. In addition to the ordinary Peruvian coast petroleum, a black bituminous material is found at La Brea, 25 miles inland, and when evaporated, is used for coating the inside of "pisco" or "aguardiente" tubs, for weatherproofing, etc., and is also suitable for lubrication.

A specimen of petroleum from a well at Tucillal, 2½ miles from Zorritos, when examined by Redwood[15] was found to have a dark brown color, no unpleasant odor, and a specific gravity of 0.940. A specimen of green surface oil from an excavation in black slate at the base of an oil-bearing sandstone was examined by the same authority and found to be as follows:

Odor, slight and not disagreeable.
Specific gravity at 60° F., 0.920; Baumé, 22.5°.
Flash point (Abel test), 122° F.
Did not solidify at 0° F.
Viscosity at 70° F. (Redwood's viscometer), 69.41.

The same authority also examined petroleum from a well at Heath Ravine, between Zorritos and La Cruz, which was found to be as follows:

Color, dark brown.
Fluorescence, little.
Odor, agreeable.
Did not solidify at 0° F.
Specific gravity at 60° F., 0.859; Baumé, 33°.
Flash point (Abel test), 38° F.

The oil from the Negritos field is brown in color, has marked fluorescence, an aromatic odor, and is free from water. Its specific gravity ranges from 35 to 42° Bé. Redwood writes that

"Owing to the character of the 'cover' the petroleum of Negritos has been better preserved than that of Zorritos, and contains a larger proportion of the more volatile hydrocarbons; the oil when taken from the wells frequently containing as much as 18 per cent. of benzine. On the other hand, the Negritos oil yields far less kerosene than the Zorritos oil. The pitch obtained by the evaporation or distillation of the Negritos petroleum is said to be quite tasteless and odorless. Peruvian oil appears usually to contain little or no paraffine."

The petroleum from the Titicaca field is unlike that of the northern fields, having a paraffine base and about 5 per cent. of paraffine wax.

Analyses of Zorritos petroleum have been made by G. E. Colby of the University of California (sample No. 1) and by the American Analysis and Chemical Co. (sample No. 2); the results of which illustrate the variation in character throughout the fields.

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[1] References are to bibliography at end of paper.

TABLE 2.—*Distillation Results of Certain Zorritos Petroleums*

Fraction	Product	Temperature (Degrees Centigrade)	Samples No. 1 (Per Cent.)	Samples No. 2 (Per Cent.)
1	Symogene, rhigolene, etc.	10° to 30°	2.80	0.37
2	Gasoline	30° to 80°	9.00	10.00
3	Benzine	80° to 150°	11.10	15.00
4	Kerosene (light)	150° to 230°	18.50	24.90
5	Kerosene (heavy)	230° to 280°	10.00	12.40
6	Lubricant (light)	....	12.80	11.90
7	Lubricant (heavy)	....	4.80	5.90
8	Asphaltum	....	31.00	19.53
	Total	....	100.00	100.00

Fractions Nos. 6 and 7 are noticeable on account of the absence of paraffine, so that they can be submitted to a temperature as low as −30° C. without solidifying. The Zorritos oil is said to contain very little tar, but the residuum from fraction No. 8, if distilled further, will give an excellent coke.

Redwood is quoted, in *Mineral Resources of the United States* (1908, Pl. II, pp. 427–9), as authority for an examination of Negritos petroleum which resulted as follows: Color, reddish-brown, fluorescent; specific gravity at 60° F., 0.841 (Baumé, 36.5); sulphur, 0.0001.

TABLE 3.—*Fractional Distillation of Negritos Petroleum, Peru*

Fractions	Temperature of Distillation (Degrees Fahrenheit)	Specific Gravity of Distillate	Baumé
1	230 to 270	0.714	67.0
2	270 to 305	0.746	58.0
3	305 to 350	0.770	52.0
4	350 to 405	0.794	46.5
5	405 to 490	0.825	40.0
6	490 to 590	0.861	33.0
7	590 to 705	0.885	28.5
8	705 to 750	0.890	27.5
9	750 to 800	0.898	26.0

The percentages of commercial products were: Benzine 22.3; kerosene (flash point, 73; specific gravity, 0.813 or Baumé, 42.5), 23.8; lubricating and intermediate oils, 47.1; coke, 2.8; loss, 4 per cent.

PRODUCTION

The following is the production of crude petroleum in Peru from 1896 to 1915 of 42 U. S. A. gal. each:

TABLE 4.—*Production of Crude Petroleum in Peruvian Fields (in Barrels) (a)*

Year	Zorritos	Lobitos	Negritos	Titicaca	Lagunitos	Total
1896	47,536	......	....	....	....	47,536
1897	68,452	....	2,379	....	....	70,831
1898	68,571	....	97,292	....	....	165,863
1899	89,166	....	145,938	....	....	235,104
1900	102,976	....	217,036	....	....	320,012
1901	74,647	....	239,488	....	....	314,135
1902	59,273	....	205,810	....	....	265,023
1903	49,047	....	269,424	....	....	318,471
1904	49,547	....	295,617	....	....	345,834
1905	37,720	75,000 (*b*)	335,160	....	....	447,880
1906	42,419	162,000	330,510	1,365	....	536,294
1907	65,476	279,000	396,750	15,000	....	756,226
1908	71,429	319,898	543,750	......	....	1,011,180
1909	70,750	429,195	740,070	....	....	1,316,118
1910	107,000	400,080	773,025	....	....	1,330,105
1911	64,286	391,290	882,698	....	....	1,368,274
1912	78,095	587,048	1,071,000	....	....	1,751,143
1913	83,343	557,355	1,036,490	....	346,073	2,133,261
1914	88,136	504,743	1,032,210	....	282,713	1,917,802
1915	72,736	664,972	1,355,925	....	392,618	2,487,251

(*a*) In part from *Treatise on Petroleum*, by Sir Boverton Redwood (1913), **3**, 109, in part from Thompson's *Petroleum Mining*, 32, and in part compiled by Miss Anna B. Koons of the United States Geological Survey.

(*b*) Estimated.

From Table 5, taken from Deustua,[6] we learn the quantities of crude oil and its refined products produced in Peru from 1903–1915 inclusive:

TABLE 5.—*Crude Oil and Refined Products Produced in Peru*

Year	Crude Oil	Crude Benzine	Refined Gasoline and Benzine	Kerosene	Lubricating Oils	Light Residuum	Fuel Residuum
	(Tons)	(Cubic Meters)	(Cubic Meters)	(Cubic Meters)	(Cubic Meters)	(Tons)	(Tons)
1903	11,639	....	2,839	2,536	...	......	7,819
1904	5,980	....	160	2,744	...	.....	4,000
1905	7,280	....	3,236	3,246	...	......	6,970
1906	10,996	....	3,632	2,266	...	......	6,495
1907	14,735	....	583	4,322	174	......	7,300
1908	18,673	....	4,671	3,089	30	......	11,729
1909	24,644	....	5,990	3,739	89	......	15,882
1910	50,821	10,544	312	5,044	248	21,029	14,004
1911	105,589	18,975	523	3,910	239	73,783	7,245
1912	137,910	30,866	1,558	4,238	221	98,790	5,883
1913	179,888	44,692	2,538	5,360	732	117,834	4,504
1914	142,404	53,502	1,352	7,007	461	81,305	4,528
1915	216,879	56,364	6,820	33,199	532	100,067	31,575

The decline of production in these fields in the year 1914 over that of the previous year is supposedly due to the curtailment of activity on account of the European war, owing to difficulty in securing tank ships.

TABLE 6.—*Production of Petroleum and Its Products in Zorritos Field* (*a*) (*in Gallons*)

Year	Crude Petroleum	Benzine	Gasoline	Kerosene	Lubricating Oil
1896	1,996,520	4,560	......	608,900	896,450
1897	2,874,980	7,940	......	959,645	964,680
1898	2,880,000	8,350	......	600,000	1,250,000
1899	3,745,000	11,220	......	806,900	2,541,000
1900	4,325,000	13,000	......	400,000 (*b*)	....
1901	3,135,000	19,060	......	282,430 (*b*)	....
1902	2,489,500	......25,920......		373,250 (*b*)	....
1903	2,060,000	......61,745......		276,100 (*b*)	....
1904	2,080,000	......46,200......		365,000 (*b*)	....
1905	1,584,242	......29,570......		300,000	....
1906	1,781,600	10,000	54,000	350,000	....
1907	2,750,000	20,000	101,000	420,000	....
1908	3,000,000	30,000	150,000	500,000	....
1909	2,971,510	......	96,520	469,610	....
1910	4,494,000	......	84,420	549,000	....
1911	2,700,000	200,000	141,240	650,000	....
1912	3,280,000	......	226,440	476,620	....
1913	3,500,424	......	324,000	565,320	....
1914	3,701,718	......	277,440	482,850	....
1915	3,054,900	......	362,230	461,510	....

(*a*) Extracted from Redwood's *Treatise on Petroleum* (1913), **3**, 109.
(*b*) Estimated.

TABLE 7.—*Production, Shipments and Stocks in Lobitos Field*

Year	Production		Shipments	Stock (Dec. 31)	Producing Wells (Jan. 1)
	Metric Tons	Barrels	Metric Tons	Metric Tons	
1905	*10,000	75,000	......	......	...
1906	*21,600	162,000	17,756	......	...
1907	*37,200	279,000	25,821	4,816	...
1908	42,653	319,898	36,131	8,860	26
1909	57,226	429,195	54,289	11,797	62
1910	53,344	400,080	......	......	...
1911	52,172	391,290	......	......	92
1912	78,273	587,048	......	......	105
1913	74,314	557,355	......	......	110
1914	67,299	504,743	......	......	...
1915	88,663	664,972	......	......	...

* Estimated.

The figures in Table 6 give the production of the Zorritos field, so far as known, by years and products.

In Table 7 is a record of the production, shipments and stocks of petroleum and number of producing wells, in the Lobitos field, from 1905 to 1914.

As to the destination of the exported petroleum: 80 per cent. is said to go to the United States; while the remainder is sent to Chile, for nitrate industry. The petroleum produced in the Titicaca field has been mostly used as fuel by the Southern Railway, the lake steamers and Bolivian industries. In 1912 the price was reported as $25 per ton delivered in Juliaca or Puno. The price of coal in Bolivia was given in that year as $50 per ton.

ARGENTINA

1. Distribution of Argentina Fields

The Argentina petroleum fields, like those of Peru, may be classified into two general groups: (A) those in the Andean region, which belong in the western Argentina district; and (B) those on the coast, belonging in the South Atlantic district. According to Arnold[1] petroleum is supposed to underlie 8000 sq. miles (20,720 sq. km.) of which 400 sq. miles (1036 sq. km.) gives superficial evidences, but the proved area does not exceed 2 sq. miles (5.18 sq. km.). According to E. M. Hermitte[9], these districts can be subdivided into fields as follows:

> "(1) Cacheuta, which lies at the southern end of the Mendoza precordillera; (2) in the Province of Mendoza and the Territory of Neuquen; (3) in the subandine zone that runs from Bolivia into the Provinces of Salta and Jujuy; and (4) Comodoro Rivadavia, on the Patagonian coast. Of these (1), (2) and (3) are of great geological interest, owing to several peculiarities."

Oil has been reported in the Province of Santa Fe (region No. 5). The known districts are, however, as follows:

A. Northwest Argentina district.
   1. Cacheuta field.
   2. Mendoza-Neuquen field.
   3. Salta-Jujuy field.

B. South Atlantic district.
   4. Comodoro Rivadavia field.

In 1914, some geologists from the United States of America, then working in the employ of the Argentina Government, were reported to have discovered an extensive new oil field in the Gran Chaco, between Paraguay and the Province of Jujuy. This rumor has not yet been confirmed; but, if true, would add another field to the Northwest Argentina district. Still another field is reported to have been recently discovered on the coast far south of the Comodoro Rivadavia field.

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[1] References are to bibliography at end of paper.

A. Western Argentina District

Since petroleum is known to occur throughout a belt of 150 miles (241 km.) in Bolivia, between 63° to 64° W. longitude and 19° to 22° S. latitude, there seems little doubt that the oil zone continues far into Argentina to Tartagal and Aguaray.

1. *Cacheuta Field.*—Information regarding the Cacheuta field is not at hand.

2. *Mendoza-Neuquen Field.*—This district is 600 to 800 miles (965 to 1287 km.) southwest of Buenos Aires, and according to Arnold[1] includes at least 6000 sq. miles (15,540 sq. km.). It runs for nearly 300 miles along the eastern Andean slope, beginning 40 miles north of Mendoza and extending southward to latitude 42° in the Neuquen region. The area showing oil possibilities is estimated by Arnold not to exceed 300 sq. miles, and the proved area less than 1 sq. mile. Serious drawbacks to drilling in this field are the scarcity of water and fuel, coal being reported worth $30 gold per ton.

3. *Salta-Jujuy Field.*—The Salta-Jujuy district, according to Arnold, occupies a roughly triangular area with sides approximating 100 miles in length in the Andean region of northwestern Argentina, 800 to 900 miles northwest of Buenos Aires, adjoining the Bolivian frontier. Its fields are practically continuous with those of Bolivia. The district includes approximately 5000 sq. miles, of which about 250 sq. miles contain evidences of petroleum; but less than a square mile has actually been tested and the results have been indifferent.

B. South Atlantic District

This district includes the developed Comodoro Rivadavia field; and, if true, the new discovery on the coast 1200 miles (1931 km.) south of Buenos Aires.

4. *Comodoro Rivadavia Field.*—The Government wells are situated near the town of Comodoro Rivadavia, in the territory of Chubut, on the Gulf of St. George, about 850 miles (1367 km.) southwest of Buenos Aires. Petroleum is believed to extend "from the Atlantic Ocean to the banks of Chico River, in the direction of Lakes Colhue, Huape and Mustero, which border on Sarmiento." The wells and results are described later. The greatest difficulty in drilling appears to consist in casing off the water.

2. Surface Indications

Surface indications of petroleum in Argentina consist of:

1. Oil and asphalt springs and seepages.
2. Asphalt veins.
3. Bituminous shales and limestones.
4. Oil associated with ore deposits.

1. *Oil and Asphalt Springs and Seepages*

A full catalog of the localities where petroleum and asphalt are known to seep out of the earth would be a matter necessitating years of study. It will suffice here to list the principal localities where seepages are known, and to classify them by districts and fields.

Andean District.—In Salta Province, oil springs have been known for many years. The following are the localities which are reported by Redwood[15] in this and in Jujuy Province, to emerge from rocks of Lower Cretaceous age:

1. Yavi Chico, on the Bolivian frontier at Longitude 65° 30′ W.
2. Tejada, 60 miles south of this.
3. Abra de la Cruz, 29 miles east of Tejada.
4. Garrapatal, 21 miles east-northeast of Jujuy.
5. Laguna de la Brea, 42 miles further in the same direction.
6. Cerro de Calilegua, 30 miles northwest of the last.
7. Tartagal, on the frontier at Longitude 63° 44′ W.

The Salta and Jujuy seepages are reported by J. P. Cappeau[2] to exude "from a grayish-white sandstone." Near some of these seepages hot springs and sulphur springs are found. A proof that subterranean vapors exist in great expansive force in the Provinces of Jujuy and Salta is derived by some persons from the occurrence of earthquakes in that vicinity.

In Mendoza and Neuquen Provinces, Arnold[1] reports the following localities:

8. San Rafael, Mendoza Province (seepages exist and a kind of asphalt has been found which has been named rafaelite).

9. Cerro Auca Mahuida.

10. On the Rio Barrancas (petroleum springs) (boundary of Mendoza and Neuquen).

11. Curileuva.

12. Garrapatal (note repetition in Jujuy).

13. La Brea.

14. Vachenta.

15. La Carene.

16. "Plaza Huincul of Challaco" north of Kilometer 81 on the Railroad.

17. Undescribed oil indications are reported as having been "discovered in recent years, at La Cortaderita (formerly Government land) in the south of the Province of Mendoza."

18. South Atlantic district. At a point 60 to 70 miles west and north of Comodoro Rivadavia, on Rio Chico, and near Lake Conles Whapi are several large asphaltic seepages.

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[15] References are to bibliography at end of paper.

2. *Asphalt Veins*

Veins of asphalt traverse the Cretaceous beds of the Sierra de Loncoche, 20 miles south of the bituminous limestones mentioned in the Salado Valley of southwestern Mendoza Province.

3. *Bituminous Shales and Limestones*

Bituminous shales of Upper Triassic age have long been known in Salta Province; and petroleum has been produced by natural distillation in many localities throughout a belt of 300 miles along the east slope of the Andes, from 40 miles north of Mendoza to somewhere in Neuquen Province.

The Lower Triassic limestones of the Upper Salado River (about 35° South latitude and 70° West longitude) are charged with petroleum.

4. *Oil Associated with Ore Deposits*

In the Cacheuta mines, a few miles south of Mendoza, and also at the Chalahuen copper mines, Neuquen, oil occurs in the veins of selenides of silver, copper, etc. Such occurrences are not believed to be an important indication, but when considered with the numerous other indications they furnish some evidence.

3. Geological Conditions

The oil of the Argentina fields occurs in rocks of Jurassic, Cretaceous, and Eocene age, of which limestones, dolomites and sandstones predominate. Commercial quantities of oil are found both in anticlines and in regions that are declared to have practically no geological structure.

According to Bailey Willis,[18] the formations of northern Patagonia are metamorphic slates, cut by eruptive granites and rhyolites, the whole overlain by more recent sediments capped by flows of basalt and andesite. "Near the coast are younger . . . clay and gravel deposits, with layers of secondary limestone." West of the center of northern Patagonia is a conspicuous group of volcanic peaks, of which Anecon Grande is the highest.

Comodoro Rivadavia Field

According to Hermitte,[9] the strata of the Comodoro Rivadavia fields are flat, except for minor warpings.

The first water-bearing beds occur at depths of 1000 to 1150 ft. (304 to 350 m.) below sea-level, and the oil begins 150 to 250 ft. deeper, the intervening strata being impervious. In one well, oil was found as much as 2000 feet below sea-level.

The formations consist of clay, sand, gravel and sandstones. The oil in the Comodoro Rivadavia fields comes from a coarse pebbly sandstone of Upper Cretaceous age, which lies on schist and granite and is unconformably overlain by Eocene and later Tertiary tuffaceous and fossiliferous beds. J. P. Cappeau has suggested that large seepages 60 to 70 miles north of Comodoro Rivadavia may represent the outcrops of sands still below the discovered sands in the Comodoro Rivadavia fields, since the sands rise northward. According to the Government geologists, the beds dip at a low angle, not exceeding 12 ft. to the mile, and are said to form a broad shallow syncline.

Andean District

According to Redwood,[15] the oil and asphalt seepages of Salta and Jujuy Provinces exude from Lower Cretaceous rocks, chiefly dolomites and conglomerates, and from sandstones and dolomites of Jurassic age. The geological structure of the district consists of long, fairly well-defined anticlines. The anticline dominating the Sierra Aguaragüe can, according to J. P. Cappeau, be traced throughout the country, and extends into Bolivia. Two wells which, according to Arnold,[1] were poorly located with regard to structure, reached depths of 150 and 450 ft. (45 to 137 m.) without results.

The fields of the Province of Mendoza and the Territory of Neuquen and (*b*) the Provinces of Salta and Jujuy, the strata are of great geological interest. The rocks near Mendoza are shales and sandstone. It is necessary to extend the casing close to the bottom of the well. The beds yielding oil in northern Mendoza are said by Doctor Windhausen to be of Upper Jurassic age. Those of the south end are reported by Willis as Eocene, the oil coming mainly from sandstones and marls. East of the main Andean belt the lower Jurassic limestones and Cretaceous beds are said to be petroliferous, especially along the upper part of Salado River in latitude 35°, longitude 70°.

4. Quality

The petroleum from the Comodoro Rivadavia field has an asphalt base, and ranges in gravity from 18.9 to 21.8° Bé. (0.940 to 0.922 specific gravity).[13] Table 8 shows the results of analyses of petroleum from certain representative wells.

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[15] References are to bibliography at end of paper.

Analyses of Various Argentina Petroleums

TABLE 8

NOTE.—The following is taken from *Bulletin* furnished by the Bureau of Foreign and Domestic Commerce, entitled "Direccion General de Explotacion Del Petroleo de Comodoro Rivadavia," dated 1914; same being *Enclosure* No. 1 in a report dated July 21, 1914 on "Oil Wells" by the American Consulate General, Buenos Aires, Argentina.

Well No.	2	4	8	11	13
Specific gravity at 15° C.	0.9218	0.9232	0.9170	0.9320	0.9168
Viscosity at 50° C.	21.54	24.60	17.31	37.00	15.46
Flash point	31° C.	30° C.	28° C.	55° C.	32° C.
Burning point	76° C.	70° C.	70° C.	125° C.	70° C.
Water (per cent.)	.....	1.0	.....	.....	0.3
Water and sand (per cent.)	0.2	1.0	.....	.....	0.3
Ash (per cent.)	.....	.....	0.015	0.129	.....
Sulphur (per cent.)	0.22	0.24	0.18	0.14	0.19
Calorific value	10,070	10,000	10,290	10,354	10,000
Destructive Distillation					
Up to 150° C. (per cent.)	4.5	3.3	4.0	2.5	4.0
150 to 300° C. (per cent.)	17.5	15.2	20.0	15.0	20.0
Residue and loss (per cent.)	78.0	80.5	76.0	82.5	76.0

As in Peru, so in Argentina, the petroleum from the interior fields is of paraffine base, as contrasted to the asphalt base of the coast petroleums. According to Cappeau:

"The oil from northwest Argentina has a bright yellow color when emerging from the hole, turning black when exposed to the air, has a paraffine base, and is about 27° Bé. gravity."

A sample from the Mendoza field was found by Redwood[15] to have a specific gravity of 0.935 (Bé. 20), a flash point (by Abel test) of 178° F. and a setting point of 32° F. The petroleum of the Province of Jujuy is reported to be black, without disagreeable odor. At Laguna de la Brea, the oil is asphaltic and heavy. An analysis by the Government chemist is stated to have given the following percentages of distillate:

Product	Per Cent.
Light oil (specific gravity 0.74)	5
Kerosene (specific gravity 0.814)	30
Lubricating oil	52
Fixed carbon	11
Gases	2
Total	100

The *Bulletin* of the International Bureau of American Republics for January, 1910, states that the wells in Mendoza, Neuquen, Jujuy, Salta and Chubut have yielded a product which compares favorably with that of Pennsylvania and Ohio. The gravity of oil in the Mendoza-Neuquen field is reported, however, to be only 20° Bé. A refinery has been erected at Campana for treating the oil from the Provinces of Jujuy and Salta. According to *Continental*, April, 1916, the Federal Commission in charge has promised to buy all the oil at $15 per 1000 kilos. The cost of production for private concerns will be only $8.

The predicted success of petroleum development in Argentina is based upon the need of the entire production for domestic uses; particularly at this time, when the unusual increase in price of coal has put all industries under a severe strain. Moreover, the advantage exists that shipping facilities may be secured near the Comodoro Rivadavia oil fields in case the oil must be exported from the country.

5. Production

Table 9 shows the production of all the Comodoro Rivadavia wells from 1909 to March 26, 1915 (according to U. S. Vice-Consul, Eli Taylor, of Buenos Aires, in *La Nacion*, Apr. 8, 1915).

Table 9.—*Production of Comodoro Rivadavia Wells (in U. S. Barrels)*

No.	1907	1908	1909	1910	1911	1912	1913	1914	Jan. 1 to Mar. 26, 1915
2	101	11,472	11,784	714	422	6,801	2,380	20,786	8,353
3	...	.....	623	.....	.....	.....	......	......	.....
4	...	.....	6,424	19,340	1,916	.....	105,091	111,565	20,949
7	...	.....	.....	699	10,781	28,004	8,045	15,512	3,949
8	...	.....	.....	.....	.....	12,202	7,877	55,900	26,323
9	...	.....	.....	.....	.....	.....	5,286	.....	.....
11	...	.....	.....	.....	.....	.....	1,844	17,738	3,667
12	...	.....	.....	.....	.....	.....	.....	28,542	417
13	...	.....	.....	.....	.....	.....	94	17,524	4,151
14	...	.....	.....	.....	.....	.....	.....	7,933	13,757
15	...	.....	.....	.....	.....	.....	.....	.....	4,078
	101	11,472	18,831	20,755	13,119	47,007	130,617	275,500	85,346

The total production for 1915 has since been reported as 516,120 bbl. These figures represent the total production of Argentina, as the output of the other fields is negligible.

According to a report recently issued, the number of producing wells in the Comodoro Rivadavia fields early in 1916 was 21. In addition, 12 borings were in progress, four of which were to be concluded by the end

of March, three having been just started and five being well under way. Work was commenced by the Commission in January, 1911, between which date and Feb. 17, 1916, the quantity of petroleum extracted from the State reserves amounted to 156,300 tons, of which amount 122,000 tons had been sold. The output of 1916 was estimated as 160,000 tons. Early in 1915 the price of oil at the wells was 35 pesos per ton. The average monthly sales to private persons averaged 100 tons; and the railway used 7000 tons per month.

VENEZUELA

1. Distribution of Venezuela Fields

The petroleum fields of Venezuela, so far as can be learned, are classifiable as follows:

A. Caribbean district (in the vicinity of Lake Maracaibo).

1. *In the district of Mara,* near the River Liman asphalt lake, where oozings of petroleum cover considerable areas.
2. *Bella Vista,* near the city of Maracaibo.
3. *In the district of Sucre,* on the eastern shore of Lake Maracaibo, where signs of petroleum have been found associated with asphalt deposits.
4. *On the Sardinate River,* extending into Colombia, where petroleum is developed on a small scale and sold locally.
5. *In the district of Colon,* in the State of Zulia, south of Lake Maracaibo, this being the largest and most accessible field in Venezuela at present developed.
6. The *Perija Field,* 50 miles west of Lake Maracaibo.

B. Orinoco district. *Pederneles Field.* This field is situated in the Delta of the Orinoco river, at the place where one of its northernmost mouths empties into the Gulf of Paria. The field includes portions of the Islands of Capure, Pederneles, and Plata. It was first discovered on account of an enormous deposit of asphalt on the northwest coast of the Island of Capure, which is ½ mile in a northeast direction and 100 to 200 yd. (91 to 182 m.) across.

2. Surface Indications

The surface indications in Venezuela consist of:

1. Asphalt lakes.
2. Smaller seepages of oil and asphalt.
3. Mud volcanoes.

1. *Asphalt Lakes*

The indications most evident to the non-professional person are asphalt lakes—"pitch lakes" as they are commonly called—of which those in Venezuela are worthy examples. These constitute the residuum from immense oozings of petroleum that has risen from underlying sands through crevices or has emerged on the outcrop of an oil-bearing sand. Much controversy has taken place among geologists regarding the precise process, but there is little doubt of the general conditions of origin.

The most prominent indication in the country is the desiccated deposit known throughout the world as Bermudez asphalt. Large amounts of it have been shipped to North America from this and other lakes, on the northern coast.

The Bermudez lake is 105 miles (168 km.) due west of the Trinidad asphalt lake, across the Gulf of Paria. It is situated near Guanoco, 3 miles from the mouth of Guanoco river, which joins the San Juan about 32 miles from its mouth, flowing into the Gulf of Paria. The Guanoco and San Juan Rivers are navigable. The Guanoco, however, has a bar at its mouth and vessels must be adapted to such navigation. Transportation from the asphalt lake to the town of Guanoco is by a tramway, 5 miles in length. In a straight line, Bermudez lake is only a few miles from salt water. It is about 1000 acres in extent, being larger in area than that of Trinidad, but much shallower. It is formed by the overflow of asphalt from several springs.

Another good asphalt deposit in Venezuela is near Lake Maracaibo and others are in Merida and near Coro. Ruins of a German asphalt and oil refinery exist a mile or so from the settlement of Pederneles, where black patches of natural asphalt are exposed along the main street.

The chief indications of oil in the Pederneles field consist of a great deposit of asphaltum, ½ mile in length and 100 to 200 yds. across, situated on the northwest coast of the Island of Capure. Those seepages include active oil springs and one prominent asphalt cone. About ¾ mile south of the main deposit are two small cones, with additional isolated deposits 2 miles northeast, and also on the Island of Plata, in the river just north of it, and at Pederneles village.

2. *Smaller Oil and Asphalt Seepages*

In the western region, frequent indications are found in the vicinity of the Lake of Coquibacoa. Petroleum also occurs in all three of the Andine States of Táchira, Merida and Trujillo, and the Táchira Petroleum Co. has produced and sold small quantities of illuminating oil for many years. Well-known oil springs exist near the town of Betijoque. On the north branch of the Tocuyo River widespread indi-

cations of petroleum exist in a practically uninhabited and unexplored region.

In the Caribbean district, oil possibilities are supposed to exist on the Island of Cubagua, along the north shore of which springs of petroleum occur. Petroleum springs also exist on the south side of the peninsula of Araya and also at Punceres. According to the *Oil Encyclopedia*, traces of petroleum can be seen rising in the sea off the coast of La Guayra.

Other indications occur on the lower part of the Guanipa River. Petroleum exudes at various points on the Caribbean coast near Manicuare, opposite Cumana, on the Island of Cubagua, and in Margarita. The States of Falcon and Zulia are rich in asphalt and petroleum, and the Andine States of Trujillo, Merida and Táchira contain large quantities of these substances.

Along navigable rivers in the Maracaibo fields, evidence of petroleum can be seen on certain rugged hills, where at least a score of small streams descending from 100 to 200 ft. (30 to 60 m.) above river level are constantly covered with a thick coating of petroleum. The substance also issues from fissures along the banks, and in one of these an excavation 3 ft. deep and 2 ft. square was filled with petroleum in about 6 hr. The land in the vicinity was covered with a deposit of tar thick enough to destroy all kinds of vegetation.

3. *Mud Volcanoes*

Mud volcanoes are reported in the neighborhood of Maturin in the Bermudez district.

3. Geological Conditions

Geology of the Asphalt Lakes

Unfortunately the results of the detailed geological work of the General Asphalt Co. in Venezuela have not been made public. Hence we know no more regarding Venezuela than we do of other less thoroughly explored parts of South America.

The asphalt deposits of Venezuela are, according to Redwood, of Miocene and Cerro de Oro (Cretaceo-Tertiary) ages. These deposits in the Maracaibo region are distributed over the low country surrounding the lake and underlie the latter, as shown by occasional so-called "oil springs" in it. The formations appear to be much-broken and folded Carboniferous rocks, overlain with Tertiary sandstones and conglomerates, which are likewise much broken and highly tilted; the asphalt oozing to the surface through certain sandstone strata throughout a considerable distance along the line of strike, which is reported by Kempton[11] as being from north of east to south of west. The flow is intermittent, and changeable in position. The sandstone stratum itself is rich in

asphalt, but the coarser and conglomerate strata contain little of the substance.

The flows are sometimes over 20 ft. (6 m.) in thickness, composed of alternating layers of asphalt and fine sand; the latter being believed by Kempton to have been deposited by the wind. The surface of the asphalt region is level, most of the hills being less than 75 ft. above the lake, and the country largely covered with a low open forest rather free from underbrush. Blow holes of gas occasionally occur along the asphalt seepages. Some water accompanies them and is frequently rather warm, highly impregnated with sulphuric acid and other minerals.

Guanoco Field

The Guanoco oil field is believed to cover the axis and flanks of the Guanoco anticlines, the southernmost of which is thought to be responsible for the great seepage known as the Bermudez asphalt lake. The anticline trends eastward through the main seepages in the asphalt lake, and is believed to extend 60 miles southwest to a point in the San Juan River; being represented by a rather continuous line of oil seepages. The four wells drilled by the General Asphalt Co. up to 1914 were sunk with reference to the main anticline, but do not appear to have been situated on its crest.

Maremare Field

The geological conditions at Maremare are not known to the writer, but were reported favorable for oil, and small seepages exist. The work done subsequent to July, 1913, however, seems to have eliminated oil possibilities, and the explorations at Maremare have been abandoned.

Pauji Field

The conditions in the Pauji field are unknown, but small seepages are reported, and conditions are said to be such as to create strong expectation of finding oil. The development work done subsequent to July, 1913, appears to have been unsuccessful, and the operations abandoned in that field.

4. Quality

Little information is available as to the quality of petroleum found in Venezuela. In one instance, it is known that the oil was thin enough to flow readily, having a specific gravity at 15° C. of 0.8837 (Baumé, 29); while another deposit was very thick, being of the color and consistency of coal tar. Both petroleums have an asphaltic base, and the thinner of the two gave the following distillation results:

Product	Distillation Temperatures (Degrees C.)	Per Cent.
Gasolene	0 to 120°	0.5
Kerosene	120° to 170°	0.5
	170° to 235°	14.0
	235° to 270°	28.0
Lubricating oil	270° to 370°	51.0
Coke		6.0
Total		100.0

The flash point of the 170° to 235° product is 62° C., while that of the 235° to 270° product is 83° C. The viscosity of the lubricating oil is twice that of water.

The second sample of petroleum gave the following distillation results:

Product	Distillation Temperatures (Degrees C.)	Per Cent.
Water		28
Gasolene and kerosene	0° to 310°	0
Lubricating oil	310° to 370°	61
Coke		11
Total		100

Petroleum from seepages in the Pederneles field is reported to be of comparatively light specific gravity. In the first well drilled at Guanoco, the heavy oil had a specific gravity of 1.02.

5. Production

The petroleum production of Venezuela has as yet hardly commenced. The General Asphalt Co. and its subsidiaries are known to have been drilling wells for several years on the basis of recommendations of their geologists; but no public statement of the amount of oil secured has been made. It is known, however, that a large amount has been secured, and we may suppose that when adequate tank-steamer and other transportation facilities can be secured, a large amount of fuel oil will be put on the market.

In 1916, there were (according to a *U. S. Commerce Report* from Minister Preston McGoodwin, at Caracas, dated Aug. 26) in Venezuela six petroleum companies engaged in development work on an extensive scale.

1. Caribbean Petroleum Co.
2. Colon Development Co.
3. Venezuelan Oil Concessions, Ltd.
4. Venezuela-Falcon Oil Syndicate, Ltd.
5. Bermudez Co.
6. Pauji Concession.

The *Gaceta Oficial* (Caracas) of Sept. 28 and 29, 1916, publishes the text of laws granting two new concessions as follows:

1. A 25 years' concession to Señor Guillermo Pimental Proconis, of Caracas, for the exploitation of asphalt, petroleum, bitumen, pitch, etc., deposits near Puerto Cabello.

2. A 27 years' concession to Señor Artistides Sóto Bracho, of Maracaibo, for the working of four coal mines and also petroleum deposits in the district of Jaji, State of Merida. The concessionaires must commence the actual working of the deposits within a period of 3 years from the date of the approval of the contracts. They will be allowed to import, free of duty, machinery and other equipment required for the working of the deposits.

In addition to the properties of the companies above enumerated, there are not more than three or four ancient and modern concessions in existence, and no other development is now known to be in progress.

COLOMBIA

1. Distribution of Colombia Fields

The prospective oil fields of Colombia are grouped by Arnold[1] into Caribbean, Pacific, Magdalena-Santander and Tolima districts. Their total area is estimated by him as 34,000 sq. miles (880,60 sq. km.), and their probable petroleum territory only 618 sq. miles. Only 2 sq. miles are stated as now proven. According to the writer's classification the fields will be grouped as follows:

A. Caribbean district.
 1. Caribbean field.
  (*a*) Tubura pool.
  (*b*) Turbaco pool.
 2. Magdalena-Santander field.
 3. Tolima field.
B. Pacific district.

A. Caribbean District

1. *Caribbean Field.*—The Caribbean field is supposed to extend from Rio Hacha, on the west edge of the Guajiro Peninsula, southwestward along the Caribbean Coast to the Gulf of Darien and Gulf of Uraba, "and inland to include the Tubura and Turbaco pools and the region as far south as 30 miles (48 km.) south of Chima, on Sinu River, and Monpos, on San Jorge River, and up the Arato River valley for 90 miles from Punta Arena. It occupies portions of the departments of Magdalena, Bolivar and Cauca." It is approximately 300 miles (482 km.) and 50 miles wide, contains 15,000 sq. miles (38,850 sq. km.) and has 300 sq. miles of probable productive territory, of which only 1 sq. mile has been proven by drilling.

[1] References are to bibliography at end of paper.

The known pools in the Caribbean field are named (1) Tubura and (2) Turbaco. The Standard Oil Co. of New York did some drilling in the Sinu Valley near Lorica.

Tubura Pool.—This pool is 20 miles east of Cartagena and quite close to Barranquilla. A Canadian company has drilled three wells from 700 to 3018 ft. deep (213 to 919.88 m.) at least one of which yielded 7 or 8 bbl. of oil daily. Colombia seems to be the only South American country in which natural gas is known in large quantity, and its principal supply is in the Tubura pool. A plan has been suggested to pipe the gas to the city of Barranquilla for use in lighting, cooking, and generating power.

Turbaco Pool.—This field is 12 to 15 miles (3.6 to 4.6 m.) south of Cartagena. Diego Martinez & Co., in association with the Standard Oil Co. of New York, drilled five wells 500 to 2200 ft. deep. Diego Martinez & Co. also own a small refinery in Cartagena, where imported oil is refined.

2. *Magdalena-Santander Field.*—The Magdalena-Santander field, "probably as important but more inaccessible and less well known than that along the Caribbean coast, includes the southern part of the Department of Magdalena, the Department of Santander, and the western edge of the Department of Boyaca, extending from Magdálena River to the eastern Cordilleras. It also occupies an area in the southeastern part of the Department of Bolivar." It covers a belt about 200 miles (321 m.) long and 50 miles wide, having an area of 10,000 sq. miles (25,900 sq. km.) of which only 200 are believed to be productive, while only 1 sq. mile has been proven. A small refinery, which handles the product of a few wells for local trade is situated at Pamplona.

3. *Tolima Field.*—Under the Tolima field are grouped the petroliferous occurrences in the upper Magdalena basin, in the Departments of Cuninamarca and Tolima, and on the edge of the San Martin and Casanare plains. While this district has an area of 7000 sq. miles, only 100 are believed to be productive, and none have yet been definitely proved.

B. Pacific District

The Pacific district in the Department of Cauca, as defined by Arnold, includes a belt 60 or 70 miles (96 to 112 km.) long extending along the Pacific Coast north of Buenaventura to Baudo River, reaching inland to Atrato River at Quibdo and as far south as Cali on Cauca River. The area of the Pacific district is given as 1800 sq. miles of which the probable productive territory is considered as only 18, none being yet proven.

2. Surface Indications

The known oil indications in Colombia consist of the following:

1. Oil, gas and asphalt seepages.
2. Mud volcanoes.

1. *Oil Seepages*

Oil seepages are abundant in Colombia. Some of the small rivers which empty into the Caribbean Sea, especially the Rio Hacha, carry on their surface oil of good quality which can be collected by skimming. At Usiacarí, 2 miles (3.2 km.) southwest of Tubera in the Caribbean district, sulphur springs exist, and 8 miles west of Tubera oil seepages are found. Indications of oil are also found in the Caribbean field near Calamar on the Magdalena River, and near Baranoa back of Salgar on the Puerto Colombia Railway, in the region of Galera Zamba and Savana Larga. About 10 miles south of Punta Arenas, 4 miles east from the Gulf of Darien, "an unconfirmed seepage yielding 7 bbl.* daily of oil, having a paraffine base and a gravity of 41° Bé." is reported as used in lamps. An area of 800 sq. miles (2072 sq. km.) on the west side of the Gulf of Darien is said to yield coal and indications of oil of 16° Bé. Redwood states that "no less than 40 petroleum springs have been found 1 to 3 miles from the Gulf of Uraba (Darien) and near Arboletes, one of which has a crater about 12 in. in diameter, and yields sufficient oil to fill a 6-in. pipe.

Considering now the Magdalena-Santander fields, and going several hundred miles from the coast, we learn that in the year 1915, petroleum seepages heretofore unrecorded were found in all the Departments bordering on the Magdalena River as far south as Girardot.

According to Arnold:[1]

> "Along the Magdalena flank of the eastern Cordilleras is a line of seepages, some reported as yielding as much as a barrel of oil a day. These extend from a point south of Girardot northward to Bucaramanga in Santander Province; the beds along this line being sharply folded and in many places fractured."

Oil indications also exist near Pamplona on the Venezuelan boundary, in an area near La Gloria on the Magdalena, and seepages at Simití in Department of Bolivar. In the Simití region, on the west bank of the Magdalena, less than 200 miles in a straight line from the sea, are reported oil seepages. Oil indications have been found on the east bank of the Magdalena and in the valley of the Cesar River above Lake Zapatosa, and in the valley of the navigable Lebrija River. In the mountain region of Ocaña oil seepages are also reported. Indications have also been found on the Carare River, which is navigable for small boats. In the mountain region, near the town of Zapatoca and the Sogamozo or Chicamocha River, is a petroleum spring, examination of which was made by Dr. Hettner and the results published in 1892 in *Peterman's Mitheilung*.

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* This may be 7 gal.

[1] References are to bibliography at end of paper.

Between the Sogamozo River region and Bogota, are a series of table-lands, in all of which so-called oil indications have been found. In the mountain side, not far from Bucaramanga, a good oil spring is said to exist, samples of which have been seen by Consul Isaac A. Manning. On both sides of the Upper Magdalena River, south of Honda, are a series of rapids where extensive indications of petroleum exist. Some of the central plateaus, like that on which Bogotá stands and others north of it on the western side of the eastern mountain range, are surrounded by coal measures, and petroleum seepages are reported near Choconta, Ubate and Chiquinquira, and near Tunja.

In the Department of Tolima, 70 miles (112 km.) south of the city of Honda and 3 miles from the Magdalena River, 7 miles from the village of Purificacion, are six natural oil springs on the west side of a hill, where the oil is said to flow from fissures in the sandstone. The oil was flowing out in small streams within a radius of 300 yd. and is said to be rich in lubricants and kerosene. Traces of oil have also been found on the surface of water in two creeks in the same locality.

A short distance south of Ambalema in the same department west and southwest of Honda, asphalt has been found. Asphalt also has been found west of the Magdalena River, especially near the Saldana River, a tributary of the Magdalena. Near the village of Chaparral, according to Redwood,[15] a deposit of asphalt has been worked since 1903, and 2000 tons have been shipped per annum. On the east side of the Upper Magdalena River, in the neighborhood of Girardot, a seepage of petroleum exists at the village of Carmen, only 5 miles from the river in a straight line, but 14 miles from Girardot by road.

According to White,[17] the Plain of Bogotá

> "is supposed to be the site of an ancient lake and has rich salt deposits and salt springs, especially at Zipaquira and Nernocon, as well as indications of natural gas, such as 'corpse candles' found in hollows near the city after heavy rains. These facts are considered by many as proving beyond a doubt that petroleum exists there."

Redwood[15] states that:

> "Other springs are reported to occur on the Plain of Medina, at the foot of the extinct crater of Guaycaraime, within that part of the southeastern Cordilleras which terminates in the Plains of Medina, about 9 miles from the River Upia, a tributary of the Meta. The oil exudes from sandstone, and is free from water. Its rate of flow was estimated, by noting the time taken to fill a ditch of 2 cu. m. capacity, at 2 bbl. per minute."

And from Arnold we learn that this petroleum is "of about 22° Bé. gravity, and comes from shale fissures."

Evidences of coal and oil are said to have been found all along the foot of the Cordilleras, from which streams flow into the Orinoco and

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[15] References are to bibliography at end of paper.

Amazon Rivers. Some of the seepages are so abundant that oil can easily be dipped out of the depressions. According to White, several mule-loads of this petroleum were collected with very little work from one well alone and shipped over the mountains to Bogotá for lubricating purposes on the railroads and in the machine shops.

On the eastern side of the Andes, springs of oil are also reported to exist at more than one point on the edge of the San Martin and Casanare Llanos, which may be classified for present purposes either in the Orinoco or Andean district. "At a point 140 miles north of Nevea," according to Arnold,[1] "on the east side of the Andes, Mr. Vassar collected samples of paraffine-base oil testing 31° to 38° gravity." As to the Pacific district, we find that oil has been seen floating on the Andagueda River, a tributary of the Atrato. In the words of Arnold:[1]

"Heavy oil is reported east of Quibdo, where the Quibdo-Medellín trail crosses Tutendnendo River, and also 20 miles from Santa Rosa and 4 miles north of Porce River, which is south of Nechi River."

2. *Mud Volcanoes*

Numerous mud volcanoes exist in the vicinity of the Caribbean Sea, one particular chain of them extending for about 10 miles. They emit muddy water, not salty, at a temperature of 70° F., together with marsh gas. In the Tubera field as many as 100 volcanoes are said to occur in an area of three acres. In the Turbaco field, the chief surface evidence also consists of mud volcanoes. Near Turbaco, 18 miles from Cartagena, gas escapes from at least 25 openings on a space not over 2 acres in extent. These were referred to as the "air volcanoes of Turbaco" by Baron von Humboldt, who gave the first and most graphic description of Colombian oil indications in his *Cosmos* (**5**, 204), where he writes as follows:

"In the neighborhood of Turbaco, where one enjoys a magnificent view of the colossal snowy mountains (Sierra Nevadas) of Santa Marta, on a desert spot in the midst of the primeval forest, rise the volcancitas, to the number of eighteen or twenty. The largest of the cones, which consist of blackish-gray loam, are from 19 to 23 ft. in height, and probably 80 ft. in diameter at the base. At the apex of each cone is a circular orifice of 20 to 28 in. in diameter, surrounded by a small mud wall. The gas rushes up with great violence, as in Taman, forming bubbles, each of which, according to my measurements in graduated vessels, contains 10 to 12 cu. in."

If a pipe be inserted in the ground, the gas can easily be ignited at the top.

3. Geological Conditions

Bearing of Development on Geological Explorations

The oil fields of Colombia have long drawn some attention from outside capitalists, who have realized, in a vague way, that oil exists there. Unfortunately the explorations have resulted in little success, due perhaps to three main causes:

(*a*) The explorations of large oil companies have, until recently, been mainly in a comparatively few localities near the coast and adjacent to certain main rivers.

(*b*) It has not been realized in all cases that geologic conditions, while fundamentally similar, present many unusual phases in every country examined, and that these discrepancies must be taken into account before condemning a district as unfavorable.

(*c*) Much of the country is an almost impenetrable jungle, through which well-drilling machinery can be transported only at great expense.

An absolutely essential factor in any undeveloped country is to understand that the most favorable territory does not necessarily lie close to big seepages, but that on the other hand, it may be a hundred miles from any seepage, in places where the geological structure is favorable; while seepages, on the other hand, exist generally on the outcrops of sands or where the formations are broken by faults or intrusions of igneous rock. When the foregoing facts are perfectly understood, certain territories in all countries will be reëxplored.

A summary of geological conditions in Colombia with reference to mineral deposits was given by Consul Isaac A. Manning,[12] but oil and asphalt were merely referred to. While some large companies have informed themselves regarding the areal geology of certain parts of Colombia, no complete public geological survey has ever been made, and conditions are relatively unknown.

Magdalena-Santander Field.—In the Magdalena-Santander field, according to Arnold,[1] oil occurs in the Cretaceous limestones and sandstones and the coal-bearing Lower Tertiary (probably Oligocene) beds. In the Magdalena Valley and in the eastern Cordilleras, oil exists in Cretaceous rocks. "Long, well-defined, and in places overturned anticlines and possibly fault zones in the Cretaceous rocks are the advantageous structures for accumulation." The oil industry that has long existed near Pomplona on the Venezuelan frontier, comes from the Villeta beds of the Cretaceous system.

Geological Conditions in the Caribbean Fields.—Arnold[1] describes the rocks of the Caribbean fields as being of Lower Tertiary (probably Oligocene age) and hence similar to those of the Magdalena-Santander district. The rocks are coal-bearing and consist mostly of dark-colored shales containing sandstones. Oil occurs in sandstone and in joints in the shale. "The structure is that of broad to sharply folded and faulted anticlines, and the surface evidences are usually, though not invariably, confined to the anticlinal areas."

Geological Conditions in the Pacific District.—The oil on the Bauda River is associated with the "coal series" above mentioned, and probably occurs in a southwestward extension of the Caribbean coastal belt.

[1] References are to bibliography at end of paper.

Geological Conditions in the Tolima Field.—On account of the central plateaus in the vicinity of Bogota and to the north being surrounded by Coal Measures and of the existence of oil seepages near them, White[17] believes that the high tablelands of the eastern range of mountains may prove exceptionally rich in petroleum.

"Geological reports indicate that oil deposits exist in the region of Purification, Melger and Carmen, as well as in that of Tocosima, farther north, extending to the plateau on which Bogota stands. It is thought that the high plateau on which Bogota, the capital, is situated, may be found to contain especially rich oil deposits. It contains some 500 sq. miles of territory and is enclosed on all sides by Carboniferous mountains, except at the point where the river, which drains the plain, passes out to form the Falls of Tequendame."

4. Quality

Petroleum from the Colombian fields ranges from a heavy asphaltic to a paraffine base oil of 41° Bé. Arnold[1] gives the specific gravity of that in the Caribbean fields as 16° to 41° Bé. (most of it being between 20° and 30° Bé.), that on the Bauda River in the Pacific district as 31° to 37° Bé., and that in the Magdalena-Santander district as more variable.

According to Redwood:[15]

"A heavy oil exudes in considerable quantities along the Rivers San Juan and Vulcan. In the delta of the Sinu River also a heavy petroleum with much sulphur is found. To the east of the Sinu River, oil of good quality (specific gravity 0.858, viscosity 0.98, flash point 101° F.) with paraffine base is met with, the analysis of which gives naphtha 2.92 per cent., kerosene 31 per cent., lubricating oil 30 per cent., paraffine 3 per cent., and residuum 27.08 per cent."

To illustrate the varied character of the petroleums from this country, another sample examined by Redwood, is mentioned, being:

"dark brown in color when viewed in a thin film by transmitted light, exhibited but little fluorescence, and had a slight odor of not unpleasant character. Its specific gravity was 0.926, its flash point 310° F. (Abel test), and its viscosity at 70° about two and a half times that of rape oil. It solidified at 5° F."

5. Production

The production of Colombian petroleum has, as yet, not really commenced. In the Magdalena-Santander field, the wells are shallow and the production small, although of good promise. In the Caribbean field only ten wells are known, the production of which is given by Arnold as less than 10 bbl. per well per day. There is no reason, however, in the opinion of the present writer, to suppose that the Colombian fields will ultimately prove to be less valuable than those of Peru.

ECUADOR

1. Distribution of Ecuador Fields

The principal petroleum region of Ecuador is situated on the Gulf of Guayaquil, being an extension of the Peruvian fields. Some indications

in the mountains near Quito have been investigated by a Dutch syndicate, but the results are unknown. We must, therefore, confine our attention here to the Guayaquil region. All the petroleum lands discovered up to the present in the Province of El Oro are adjoining or near the Gulf, extending coastwise approximately 60 km. (37 miles) and toward the interior approximately 10 km.

Santa Elena Field

The best-known oil field of Ecuador is that of Santa Elena, 64 miles (102 km.) west by south from Guayaquil. The surface indications are said to be exactly similar to, but more prominent than, those on the adjacent Peruvian coast. The whole of the peninsula between the Pacific Ocean and the Bay of Santa Elena is considered as more or less petroliferous. In an unpublished report, M. J. Stephan states that "the large supply of oil at present obtained at Aguiquimi . . . and at Santa Elena Paula . . . clearly points out the existence of a rich oil zone in a deeper horizon at these particular points." Petroleum is found at many places near the Promontory of Santa Elena. According to unpublished reports of Charles Maddock, definite seepages of oil are found in many places over an area of 600 sq. miles (1554 sq. km.) Judging from the depths at which it is found in the adjacent field of Peru, authorities believe that oil in commercial quantities will be reached at about 1000 ft. (304 m.).

The topography of this field is undulating to hilly, with many valleys and ravines, portions of which grade into a plain bordering the Pacific Ocean and the Bay of Santa Elena. The climate of the Santa Elena fields is described as "very beautiful and healthy," the temperature at noon seldom going above 90° and fever being unknown. Labor is cheap and plentiful, being—according to Dr. Stephan—25 to 40 c. per day. In the province of El Oro timber for construction purposes can be had in abundance in the mountains. An established telegraph service exists between Quito, Guayaquil, Machala, Payta, Tumbez and Zorritos. Telephone service is established between Puerto Bolivar and Machala. Public highways have been constructed to nearly all towns and cities, and regular steamboat service exists within Panama, Guayaquil and the southern republics.

Achagion Field

This district is situated about 125 miles (201 km.) south of Guayaquil. Some hand wells have been excavated to depths of 6 to 20 ft. (1.8 to 6 m.) but these are no longer worked.

2. Surface Indications

In the Pacific district, according to many visitors, the principal surface indications consist of "gum" deposits and oil seepages at San Raimondo

on the coast and at Santa Paula and Achagion, 2 or 3 miles (3 to 4 km.) inland. In this field the formation containing the observed petroleum, known by its black color, is blue shale of unknown thickness, superficial at some points and elsewhere covered with marine débris, very spongy and resting on impermeable sandstone, impregnated with oil. It is said to be horizontal, of variable thickness, and underlying an area extending 6 miles north and south and about 20 miles inland. At San Raimondo, "the beach is found to be saturated with oil when the tide is out. In places oil oozes from the outcrop of sandstone."

Stephan states that bituminous layers are exposed in many places along the valleys; and that in other places, small holes dug in the ground showed seams saturated with oil and invariably a strong development of gas; and also that the oil occurs in sandstones or decomposed shales, either on the surface or in small cavities a few feet from the surface. One can easily detect the seepages by the discoloration of the overlying soil, as well as by the distinct smell noticeable a few hundred feet distant. Traces of petroleum also exist 30 miles east of Point Santa Elena and south of Puná Island.

E. J. Rye, in a private report dated Feb. 20, 1908, mentions certain shale deposits overlying hard sandstone on the coast of Ancon Bay, Santa Elena, stating that the shale deposits are so saturated with oil "that it is easily discernible by its heavy black color, while in a great many places it forms pools of oil." In places "shafts or wells have been sunk from 3 to 7 m. in depth and the pressure of gas from these wells has prevented any further development by hand power, although the supply of oil had gradually increased as a greater depth was attained." During the 2 years following the heavy earthquake shock about the year 1906, which was felt along the American coast from San Francisco to Valparaiso, the hand-dug wells at Santa Elena produced oil in larger quantity and of superior quality to that obtained in 1890. At the same time many springs previously unknown, started to flow.

Mr. Rye states that "on account of the obnoxious smells from gases, the wells cannot be tested over a depth of 30 ft." by the improvised methods which are used. He states, furthermore, that several natural petroleum springs exist along the coast of the Bay of Ancon, within 100 to 300 m. from the shore "which distribute the oil freely in large quantity all along the sea beach." Thoret states that in the Province of El Oro, seepages are frequently observed, being most evident at Barranco Blanco, Eli, Dos Bocas, Huaquilla, Santa Rosa and Puerto Pilo. Oil is also reported at points on the coastal plain north of Guayaquil, particularly at Atacamas.

Coming now to the Andean district, an oil spring is reported on the east flank of the Andes, on the south side of Pastasa river, 130 miles (209 km.) east by north of Guayaquil, and 40 miles (64 km.) from Canelas.

This spring is said to flow continuously. Asphalt exists on Cojitambo Hill, 13 miles (20 km.) northeast of Cuenca. Redwood[15] mentions a less promising occurrence a day's journey north of Quito, where oil is said "to exude from dioritic rocks." Reporting on the increased interest shown in the petroleum possibilities of Ecuador, Frederick W. W. Golding, Consul-General at Guayaquil, writes under date of Sept. 16, 1916:

"Bituminous seams bearing small quantities of petroleum occur in various places in the northern provinces of Ecuador, where several claims have been located. The seepage there has been utilized by the Indians for cooking their food."

3. Geological Conditions

An excellent article on the geology of Ecuador by W. A. and Tedore Wolf[19] and articles by various mining engineers and explorers have given information of value. The eastern range of the Andes consists of rocks of Archean age, while the coastal regions of Ecuador consist of Tertiary deposits; and the intermediate region, which includes the western range of the Andes and the inter-Andean region, is mainly of Cretaceous sedimentary rocks and eruptives. Only the western two of these three belts has any oil possibilities.

The Andes Mountains are of recent geological origin, having presumably been uplifted in the Tertiary period, but certainly later than the Cretaceous. The eastern range of the Andes consists of gneiss, schist, and crystalline slate, and in some places true granite and syenite, all of which are unfavorable to oil. The interesting point geologically seems to be that, while rocks of Archean age exist in Ecuador, there is apparently a great gap extending through the whole of the Paleozoic and the first two periods of the Mesozoic, in which no rocks seem to have been formed. The conditions are quite different from those in Colombia and Peru, where Jurassic and older rocks are sometimes found; and further explorations in Ecuador may reveal them there also. The most important oil belt and the only productive one to date is the western one, where the principal seepages have been found. Judging by deep artesian well borings, the Cretaceous formations near Guayaquil appear to rest directly upon crystalline schists and granites.

In the inter-Andean region, on account of the frequent association of volcanic material and the irregular distribution of the Cretaceous sedimentary rocks, the rocks are not believed to have great oil possibilities. The formations appear to dip mainly toward the west, and strike parallel to the mountains. Another reason for believing the Tertiary rocks of the inter-Andean region unfavorable for oil is that they consist mainly of lacustrine deposits as distinguished from the marine sediments along the coast.

According to Charles Maddock and E. J. Dye, in private reports:

"The geology of the peninsula west of Guayaquil is very well exhibited on the cliffs and presents a series of sandstones, shales and marls which are undoubtedly a continuation of similar beds in Peru and accordingly belong to the Miocene series."

The Negritos field in Peru is in this same formation.

"The petrographical resemblance of the rocks in both the Republics can leave no doubt as to their identity and this constitutes a strong *a priori* argument that inasmuch as the Miocene beds of Peru are highly petroliferous, such will be found to be the case with the Ecuadorean Miocenes when they are subjected to a thorough test with the drill."

The only well which has been sunk passed through formations similar to those of the Peruvian wells. Arnold,[1] however, classifies the productive shales and sandstones in Peru as of Eocene age.

4. Quality

The petroleum of Santa Elena has a specific gravity varying from 0.933 to 0.984 (20^c to 12° Bé.). Stephan states in an unpublished report that the oil found between the Pacific Ocean and the Bay of Santa Elena is dark green in color, and can be classified as a fuel oil containing about 18 per cent. of kerosene. Stephan suspects the existence of lighter oil at greater depths. The gravity ranges between 12° and 22° Bé. The following analysis was made under date of Feb. 10, 1913, by Robert Levi, chemist for The Machala & Santa Rosa Oil Co., Ltd., of Guayaquil. This analysis was translated literally from the newspaper *El Telegrafo* dated Mar. 5, 1913. There is, of course, some chance of error, due to translation and copying.

Color: Black.
Odor: Petroleu.
Reaction: Duly acidified.
Density: 0.884 with 15° of celsitude.

The fractional distillation yields 78 per cent. of the original product, as follows:

Fraction	Temperatures	Parts	Corresponding Per Cent.	Products	Total Per Cent.
1	100°	500	6.041	Benzine and gasoline	8
2	100° to 150°	200	2.055		
3	150° to 210°	400	5.013	Kerosene	55
4	210° to 270°	22,050	28.085		
5	270° to 320°	17,050	22.044		
6	320° to 400°	15,000	19.023	Lubricating oil	33
7	400° to 450°	17,050	8.097		
8	450° to 500°	5,000	6.041		
			96.399		95

[1] References are to bibliography at end of paper.

These results are the average of six samples gathered respectively from "Huaquilla," "Estero narango," "Estero dos bocas," "Pozo salitrillo," "Pozo de piedra" and "Barranco blanco." All of these fields are situated in the Province of "El Oro."

5. Production

The production of petroleum in Equador has not begun, in a strict commercial sense, although Stephan states in a private report that in one place between Santa Elena and the Pacific Ocean about 2000 bbl. per month are produced from small pits.

BRAZIL

The question whether petroleum exists in commercial quantities in Brazil is still open to conjecture. Explorers have studied the matter from time to time, but none of them seem to have discovered real evidences of petroleum in quantity. The present known oil resources of the country are apparently confined to oil shales in the Bahia region and "a rather indeterminate area of favorable oil indications somewhere in the interior." The oil regions of southeastern Colombia, especially that south of Nevea, probably extend over the boundary into Brazil. Branner believes, however, that no great fields are likely to be found in Brazil. Explorations for coal have been made by an American geologist, I. C. White, who likewise was not especially optimistic regarding the country's oil resources, but reported bituminous shale. Among Brazilians there is some talk of petroleum possibilities in the States of Sergipe and Alagoas, but nothing has come of it beyond samples being submitted for analysis. Nevertheless, an underlying hope exists of the ultimate discovery of petroleum fields in Brazil.

Surface Indications and Geological Conditions

Brazil is commonly supposed to have no oil indications, and the reports of travellers into the far interior of that country are superficially not encouraging. A few reports have reached us, however, which indicate some signs of oil in Brazil. Redwood[15] writes as follows on the subject:

"Large quantities of a rich, dark brown, laminated, bituminous deposit, locally known as 'turfa,' occur in the Camamu basin in the Province of Bahia. No signs of plant or animal remains have been found in it, but it has been suggested that it was produced by the aggregation and decomposition of vegetable matter in mangrove swamps, such as still exist in the locality. Small quantities of asphalt have been found in conjunction with the turfa. Mr. Wallace has obtained by distillation 58.48

[15] References are to bibliography at end of paper.

per cent. of volatile matter from turfa dried at 212° F., and estimates that 1 ton would yield 68 gal. of crude oil of specific gravity 0.888, and 6½ lb. of sulphate of ammonia.' . . . . .

"Impure bitumen exists at Morro do Taio, Santa Catharina, and at Piropora in Sao Paulo. . . . . .

"On the River Itahipe, northward of Ilheos, turfa deposits occur near the spur of crystalline rocks that here reaches the coast. The Island of Joas Thania on the Marahu River, about 80 miles southward of Bahia, has rich turfa, and traces of petroleum have been noticed in dioritic intrusions through the shales, further up the river. Fifty miles northward of this, turfa of good quality is found on Tinhare Island. The next noticeable areas are in Alagoas, at Riachadoce and Camarajibe, respectively 25 and 45 miles north of Maceio. On the Parahyba del Norte, petroleum is alleged to occur below Sao Paulo, at Jesus de Tremembe, and oil-shales in the Sierra de Araripe, Ceara."

The *Oil Encyclopedia* states that the Province of Bahia possesses several petroliferous springs, which yield a small quantity of good quality oil, of gravity 28° Bé., but the deposits have been only partly explored.

James Orton[14] gives some geological descriptions, which, when analyzed, show nothing actually favorable for oil, but indicate that its occurrence is not impossible.

There is no doubt that certain formations which contain oil in Colombia, Bolivia, and Peru, extend into Brazil. It seems probable, though, that the oil shales of Brazil may demand attention before the actual deposits of petroleum are developed.

According to Redwood,[15] impure bitumen occurs:

"probably in outlying portions of the great belt of bituminous shales of Eocene age, that extends intermittently from Porto Alegre along the coast to near the mouth of the Amazon, a range of 18° of latitude. The granite and other ancient rocks against which the system abuts landwards reach the coast in occasional spurs, between which the Tertiary deposits occupy bays of 20 to 50 miles in depth. Oil shales suitable for distillation, and locally termed turfa, probably occur more or less throughout this vast area, but have been specially noticed at the following points."

Arnold[1] says:

"The oil shales are of Eocene age and extend intermittently from Porto Alegre along the coast for over 1200 miles, nearly to the mouth of the Amazon. Portions of the shales suitable for distillation have been reported at the following, among other localities: North of Ilheos, on Itahipe River; on the Island of Joao Thanis, in Marahu River, 80 miles south of Bahia; on Tinhara Island, 30 miles south of Bahia; at Riachadoce and Camarajibe, 25 and 45 miles respectively north of Maceio; in the Province of Alegoas; and in the Sierra de Araripe, in Ceara."

According to Day,[4] a very extensive deposit of rich shale was examined by Branner which is said to contain more oil than the shales of the Camarajibe district. An average of five samples shows 33 per cent. of volatile hydrocarbons. In describing these deposits Branner says:

[15] References are to bibliography at end of paper.

"The oil shales of the Brazilian coast are of Cretaceous age, and the parti-colored beds exposed in the bluffs along that coast are for the most part the weathered portions of this same Cretaceous series. The Cretaceous strata rest upon granites, gneisses, and other crystalline rocks, with a bed of very coarse conglomerates forming the base of the series. The only known exception to this is in the Sierra d'Itabaina, in the State of Sergipe, where there is a series of beds between the granites and the Cretaceous that appear to be Paleozoic, though no fossils have been found in them. The failure of the Marahu company was evidently due to extravagance and mismanagement, and cannot be regarded as a sufficient reason for condemning the oil shales of Brazil as unworkable.

"The total thickness of the Cretaceous beds does not much exceed the total thickness of the mottled and parti-colored beds exposed on the coast—that is, from 30 to 90 m. (100 to 300 ft.). This is shown by the fact that at many places the basal conglomerates are exposed, while at several points the crystalline rocks themselves are uncovered.

"No oil shales are now known in Pernambuco, Parahyba, Rio Grande del Norte, Sergipe, or Espirito Santo; but they may be expected in any of those States within the Cretaceous area."

CHILE

1. Distribution of Chile Fields

Although surface evidences of petroleum are not abundant, indications exist, according to Redwood,[15]

"in northern Chile, in the Province of Tarapaca, south of Patillos. In the southern part of the republic an extensive area southward of the Maullin River is said to have indications of natural gas from Tertiary deposits. Oil is also reported to have been met with at Puerto Porvinir and Agua Fresca in the Magallanes Territory."

Northern Region

As stated in the *Pan-American Bulletin:*

"The Copacoya petroleum fields in northern Chile are situated near the Bolivian frontier on the eastern slope of the valley formed by the Tatio Mountains, in latitude 22° 10′ 25″. In the central part of the valley and about 12 km. distant from the petroleum zone are 30 or more saltwater geysers and hot springs. The petroleum zone is not under exploitation and has not been fully explored, but experts report that there are unmistakable indications showing the existence of petroleum in the region referred to."

Southern Region

According to a statement from Commercial Attache Verne L. Havens of Santiago, dated Jan. 25, 1915, "the country surrounding Punta Arenas in Chile contains petroleum and asphalt." Small quantities of asphalt are said to have been found throughout the Punta Arenas region. The *Petroleum Review* states that deposits of petroleum are known in southern Chile in a zone extending from Chiloe to the Maullin River, the

first discovery having been made near the town of Carelmapu. This report has not been corroborated.

2. Surface Indications

The indications which have contributed to the common belief of the existence of oil fields in Chile are as follows:

1. Asphalt.
2. Gaseous emanations.
3. Bitumen brought up by the tide.

1. *Asphalt*

In the country surrounding Punta Arenas, small quantities of asphalt have been found for years.

2. *Gaseous Emanations*

According to a Chilean Government report,[7] emanations of inflammable hydrocarbons have been observed for several years in the neighborhood of Punta Arenas and Tierra del Fuego. These indications were the result of several tests made to ascertain the existence of oil. Although the first tests were fruitless, a gas emanation was encountered in October of last year at a depth of about 300 ft., near Minas River, 1 mile (1.6 km.) west of Punta Arenas. Thin oil films were reported on the water which was pumped out. At the time of this discovery the Minister of Industry and Public Works authorized a geological study of the locality, resulting in the necessity of a thorough investigation in the vicinity of Punta Arenas to ascertain the existence of oil.

Gaseous emanations have been found on Dawson Island, in a deep cove 1000 ft. (304 m.) inland near Harris Bay (latitude 53° 50′, longitude 70° 26′) and in Lomas Bay, on the western coast in latitude 53° 49′, longitude 70° 34′.

According to the same Government report, a gas spring exists in the Strait of Magellan, at the mouth of Quemas Malas River in latitude 53° 23′, longitude 70° 57′, where numerous bubbles can be seen at low tide. The gases produce a strong odor much like hydrogen sulphide, are very inflammable and give a dim yellow flame. When set on fire they are extinguished only by a strong wind or by high waves of the sea.

According to the *Petroleum Review*, a sample of this gas sent to Paris for analysis gave the following results:

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[7] References are to bibliography at end of paper.

Elements	Per Cent.	Per Cent.	Equivalent in Combined Gases
Oxygen	3.21	15.28	Contained air.
Nitrogen	12.07		
Oxygen	1.44	1.62	Water vapor.
Hydrogen	0.18		
Carbon	68.77	83.10	Hydrocarbons.
Hydrogen	14.33		
Totals	100.00	100.00	

Consul Alfred A. Winslow, in the *Daily Consular Trade Reports*, mentions the discovery of natural gas in the township of Carelmapu, in the Province of Llanquihue, about 500 miles south of Valparaiso, adding:

"Enough gas pressure has been secured to run a cook stove, a heating stove, and two gas jets at one time. Soundings have been made to the depth of 500 ft. with good results."

3. *Bitumen Brought Up By the Tide*

The government geologist, Dr. Ernesto Maier of the University of Chile, describes[7] asphaltum having a metamorphosed appearance in Lomas Bay on Dawson Island. The waves of the sea frequently throw on the coast pieces of a dark material resembling coal in color, and which for some time was supposed to come from a colliery on the coast near by. Another investigator, Fortunato Ciscutti, found blocks of the same material on the land, but was unable to discover the outcrop. An analysis of the substance made in 1909 in the laboratory of R. H. Harry Stanger, London, gave the results shown in Table 10.

TABLE 10.—*Analysis of Substance from Dawson Island*

Specific Gravity at 60° F., 1.1342	Per Cent.
Calorific power, 8875 calories	
Water	1.64
Volatile substances at red heat	73.51
Ash	3.74
Coke	10.91
Total	99.80
Absolute Analysis of the Dried Substance	
Carbon	76.47
Hydrogen	7.91
Oxygen	8.73
Nitrogen	1.93
Sulphur	0.97
Ashes	3.99
Total	100.00

3. Geological Conditions

The following remarks are extracted from a Government report[7] referring to the question of the existence of petroleum in Magallanes:

"The Cretaceous and Tertiary sediments of the coast of Chile are not known in the vicinity of Magallanes. The marked distinction between the coastal range of mountains and the main Andes range exists in Magellan as elsewhere, so that when crossing the Straits of Magellan to the east from the islands west of the Jeronimo canal, the mica-schists of the coastal range can be seen traversed by enormous dioritic dikes. The main range begins east of the Jeronimo Canal, which is principally composed of metamorphic rocks, having at this point an approximate width of 50 km. Neither of these regions is suitable for oil fields.

"On the eastern side of the range the first fossiliferous beds are in Mount Tarn and near Port Hambre, consisting of Lower Cretaceous calcareous clays and slates overlain south of Agua Fresca Bay by the Tertiary beds. These Tertiary and more recent formations occupy the entire eastern zone of the Magellan lands, with the exception of scattering basaltic masses. Hence, if oil exists in this territory, it must be looked for on the eastern side of the range."

Dr. Johanness Felsch, Geologist of the Ministry of Industry and Public Works, states as follows:

"Since the older formations rise to the surface in approaching the main range of mountains, the oil-bearing strata, if present, will be found in this region at a less depth than farther from the mountains."

The following geological particulars are extracted from a translation of *Geology of the Neighborhood of Puntas Arenas and Tierra del Fuego* by Felsch:[7]

"The escaping gases were proved to emanate from Tertiary strata, and in the immediate vicinity of the gas seepage near Boqueron Cape in Tierra del Fuego and in the valley of Tres Brazos River, 20 km. south of Punta Arenas, rocks of Tertiary age were found to contain drops of oil."

The purposes of the Government investigations were as follows:

"*First.*—To ascertain the stratigraphic succession underlying the Tertiary strata.

"*Second.*—To determine the geological structure, especially in the vicinity of the gas springs.

"Cretaceous strata are found south of the Tertiary region in a belt running southwest to northwest, and bounded on the southwest by Port Yartan, Tierra del Fuego, Port Hambre, Peninsula Brunswick, Amarillo River, Tres Morros and Punta Steinmann."

BOLIVIA

1. Distribution of Bolivia Fields

The petroliferous fields of Bolivia, situated in the Andean district, lie along the southeastern frontier of the country, and indications are quite

[7] References are to bibliography at end of paper.

continuous in a belt extending northwest and southeast through a distance of 150 miles (241 km.) as far as the Argentina boundary at Yacuiva. The zone is a diagonal one, traversing the eastern provinces of Santa Cruz, Sucre and Tarija, between parallels of 63° to 64° West longitude and 19° to 22° South latitude. According to M. A. Rakusin in the *Troudi* of the Grosny branch of the Russian Technical Society, published in 1913, there are at least three fields, viz.: (1) Piema, (2) Kuarazuti and (3) Lomas de Ipaguaciu. The productive formations are supposed to extend from near Santa Cruz in the center of Bolivia, southward through Sauces to Piquirenda, Plata and Guarazuti, in the Province of Tarija and into northern Argentina.

Arnold[1] states that:

"Geologic investigations in the area between the Incahuasi and Aguaraygus ranges, south of Sucre, have shown the presence of a considerable area of prospective oil land."

According to the *Pan-American Bulletin:*

"Careful geologic investigations of the eastern slope of the Bolivian Andes confirm the claim that a petroleum belt exists along the entire range of these mountains from Ayacuiba to the Madre de Dios River."

The *Pan-American Bulletin* mentions a narrow belt in eastern Bolivia, on the foothills of the Cordillera Real," extending in the direction of Santa Cruz, to the village of Monte Agudo, a distance of some 300 miles." Indications are also reported in the Beni district. In view of the cumulative evidence, it seems likely that Bolivia will some day be a great producer of petroleum.

2. Surface Indications

According to Redwood:[15]

"Traces of oil are recorded in Chiquitos and Cordillera, at Sauces and near Santa Cruz. Further southward eleven copious springs of petroleum are mentioned as occurring within an area of some 40 sq. miles in the Province of Tarija, close to the Argentine border at Piquirenda, Plata, and Guarazuti. These are derived from the same Lower Cretaceous rocks as those immediately across the frontier, which probably constitute, as there, a second belt about 100 miles west of that indicated."

The existence of oil has long been known from seepages, and recently these have been found to occur continuously along a belt extending NW.-SE., over 150 miles, down to the Argentine boundary at Yacuiva. Thus the petroliferous zone extends diagonally across the Provinces of Santa

[15] References are to bibliography at end of paper.

Cruz, Sucre and Tarija, between parallels of 63° to 64° West longitude and 19° to 22° South latitude, and thence far into Argentina.

A report made by Director-General Barrett, of the Pan-American Union, states that the Espejos (Mirror) Spring, 36 miles from Santa Cruz, is a fair sample of the character and kind of surface indications in the region between the northern boundary of Argentina and the Madre de Dios River in northern Bolivia. This river is close to the southern boundary of western Brazil, and with the Beni, into which it runs, finally joins the Mamore River to form the Madeira.

3. Geological Conditions

The oil along the eastern base of the Andes in Bolivia comes, according to Arnold[1], chiefly from Lower Cretaceous dolomites. The formations are much folded and faulted. According to Rakusin, the three principal petroliferous areas in southern Bolivia belong to the so-called "Sistema de Salta," extending from northern Argentina into the center of Bolivia. The strata are of the chalk formation and, according to him:

> "all the most important petroliferous areas of South America must belong to the chalk formation, with the only exception of the petroliferous deposit of Kacheuta Mendoza, which belongs to the Upper Trias."

According to the same writer, the oil seepages at Peima are found in two conglomerate horizons separated by clay slates, and similar conditions are said to exist in Kaurazuti. The deposits of Ipaguazu seem to be different, however. "An anticlinal of average depth" exists in the Peima fields, and the Kuarazuti structure is reported as a continuation of the same folds a few miles distant and identically similar. It is reported that the probable thickness of overlying strata in the southeastern Bolivian fields ranges from 160 to 660 ft. (48.7 to 201 m.). If this be true, the cost of development may be correspondingly small. No wells have been sunk in this neighborhood, and investigations have gone no further because of the inaccessibility of the localities. According to Arnold, "Springs and seepages of oil issue from Lower Cretaceous dolomites along the eastern base of the Andes." The *Bulletin* of the Pan American Union[1] says that: "Extensive petroleum deposits of good quality have been discovered at Calacota, on the Arica and La Paz Railway." This, if true, is a continuation of the Titicaca fields of Peru.

4. Quality

Little is known of the quality of Bolivian petroleum, but some analyses shows an extremely high percentage of lubricating substances. The oils

so far recovered vary in composition, from those of asphaltic base with specific gravity 0.975 (13.5° Bé.) and containing 4 per cent. gasoline to an oil of 0.810 specific gravity (43° Bé) containing 40 per cent. gasoline. The heavier oil is confined in the lower sands. Along the eastern base of the Andes, the petroleum is of high grade, from 35° to 47° Bé.

According to Rakusin, in the *Troudi* of the Grosny branch of the Russian Technical Society published early in 1913, a test of Bolivian petroleum has given the following results:

Color	Light brown
Specific gravity	0.898 (26° Bé.)
Viscosity in Engler's apparatus at 35° C.	4.20
Viscosity in Engler's apparatus at 70° C.	1.70
Flash point	70° C.
Calorific value, calories	10,864

The oil does not give off any light distillates, but 31 per cent. of kerosene is present. The remainder consists of heavy residuum, of which sulphur is 0.07 per cent. Mr. Rakusin has made polarimetric investigations which point to the conclusion that the petroleum has filtered up from considerably deeper strata. According to the *Pan-American Bulletin,* in reference to the petroliferous belt from Ayacuiba to the Madre de Dios River:

"The Espejos petroleum spring, 12 leagues from Santa Cruz, is a fair sample of the quality of petroleum encountered in the belt referred to. Oozings taken from the surface flow of this spring, according to the recent analyses made in London, contain 78.2 per cent. of kerosene, 17.5 of lubricating oil, and 4.3 per cent. of coke. No shafts have been sunk in this neighborhood, but the indications would seem to argue that gushers may be found by boring to the proper depths. Up to the present time, the Espejos spring remains unexploited."

Pools of relatively pure oil and also of "pitch" exist in eastern Bolivia, on the foothills of the Cordillera Real. These indications extend in the direction of Santa Cruz to the village of Monte Agudo, a distance of some 300 miles (482 km.). This oil-bearing belt appears to be comparatively narrow. Similar indications are reported in the Beni district, but they are less well known.

5. Remarks on General Conditions

No oil has yet been produced in Bolivia, and the transportation difficulties have been cited as an argument that oil is not likely to be produced in the near future. The only way of reaching the Santa Cruz region at present is by cart road across Bolivia or Argentina. Two railroads are projected to the fields, however, one to run west from the Paraguay River and a second north from the Argentine frontier, and these will tap the oil region.

THE GUIANAS

British Guiana

Persistent rumors have come in from time to time of oil in British Guiana, the most authentic being the following quotation from a report by G. E. Chamberlain,[3] Consul at Georgetown:

"After an extended investigation of the Waini River district in the autumn of last year, Mr. E. C. Buck, director of public works, reported that the oil indications were most favorable."

Nothing further is known of indications in British Guiana, beyond the fact that Redwood[15] mentions some "finds" of asphalt near the coast.

Dutch Guiana

According to Redwood[15] "a work of the 18th century refers to the bitumen of Surinam," and Arnold[1] states that oil seepages have been known in that country since the 18th century and that three localities are known, as follows:

1. On the south side of the Surinam River, 6 miles (9 km.) below Kabele station and 97 miles by rail south of Paramaribo, where exposures of shale and sandstone yield small quantities of high-grade amber-colored oil.

2. An area on the Marowijne River, 100 miles (160 km.) above Albina, where the seepages are in shale, and the quality of the oil is excellent.

3. Between Surinam River and the railroad, about 48 miles (77 km.) above the head of deep-water navigation, where the gravity of oil is about 45° Bé., occurring in small seepages. The seepages on the Marowijne River are found in shale, cut by serpentine dikes, but 5 miles from the nearest serpentine. In general, the geology of the country is not supposed to favor large pools.

French Guiana

According to Arnold[1] oil seepages occur southeast of the Marowijne River, the formations being continuous with those having seepages in Dutch Guiana, but he believes that the possibilities are insignificant.

THE FALKLAND ISLANDS

An article published in 1912 in the *Bulletin* of the Imperial Institute describing the mineral deposits, states that a sample of bitumen was received from the Falkland Islands in September, 1909. This sample consisted of dull black bitumen, having a specific gravity of 1.01, and a letter accompanying the sample mentioned several similar outcrops in

[3] References are to bibliography at end of paper.

various parts of the Islands. The material ignited easily, burning with a long luminous smoky flame, and leaving a reddish ash which contained a high percentage silica. The analysis is as follows:

Analysis of Bitumen from Falkland Islands

Material	Per Cent.
Volatile matter and moisture	88.0
Fixed carbon	8.3
Ash	3.7
Total	100.0
Sulphur	1.12 per cent.
Calorific value	9568 calories.

The material was slightly soluble in chloroform and turpentine, both of which dissolve bitumens readily. When destructively distilled it yielded much gas, and left a small quantity of coke. Distilled at a lower temperature at about 500°, the sample furnished oil amounting to 75 per cent. of the total weight; this oil being dark greenish-brown in color, with a specific gravity of 0.892 at 15.5° C. The flash point was under 40° F., rendering the substance unsuitable for use as a liquid fuel.

On submitting the oil to fractional distillation light petroleum commenced to distil off at 48° C., and amounted to 15.3 per cent. by weight of the crude oil, and 11.5 per cent. by weight of the bitumen. It had a specific gravity at 15.5° C. of 0.740 and only a slight color. The residue remaining after the removal of this light petroleum, and which may be termed fuel oil, had the following characteristics:

Specific gravity at 15.5° C.	0.918 (23° Bé.)
Flash point (by Abel closed test)	74° C.
Solidifying point	1° C.
Calorific value	10,551 calories
Sulphur contained in the oil	1.14 per cent.

This fuel oil amounted to 63.5 per cent. by weight of the "bitumen" distilled. The chemical name of the mineral is not given, but it resembles albertite, which was at one time mined on a considerable scale in New Brunswick in Canada, previous to the discovery of petroleum in that country. The principal value of the discovery in the Falkland Islands is in its indication that the country was once petroliferous and that traces of desiccated oil still remain in the strata.

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DISCUSSION

I. C. WHITE, Morgantown, W. Va. (written discussion*).—The writer's conclusions with reference to the possibility of finding commercial deposits of petroleum in Brazil were published in 1908 in the Final Report of the Coal Commission of Brazil.[1] Since this volume is not generally accessible, and the author has nothing to add thereto, the following quotation found on pages 243–7 may prove of interest:

The Possibility of Petroleum Deposits in Brazil

"The writer has often been asked to express an opinion as to the possibility of finding petroleum in commercial quantity in Brazil.

"The question is one that does not admit of an adequate answer except for the regions visited by the writer, which as already stated, include only a portion of that great territory, and only the southern portion of Brazil; namely, the states of Rio Grande do Sul, Santa Catharina, Paraná, and a few rapid excursions across São Paulo, and Minas Geraes, as well as the brief visit to Marahú in Bahia.

"That the sedimentary beds of the Brazilian Permian once held some petroleum is attested by the evidence of the Iraty black shale which gives off the characteristic odor of this substance from São Paulo to Rio Grande do Sul.

* Received Oct. 12, 1917.

[1] Commissão De Estudas Das Minas De Carvao De Pedra Do Brazil, Relatorio Final Por I. C. White, Chefe Da Commissão, Rio De Janeiro, Imprensa Nacional, 1908, Quarto.

"The natural coke at Limeira and the Albertite near Lages in this same shale also attest its petroliferous character.

"The sandstone saturated with asphaltic residua at Bofete in São Paulo, tells the same story; namely, that these deposits of the Santa Catharina system of rocks once held petroleum, and if reports are to be credited, a small quantity of petroleum was actually found in the deep boring at Bofete near the base of the Santa Catharina beds. But another factor must be taken into account. The rocks of this system are everywhere in Brazil traversed by fissures through which great flows of eruptives welled out to the surface, rupturing the strata and often baking them into a partial metamorphic condition. These seismic activities would certainly release all or most of the volatile products in any petroleum deposits, and leave the remainder as an unavailable asphaltic residuum just like that found in the sandstone at Bofete.

"It is true that large areas are comparatively free from any visible evidence of these old eruptive flows, and it is barely possible that in some of these, petroleum deposits might yet exist in available form, but this is quite doubtful, and the chances are all against the finding of petroleum in commercial quantity anywhere in South Brazil.

"The result of the deep drilling at Iraty, which is distant from any known outcrop of igneous rocks, confirms this conclusion, that it is useless to expect petroleum deposits of any considerable quantity anywhere in South Brazil.

"What North Brazil may hold in this line, the writer cannot venture to predict, since his travels have not extended into that region, but if one might reason from the presence of the great deposits of asphalt in the adjoining countries of Venezuela, etc. (since all deposits of asphalt are simply *residua* of former immense oil pools elevated to the surface, the original cover having been removed by erosion) one might foretell that if any large deposits of petroleum are ever found in Brazil, they will be located in the territory drained by the great Amazon river of North Brazil.

"The reported occurrence of petroleum in Porto Bello, Enseada do Brito, and other points along the coast of Santa Catharina, are without foundation, since the rocks are granite at all of these localities; while a specimen of petroleum collected from the region south of Marahú, in the state of Bahia, where petroleum is reported to exist on the water, proved to be the refined article, and was doubtless derived from one of Devoe's packages."

The Estimation of Petroleum Reserves*

BY ROBERT W. PACK,† WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

INTRODUCTION

SOONER or later in the development of any natural resource it becomes highly desirable to know the quantity of this resource in the country as a whole, as well as of the part that is being developed, for the general course and conduct of the development work must be governed, to some extent, by the total supply available. An accurate estimate of the quantity of any natural resource is difficult to make; yet if one is diligent enough one can obtain a pretty accurate idea of the quantity of marketable timber, the acreage of tillable land, or the horsepower that may be developed from the streams, for these are things that may be measured. But any attempt to estimate the quantity of some mineral in the ground appears to enter the realm of imagination and it seems that the estimate can be nothing more than a blind guess. It may appear to be especially futile to attempt to determine the quantity of oil and gas that is available in the ground, for these minerals are so vagrant that even if the quantity now available should be known quite exactly, the quantity that will eventually be obtained may be much less, for certain losses in the extraction of both oil and gas are unavoidable, and just what these losses will be cannot be predicted.

Though necessarily inaccurate, estimates of such mineral reserves are, however, of distinct value, for they summarize the existing knowledge of the deposits of the mineral and thus enable those who are interested in some industry that depends in one way or another on this mineral to formulate a rational commercial policy.

Some time ago, when the price of gasoline suddenly rose in a rather spectacular manner, the writer was one of the U. S. Geological Survey geologists called upon to help to estimate the quantity of oil that could be obtained in the United States, for the fear had become widespread that perhaps the rate of increase in the price of gasoline was a measure of the rate of depletion of the supply of petroleum. It thus became his

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* Published by permission of the Director, United States Geological Survey.

† Geologist, U. S. Geological Survey.

official duty to make some sort of an estimate of the quantity of oil that still remained available in the ground in California. In making this estimate the writer attempted to predict the future output by a study of the past output of the productive fields, for such a mode of prediction is generally by far the most satisfactory one that can be used and has been employed, in one form or another, a great many times. There are, however, various ways in which historical and statistical data may be used, some being best suited for one set of conditions, some for another. The writer believes that the particular methods which he used in handling the data, and which are described below, are peculiarly adapted to the conditions in California, and that they are in considerable part new.

It is the purpose of this paper (1) to outline briefly one or two of the methods that have been used by others in estimating the available supply of oil, and (2) to describe somewhat more in detail the methods used by the writer, and to state why such methods were chosen and what degree of accuracy can be attained by using them. The general method followed might be applied to any oil field, provided the necessary data are available.

GENERAL METHODS

There are two general methods of estimating the quantity of oil and gas in the ground. The first of these to be described is what is commonly known as the saturation method; the second has been termed the production curve method. The two methods are not quite comparable, for the first is used to determine the quantity of oil in the ground, and the second the quantity of oil recoverable, assuming that the conditions governing production will continue to be about the same as they have been in the past. The two methods likewise have separate applications, for the saturation method may be used to determine roughly the quantity of oil or gas in an area that has not been tested, whereas the production curve method is applicable to a partly developed field but only indirectly to an undrilled area.

Saturation Method

The saturation method consists essentially of a determination: (1) of the volume of the strata that serve as reservoirs for the oil; (2) of the porosity of these strata—that is, of the "voids" in them—and thus of the maximum volume of the space that may be occupied by the various fluids; and (3) of the percentage of the "voids" that is occupied by oil. This method is the one that has commonly been used ever since estimates of the oil reserves of the country were first attempted. The method has recently been described in some detail by Washburne.[1]

The saturation method obviously involves the consideration of several

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[1] C. W. Washburne: *Trans.* (1915), **51**, 645.

factors that cannot be measured directly. The first factor to be considered—the volume of the sand that serves as a reservoir—is the factor that may be determined most accurately, and in fields where the strata show relatively little variation in lithology and thickness over large areas, such as the Appalachian fields, it may be possible to calculate the volume with considerable accuracy from measurements made on the outcrops of the oil- or gas-bearing sand, or from those given by a few scattered wells. In a very great many areas, however, and of these the California fields are fairly typical, the formations are so lenticular and the reservoir sands so inconstant both in thickness and in lithology that no reliance can be placed on the calculations of the volume of the reservoir made in this manner, even if the points at which the measurements were made lie fairly close together.

The second factor, the porosity of the reservoir sand, is even more difficult to estimate. In making an estimate of the oil, it is clearly useless to determine the porosity of the oil-bearing sands in areas where the total volume of the strata cannot be determined with some degree of precision; but even in areas in which the thickness of the formations is fairly constant, and in which the volume of the beds acting as reservoirs may be calculated with fair precision, the porosity may vary greatly, either because of variation from place to place in the grain of the rock or because of the local introduction of a secondary cement that occupies a considerable part of the original void space of the rock; and the variation may be so great that the space which may be occupied by fluids cannot be calculated. In these areas the porosity of the oil-bearing bed as determined from samples of rock taken on the outcrop, or from a few samples obtained in drill holes, is apt to be very different from the average porosity of the stratum over a large area.

The third factor, the percentage of the void space that is occupied by oil, is, of course, not directly measurable, so that it is necessary to assume either that the voids are all so occupied or that some arbitrarily chosen proportion of them are.

Should all these different factors be correctly determined, the resulting figures will record the total quantity of oil in the field. Not all this oil is recoverable, however, and the figures so obtained must be multiplied by an "extraction factor" in order to obtain figures that will represent the available oil.

Even under the most favorable conditions, then, the saturation method is capable of giving only a rough approximation of the recoverable oil in any area; but rough as it is, this method has a distinct value, for it will give directly a first approximation of the maximum amount of oil to be expected from an untested or very sparsely drilled area in which the formations are fairly constant in thickness and lithology. This method is not, however, applicable to areas where oil is held in solution

cavities or in fracture zones, nor is it of any very great value in the consideration of pools that occur in the variable Tertiary strata of California.

Production Curve Methods

The production curve method is a mode of predicting the future production of an area and is based on a study of the past production of that area. The method is largely graphic, and although various modifications of it have been used all are essentially extrapolations of one or more curves that show the production or development of the field. The advantages of this method over the saturation method are: (1) it is concerned with actual production, so that the assumption of any "production factor" is unnecessary; (2) the factors that must be estimated or assumed are fewer, and on the whole less liable to great variation, than those that must be estimated or assumed for the saturation method.

Two general methods of using production figures are employed in estimating the reserves of oil in a given area. One uses figures showing the production per unit of time of the whole field or of a group of fields within a given area; the other concerns itself only with the production of certain small areas, which are considered as types, the results obtained in their study being applied to the field as a whole.

In support of the first method, it has been urged that a consideration of the combined production of a large number of wells, and of wells distributed over a large area, automatically takes account of various irregularities in the production of individual wells—of irregularities due to variation in demand, to increased drilling, and to the discovery of new pools—it being always assumed, of course, that the conditions in the future will vary about as they have varied in the past. But in the very fact that the curve incorporates these factors lies its weakness, for in the nature of things drilling cannot continue indefinitely into the future, nor is it reasonable to expect the continuous discovery of new pools. These two factors should therefore be taken into account separately, and their effect should be eliminated from the curve showing total production.

The second method—that of studying the history of a small group of wells—has an advantage in that it permits a more intensive study, but it has a disadvantage in that the data necessary for this study are much more numerous and are seldom available. Moreover, in applying to the whole field the results obtained in the study of a small part of it, it is necessary to estimate the degree in which this area represents the conditions that exist throughout the field.

Method Using the Record of the Production of the Entire Field

Projection of Curve Showing Quantity of Oil Produced.—Probably the simplest method of predicting the future of a field is to project the curve that represents the field production per unit of time. Such a method

will give reasonably correct results only for an old field in which the production of oil per unit of time has been constantly decreasing. When applied to the Appalachian fields it gives fairly satisfactory results. The method of representation shown in Fig. 1, in which the shaded area below the line *ABC* represents the volume of oil that has been produced to date from an old field, that has long been producing a constantly decreasing quantity of oil in a given length of time. Assuming that this rate of decrease will continue until the field is completely drained, the amount of oil now remaining in the field is represented by the unshaded area below the line *CD*.

This method of prediction is open to the criticism that it does not discount the effect of new wells but that it tacitly assumes that new wells will continue to be drilled up to the very moment that the pool is drained,

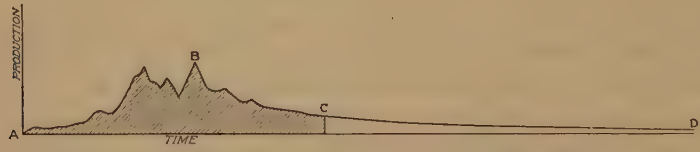

Fig. 1.—Estimation of reserve of petroleum by extrapolation of curve showing field production in unit time.

or, if the curve represents the production of a large area rather than of a single pool, that new pools will continue to be discovered up to the moment when the oil in the region is exhausted.

Curve Showing Percentage of Decrease in Production.—A variation of the graphic method of estimating reserves just described is to plot, not the figure showing the actual quantity of oil produced, but a figure showing the relation between this quantity and the maximum quantity of oil that the field has produced during some unit of time. The curve so obtained—the so-called percentage curve—was used by Arnold, and Fig. 2 is adapted from his plate.[2]

This mode of estimating oil reserves, like that just described, is applicable only to those fields that have passed the zenith of their productivity and are producing in any unit of time a constantly decreasing quantity of oil. The curve in Fig. 2 represents the life of the field after the time when the maximum yield per unit of time was obtained. It is so constructed as to show the quantity of oil obtained during each unit of time that followed the period of maximum yield, the quantities being expressed in percentages of that maximum. This method is also open to the objections that have just been raised against the method of projecting the curve that shows the total production directly, particularly to the objec-

[2] Ralph Arnold: The Petroleum Resources of the United States. *Economic Geology* (December, 1915), **10**, 695–712.

tion that it in no way takes account of the effect of new wells. The sole object of plotting the statistics of production in this form was to obtain a curve that would serve as a type to show decline in production for any and all fields. The rate of decrease in production, however, is so different for different fields that it seems very doubtful whether a single curve can express the decline for more than a single field, or perhaps for a group of fields, in rock of the same general type or of the same general geologic structure.

This method of plotting production in terms of percentages of total production is essentially an attempt to apply to the country as a whole

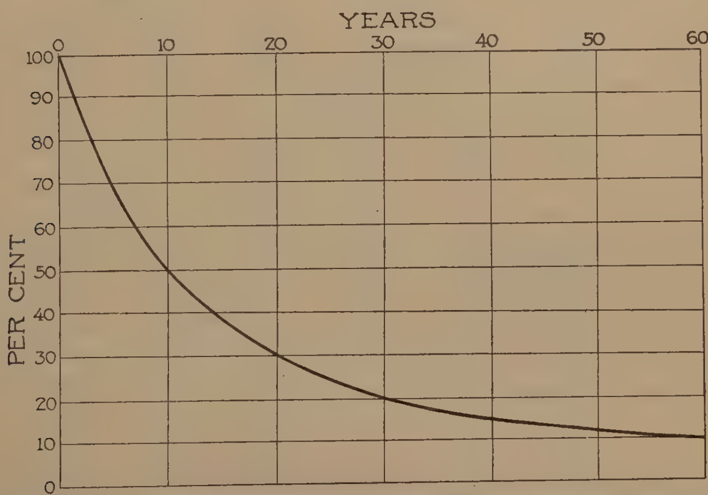

FIG. 2.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF "THEORETICAL CURVE OF FINAL DECREASE."

the methods employed by the California evaluation committee, whose work is explained below. That committee, however, concerned itself with small areas for which a mass of detailed information was available and constructed a number of curves, each showing the rate of decline for a certain particular field. An inspection of these curves, all of which were prepared for fields in California, shows that even within that State the decline is different for different fields. It is evident that similar curves constructed for fields in Oklahoma and Pennsylvania would show still greater differences. Moreover, the production curves of the evaluating committee were so constructed that the effect of continued drilling is to a measure discounted. The "theoretical curve of final decrease" used by Arnold does not in any way consider the effect of drilling new wells, but, like the method shown in Fig. 1, assumes that drilling will continue up to the very moment that the field is totally drained. This method, therefore, has less to recommend it than the method of projecting the curve showing the exact quantity of oil produced in some unit of time.

Method Using Record of the Production of Small Areas

Curve Showing Production of the Average Well.—Some time ago, when a plan was on foot to combine certain producing properties in California, it became necessary to determine the relative value of the different properties, and especially the reserves of oil contained in each. A committee, of which M. L. Requa was the moving spirit, attempted to determine the reserves by making a detailed study of the history of the wells. The method that they followed was essentially the study of a small area in great detail, and the application of the results of this study to the field as a whole. The immediate problem was not to determine the reserve

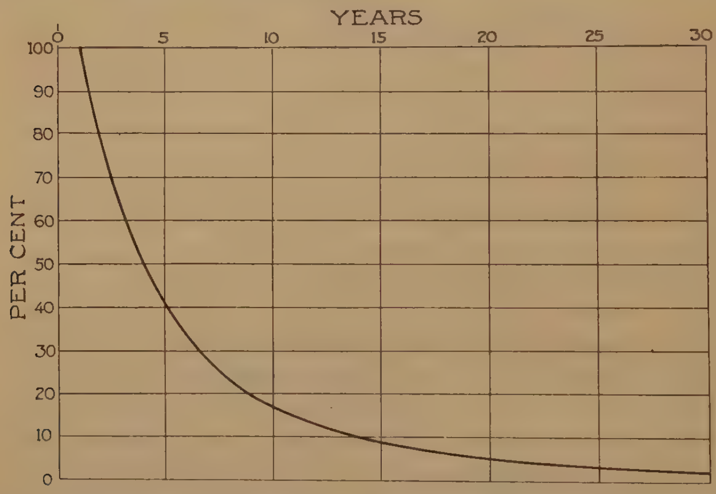

FIG. 3.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF CURVE SHOWING PRODUCTION OF AVERAGE WELL EXPRESSED IN PERCENTAGES OF PRODUCTION OF WELL DURING FIRST YEAR OF ITS LIFE.

of oil in the whole field, for that was of only incidental interest, but the amount remaining in certain parts of the field. The areas studied, however, were regarded as types of the fields or of the parts of the fields in which they lay, and the curves showing the predicted production for the small parts were applied to the whole field.

Then, too, the manner in which they used the record of the production of the wells was unique. Instead of plotting a curve that showed the rate at which oil had actually been taken from the area—that is, instead of constructing what might be termed a chronologic curve—the figures showing the production of the wells were so grouped that the quantity of oil obtained from a well during each year of its life was grouped with the quantity of oil obtained from all the other wells during the corresponding years of their lives, irrespective of the actual time at which the wells were drilled. Thus, although well B may have been

drilled 10 years later than well A, the oil that well B produced during each year of its life was grouped with the oil that well A produced 10 years before. The figures showing the production of the wells as thus grouped were then expressed in percentages of the production of the first year, and this curve was taken to represent the production of the average well in the area. To get a figure representing the quantity of oil that the average well will produce, it is necessary only to determine what the average production during the first year of their lives has been for the wells that have been drilled, and to substitute this number in the curve. The oil produced during the following years is then expressed in percentages of this number.

In order to obtain a figure representing the total production of an area, it is necessary to determine the number of wells that would be necessary to drill the area completely, and to multiply this number of wells by the figure showing the total quantity of oil obtainable from the average well, as shown by the curve. A curve constructed for one of the California fields is shown in Fig. 3.

This method has the distinct advantage over the methods hitherto described in that it attempts to determine the effect upon production of the rate at which new wells are drilled. The degree to which this discounting is really effected is, of course, dependent in large part upon the degree to which the property is drilled, for the mutual interference of wells increases as wells are multiplied.

METHODS USED BY THE WRITER

In casting about for some way in which to estimate the oil reserves in California, the writer, after considering the methods just described, concluded (1) that the production curve rather than the saturation method should be used in estimating the reserves in the producing fields, and (2) that the untested areas should be estimated by comparing them with the producing areas that they most closely resembled geologically, after the reserves in these areas had been estimated by the production curve method. The best way of handling the production record did not seem to be quite so clear, for none of the methods appeared quite to fit the requirements. It seemed better to use a method that considers the production of the whole field rather than one that considers only a small part of it, both because the data necessary for the detailed study of a small area are seldom available and because of the difficulty in choosing areas that might fairly represent the whole field. It appeared, however, that in dealing with the record of the total production some way should be found to discount the effect of the new wells.

Methods Using Figures Showing Field Production

In order to fulfil approximately these requirements, the method

described below was devised. Fig. 4 shows the application of the method to a California field:

1. The field production per unit of time was plotted and a curve, *ABCF*, similar to that in Fig. 1 was obtained.

2. Two points, *B* and *C*, which represented dates when the field production was the same, were selected. These points were as far apart as possible and corresponded to the general level of the curve about them—that is, they represented dates when the field production was fairly constant.

3. The number of the new wells drilled between the two dates represented by points *B* and *C* was then determined, this, of course, being a matter of record, and,

4. The number of new wells required to drill the field completely was then estimated from a knowledge of the geology of the field, it being

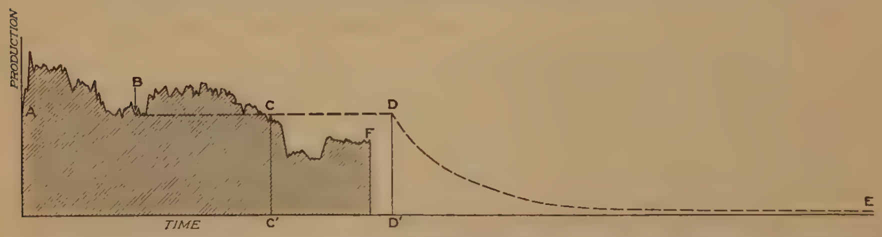

FIG. 4.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF CURVE SHOWING FIELD PRODUCTION IN UNIT TIME ASSUMING THAT DRILLING MAINTAINS PRODUCTION AT CONSTANT FIGURE.

assumed that the manner of locating wells would in the future be about the same as it had been in the past.

These factors having been determined, the supposition was made that, if new wells were drilled after the time represented by the point *C* at the same rate as they had been drilled during the period represented by the line *BC*, the production of the field would be maintained at a constant figure. This is not precisely correct, for the initial production of the wells that are drilled late in the life of the field is not nearly so great as that of the wells drilled while the field is new. The number of new wells necessary to maintain production will therefore increase as the field grows older. An idea as to what this rate of increase is in the number of wells necessary, may be obtained from a study of a cumulative curve showing the time at which new wells have been drilled, and a curve showing the production of the field. The correction may be neglected in making the first approximation of the reserve of petroleum in a field, for other factors which enter into the calculation are liable to so great error that such a refinement seems insignificant, but in making a detailed study of a small area where a considerable number of reliable data are available, such a correction should be made. When the rate has been

determined at which the drilling of new wells must progress in order to keep the output of the field constant, the time at which the field would under these conditions be completely drilled is calculated. This time is represented on the diagram by the point *D*. The quantity of oil which under these conditions would be produced between the time represented by point *C* and the time represented by point *D*—the time at which the field would be completely drilled—may be calculated or may be measured directly from the diagram. After the field is completely drilled a considerable quantity of oil will still remain in the ground and the field will continue to produce at a constantly decreasing rate. The rate of decrease may, however, be estimated fairly well, for it is unaffected by the flush production of new wells.

The most satisfactory method of making this estimate is to choose some small area containing a number of fairly old wells, preferably one

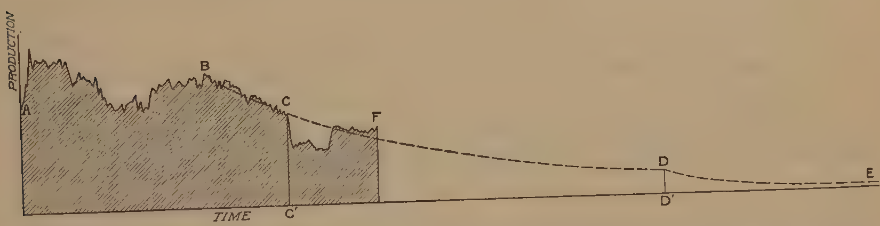

FIG. 5.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF CURVE SHOWING FIELD PRODUCTION IN UNIT TIME, ASSUMING THAT DRILLING MAINTAINS PRODUCTION AT A UNIFORMLY DECREASING RATE.

in which all the wells were drilled at about the same time, and to plot the production per unit of time of this group of wells after the date at which the last well was completed. The resulting curve may be taken as typical of the last period of the life of the average well in the field and it may be applied directly to the curve *ABCD* at the point *D*, which represents the time at which the field is completely drilled.

It may so happen that the field has produced a constantly decreasing quantity of oil since almost the first year of its life and that it is not possible to find two dates far enough apart at which the total production was the same. It may be possible, however, that for a considerable time the decrease in production was fairly constant. If so, a calculation similar to that required in the preceding method may be made, with the difference that drilling is theoretically continued at a rate sufficient to maintain not a constant production but a regular decrease similar to that which takes place during the period chosen. Then, in a manner like that just described, the date at which the field would theoretically be completely drilled may be calculated and the production up to this time estimated. The oil remaining in the field after the completion of drilling

could then, as before, be estimated by assuming that the rate of decrease of production would be like that shown by a typical group of old wells.

An application of this method is shown in Fig. 5. Here the shaded area below the line *ABCF* represents the volume of oil produced to date from the field. On the curve *ABCF* a section *BC* is chosen during which the decrease in production is fairly constant; the number of new wells drilled during this time is ascertained and, on the assumption that this rate of drilling will be continued from the time represented by point *C* until the field is completely drilled, the time at which the field will be completely drilled is calculated. This time is represented on the diagram by point *D*. The quantity of oil that will be produced between the time represented by point *C* and the time at which the field is completely drilled is represented by the area inclosed by the lines *CD*, *DD′*, *D′C′*, *C′C*. The oil that is contained in the field after drilling has been completed is represented by the area lying below the line *DE*.

In this method, as in the preceding one, the assumption that if new wells are drilled after the time represented by the point *C* at the same rate as they had been drilled during the period represented by the line *BC*, production would decrease at the rate at which it decreased during the line *BC*, is not quite correct. Because of the smaller initial production of wells drilled late in the life of the field, the number of wells necessary to maintain this rate of decrease in production of the field will gradually increase as the field is drilled up. A correction similar to that suggested in the method just described may be made here.

Method Using Figures Showing Field Production Expressed in Terms of Production per Well per Unit of Time

During the first few years of the life of a field, drilling is usually so actively carried on that the quantity of oil produced in the field as a whole during a given unit of time is constantly increasing. It is obviously impossible to apply either of the methods just described to this early period in the life of a field, for they are applicable only to a period when the field production per unit of time is decreasing. For many fields, however, a fairly satisfactory estimate may be made by expressing the production of the field per unit of time in terms of the production per well per unit of time, for although the production of oil obtained during a given length of time from the field as a whole is increasing, the production of the average well during equal intervals of time is the same or is decreasing. This method of estimating the petroleum reserves is not so accurate as either of the preceding methods, because it is based upon fewer data. It should not be used, therefore, if either of the other methods is applicable.

Fig. 6 shows the application of this method. The production of the field is represented by the line AB. This curve shows a constantly increasing value for the production of the field during a unit of time, so that the methods represented by Figs. 4 and 5 are not applicable. The production per well per unit of time is therefore determined, and the curve CD, showing these values, is plotted. On the curve CD the points E and F are chosen as representing dates at which the production of the average well—the production per well per unit of time—is the same. The rate at which new wells were drilled during the time EF is ascertained and, as before, the length of time necessary to drill the field completely is

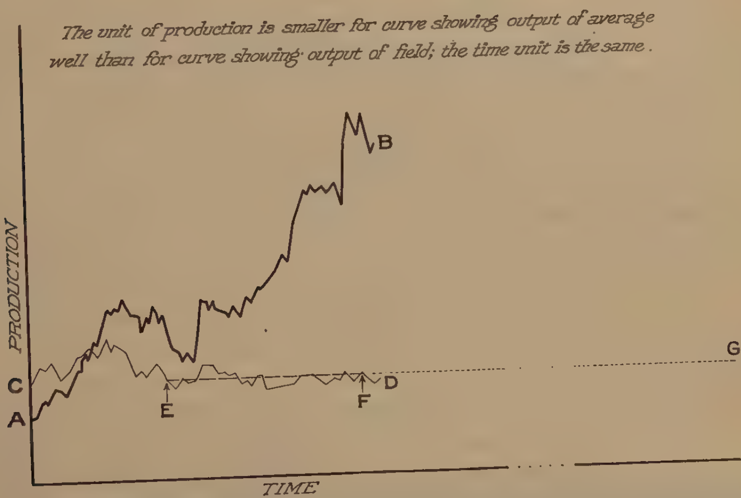

Fig. 6.—Estimation of reserve of petroleum by use of curve showing production per well per unit time assuming that drilling maintains production of average well at constant figure.

determined, it being assumed that drilling will be continued at a rate that will maintain the production of the average well at a constant figure—a figure equal to that of the average well at the dates represented by points E and F. A first approximation of this date at which the field would be completely drilled may be obtained by assuming that production would be maintained at a constant figure if drilling be carried on after the date represented by point F, at the same rate as it had been carried on during the period represented by the line EF. To be accurate, this assumption as to the rate of drilling after the time represented by point F should be corrected to take into account the smaller initial output of the wells that are drilled late in the life of the field. The time at which the field would be completely drilled is represented on the diagram (Fig. 6) by the point G. Then if

a = the number of wells producing at E—the first date,

b = the number of new wells drilled during a unit of time in the period between E and F—the two dates chosen,

c = production of the average well in a unit of time during the period EF,

d = the number of wells necessary to drill the field completely, (This number may be estimated from a knowledge of the geology of the field and of the field practice in spacing wells),

S = the total amount of oil that would be produced between dates E and G.

And assuming that new wells are drilled after the time represented by the point F at the same rate as they had been drilled during the period represented by the line EF, the total quantity of oil produced by the field from the date E until the field has been completely drilled would be given by the formula:

$$S = ac + (a + b)c + (a + 2b)c + (a + 3b)c + \ldots + dc$$

$$= \frac{n}{2}(ac + dc), \text{ where } n = \text{the number of terms in the series.}$$

but:

$$n = \frac{d - a}{b} + 1$$

therefore:

$$S = \frac{d - a + b}{2b}(a + d)c.$$

The total quantity of oil produced before the time represented by E should be a matter of record. The oil remaining in the field after it had been completely drilled under the conditions stated may be estimated, as in the two methods just described, by assuming that the rate of decrease in production will be the same for the field as it is for a small group of old wells.

If the smaller initial production of the wells drilled late in the life of the field is considered and the rate of drilling of new wells corrected to show this factor, the formula given above will not apply. The curve shown in Fig. 6 may, however, be projected in a manner similar to that in which the curve in Fig. 4 was projected. This curve will show the total amount of oil that the average well will produce, from which the total reserve in the field may be calculated.

Should the field production be increasing, but the production of the average well decreasing, the size of the available reserve may be estimated by a method similar to that shown in Fig. 5, but with the difference that the production figures plotted express the production of the average well, and not of the field as a whole.

Probable Degree of Accuracy of Estimates

All the methods of estimating petroleum reserves described above involve the consideration of facts that, in the very nature of things, cannot be determined precisely, and the results obtained, therefore, cannot be precisely accurate. The results obtained by the methods that take into account the greater number of the variable factors are probably the more accurate. The writer believes that the methods he used in estimating the reserves in California express correctly the order of magnitude of the reserve, but he would not claim for them greater accuracy than that.

Discussion

C. W. Washburne, New York, N. Y. (communication to the Secretary*).—We are indebted to Mr. Pack for his detailed description of a rapid method of estimating the oil reserves in large fields. Although his method will give only a rough approximation of the reserves, nevertheless it is much more reliable than the saturation method. The saturation method should not be used when any other method is available.

The paper describes a generalized method, which is especially useful in estimating the total resources of a Nation or State.

Properly, it is a method of estimating the oil that can be extracted by common methods within a given period of time. It does not account for the additional amount of oil which some day will be extracted by new methods, such as water-flooding, the use of compressed gas, etc. These limitations can be overcome only by future investigation.

In valuation work it is customary to estimate the future production in a given number of years, by using decline curves. These may be expressed either in percentage of the average daily production of the first active week or in percentage of mean daily production during the first year.

Requa's method of finding an average decline curve for a typical well (see Mr. Pack's paper, page 974) is the one I have been using in appraising properties. It is the best way to develop a control curve for estimating future production.

In valuing a property, the ideal method is to estimate the future production of each well and to add the total. In recent work involving many hundred wells, I found this impossible within the time available. Accordingly, I devised the following scheme by which the curve may be applied to the entire production of a property.

Having determined a typical decline curve for each field, it was necessary to determine the position on the curve occupied by the present pro-

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* Received Aug. 8, 1917.

duction of a lease. It would be wrong to use the average age of the wells in determining that point, because of the variation in size of wells. The percentage of flush production on a property with one large new well among many old wells, is much larger than would appear from finding the point on the decline curve corresponding to the average age of the wells. It is necessary to find the point corresponding to the average age of a barrel of production. Commonly this is much less than the average age of the wells.

The average age of a barrel of production is determined by tabulating (*a*) the daily production and (*b*) the age of each well, and by finding (*c*) the sum of the products of (*a*) times (*b*) and dividing by the sum of (*b*). It is necessary to consider production from drilling deeper as though it were from a different well. The following table is not an actual case, but it illustrates the method of determining the average age of the production on a property.

Well No.	(*a*) Barrels	(*b*) Age, Months	(*c*) Product of Terms
1	10	18	180
2	12	14	168
2–*b*	25	5	125
3	15	8	120
	—		—
Totals	62		593

The oil 2-*b* is that estimated to result from the deeper drilling of well No. 2, 5 months ago.

Dividing the totals of columns (*c*) and (*a*) we find the average age of a barrel of production to be only 9.6 months, although the average age of the three wells is 13.3 months, and the average age of the four drillings including 2-*b*, is 11.2 months. The first figure, 9.6 months, is the one to be used in finding the point on the decline curve represented by the present production of the property. Having found that point, the curve gives the daily production at future dates from which it is easy to calculate the total production to any required date.

This method saves much time where there are many wells on a property. Where there are few wells, it is best to estimate the future production of each well.

Experience has shown that it is best to separate wells on young properties into two groups, one group including all wells whose age places their production on the steep part of the curve and the second group including all wells that lie on the flatter part of the curve. Calculate the average age of production and the future production of each group separately.

The future production of undrilled locations may be estimated from the same curve by estimating the probable initial production of the locations that are geologically favorable. No others should be considered. If the probable future production is the basis of an appraisal, a large factor of safety must be applied to reduce the estimated production of undrilled locations, because of the uncertainty of geological inferences. The method is not justified except on producing properties.

The publication of more decline curves will be of great assistance to petroleum engineers. I greatly regret that the curves I have collected must be treated as confidential information. If any of the older companies would publish the decline of production on their leases, and if they would keep better records of the individual capacity of wells, it would be a boon to engineering science. Incidentally, it would enable them to make more accurate estimates of the value of a barrel of present production of given age in any field, and also of the value of their own properties.

The Southern Extremity of the "Clinton" Gas Pools in Ohio

BY L. S. PANYITY,* COLUMBUS, O.

(St. Louis Meeting, October, 1917)

THE Cleveland (O.) gas pool described by Frank R. Van Horn,[1] is the northern extremity of the great "Clinton" sand gas development in

General Trend of "Clinton" Sand Pools in Ohio

Richland Township, Vinton County

FIG. 1.

Ohio. Numerous gas pools have been developed in this formation, which are now more or less connected, link by link, forming an almost continuous gas pool, starting at Cleveland, and running thence southward through Cuyahoga, Medina, Wayne, Ashland, Richland, Knox, Licking,

* Geologist, Ohio Fuel Supply Co.

[1] *Trans.* (1917), **56**, 831.

Fairfield, Hocking, and Vinton counties. The present southern extremity of these connected pools is in Richland Township, Vinton County (Fig. 1).

The writer offers his analysis of the Richland Township development which may serve to compare the northern and southern extremities.

Geology

The geologist is greatly handicapped by the lack of a suitable key-horizon, inasmuch as the outcropping rocks are members of the "Big Injun" series of the Logan Group (Mississippian). The underlying shales (Waverly shales) show that considerable erosion took place before the deposition of the "Big Injun," which accounts for the presence of an erosion plane between these two formations, further complicating the study of the geological structure from surface outcrops.

A reliable structural map of the "Clinton" is obtainable only by spirit-level elevations at the mouth of the various wells drilled, and the elevation of the sand calculated from the logs of the wells. The accompanying isobath map has been prepared in this manner (Fig. 2). Since the completion of this map, numerous wells have been drilled in the township, which, if taken into consideration, might cause a slight variation in the appearance of the contour lines. The monotonous monoclinal structure of the "Clinton" sand is evident from this map.

A vertical section of the formations penetrated by the drill may be constructed from the following table:

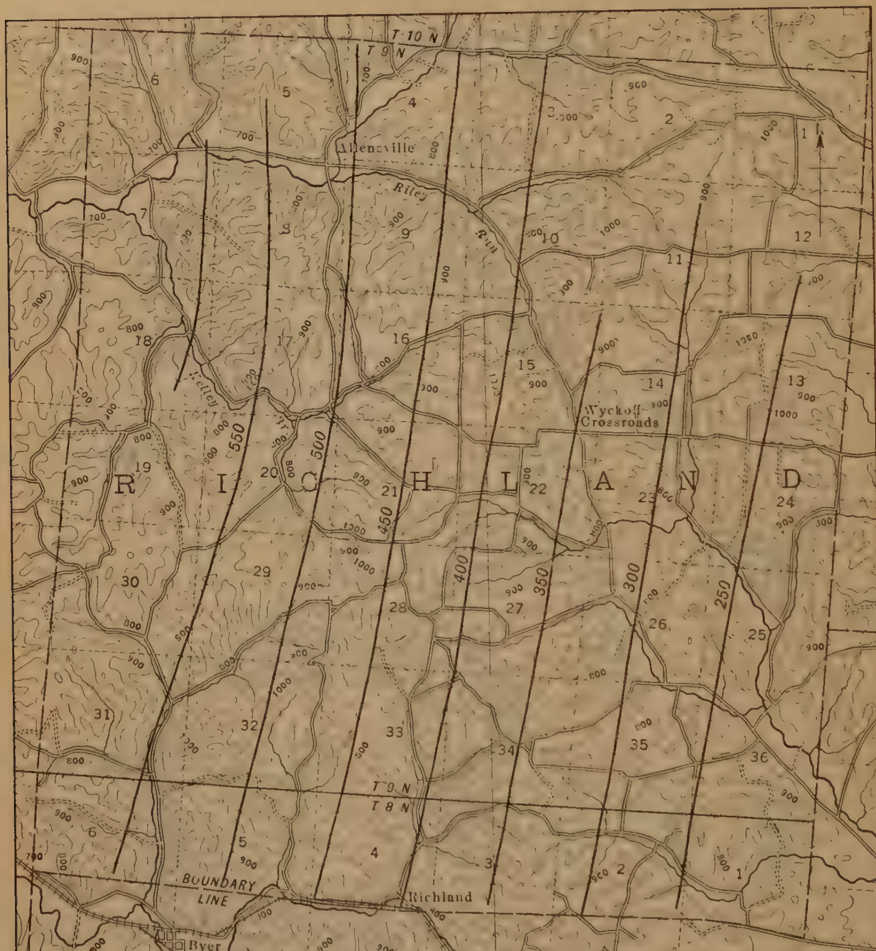
"Big Injun" and Waverly shales	In surface outcrops.
Berea Sandstone	20 to 80 ft. thick
Bedford (Carboniferous) and Ohio (Devonian)	750 to 900 ft. thick
"Big Lime" (Corniferous and Niagara)	570 to 675 ft. thick
Interval "Big Lime" to "Clinton"	175 to 210 ft.
Interval "Clinton" to Trenton	1,350 ft. (estimated).

The thickness of the Berea is variable from place to place without any regularity; but in the other formations cited, the smaller measurements are met with in the western part of the township, the thickening taking place toward the east (Fig. 3).

The Berea lies at an elevation of about 250 ft. above sea level at the western line, and is close to sea level at the eastern line of the township. The average dip for the Berea is about 35 ft. per mile. This compares with an average dip of 80 ft. per mile for the "Clinton" in the same territory. The difference in dip is due to the lateral variation of the intervening strata.

Although the "Clinton" sand development has shown us but few pronounced anticlinals, there remains a great deal of unstudied territory where the pools may be associated with some sort of folding. Work is

now under way in another section of the State, where anticlinal structure is expected.



STRUCTURAL MAP OF THE "CLINTON" SAND

Scale of Miles

0 1 2 3

CONTOUR INTERVAL 100 FEET — DATUM - MEAN SEA LEVEL

STRUCTURE CONTOUR 50 FEET — DATUM - 2000' BELOW SEA LEVEL

Fig. 2.

Oil and Gas

Oil and gas have been struck in various geologic horizons; the first one of which, in descending order, is the Berea. A gas pool is in operation from this formation in the southeast quarter of Richland Township, and

some scattered wells producing gas from the Berea are found south of Richland Township. The Berea contains water in this territory.

Some gas has been encountered in the Ohio shales, and one or two wells produce a little oil from the top of the "Big Lime" (which is probably the Ragland sand of Kentucky), but so far no commercial pool has been developed below the Berea until the "Clinton" is reached.

South of Richland Township but two wells are producing gas from the "Clinton" and they are in the immediate vicinity of the southern township line. Numerous tests have been drilled in a southerly direction as far as the Ohio River, but so far, without encouraging results to the operators.

The gas pools in the township are described by the driller as very "spotted." All observed conditions indicate lenticular sand bodies; and differential cementation has been a great factor in the accumulation

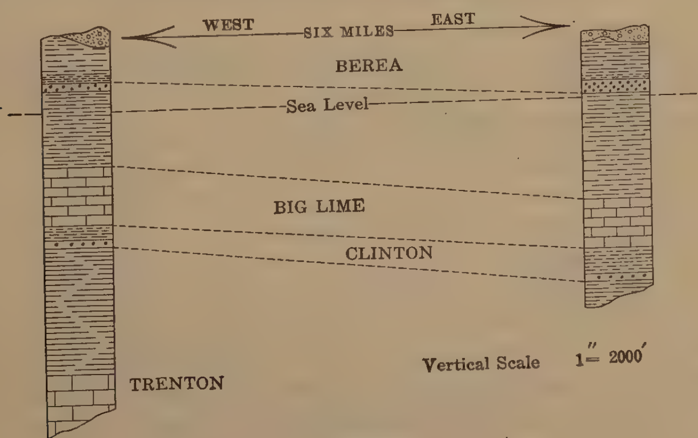


FIG. 3.

of gas along this monoclinal. Another important element under consideration is the fact that connate water is rarely met. The stratum is considered by the driller, free from water. No doubt in further tests in localities where the "Clinton" lies at a greater depth below sea level, connate water will be encountered, when, of course, structural conditions will be of great importance.

In following up "wildcat" wells, tests located along the strike of the formation have been more successful than those located otherwise.

The initial open flow volume of wells is about two to four million cubic feet per day. The largest open flow measurements showed eight to nine million cubic feet of gas per day, when they were drilled in. The rock (or closed) pressure of the wells is between 650 and 750 lb. per square inch.

It is rather early to make any statements regarding the life of these

wells, but no doubt the wells will produce for a greater length of time than did those in Cleveland, inasmuch as the large acreage at the disposal of the operators will enable a more judicious spacing of the wells, and will prevent the recurrence of the overcrowded condition which characterized the Cleveland town-lot excitement.

No producing horizon below the "Clinton" has yet been found. The Trenton limestone, which is the producing horizon of northwestern Ohio, is estimated to be about 1350 ft. below the "Clinton." The depth at which this stratum would be reached in this territory would be about 3500 to 4000 ft. The nearest wells to this locality that have reached the Trenton are at or near the towns of Waverly, Ironton and Oak Hill. In all cases the Trenton was found barren of oil or gas. The chemical analyses of the Trenton in these wells are not known to the writer, but it is doubtful whether the magnesium contents were large enough to create a sufficiently porous rock which might serve as a reservoir for petroleum. In a well drilled through the Trenton into the St. Peter sandstone, north of Mount Vernon (O.) the Trenton was found at a depth of 3645 to 4515 ft. A chemical analysis showed the magnesium carbonate contents of the Trenton in that well as follows:

Depth, Feet	$MgCO_3$ Per Cent.
3,645	3.94
3,718 to 3,723	3.91
3,723 to 3,728	5.19
3,728 to 3,733	4.16
3,733 to 3,742	3.81

When comparing these figures with the figures obtained from the analyses of the Trenton at Lima (O.), where it is a prolific source of oil and gas, and the magnesium carbonate contents range from 25 to 45 per cent., the prospects for a Trenton pool in this territory appear to be very slight.

The present eastern extension of the Trenton oil production is about 2 mi. west of Caledonia, Marion County, where a small producer has been recently drilled and oil found 585 ft. in the Trenton, the few feet of pay sand giving the following chemical analysis: SiO_2, 17.02 per cent.; $CaCO_3$, 46.65 per cent.; $MgCO_3$, 25.75 per cent.; Balance, 10.58 per cent.

This well has created the usual excitement and further work is now contemplated in an attempt to open up a Trenton pool in that locality.

Relation of Sulphur to Variation in the Gravity of California Petroleum*

BY G. SHERBURNE ROGERS,† PH. D., WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

Introduction

One of the features of oil-field work that puzzles operator, chemist, and geologist alike, is variation in the gravity of the petroleum produced on neighboring leases or even from adjoining wells. Few fields in the United States yield oil that is of uniform gravity throughout; in some fields the different grades are produced from different sands, in a few from different formations, and in many from different areas in the same sand. In a few cases, perhaps, a common assumption—that the heavier oil is derived from the lighter through the escape of the more volatile constituents—is valid, but many cases can not be explained in this way. Although in the California fields the specific gravity of the oil generally decreases with depth, there are numerous exceptions to this rule; and, as in many other regions the light oil is found nearest the surface, it is evident that evaporation underground is generally an unimportant factor.

It is generally recognized that the great differences between petroleum from widely separated fields are due largely to differences in age and degree of metamorphism and to variation in the composition of the original organic matter; but in the writer's opinion some of these broader differences, as well as most of the minor variations in the character of the oil from any one field, are due to the local action of natural reagents such as sulphur or oxygen. This is particularly likely to be true if the oil has migrated from one formation to another, as in the California fields; and in some areas it appears that variation in sulphur content may even afford a rough index of the extent or distance of the migration. It is the purpose of this paper to call attention to the chemical action of sulphur and oxygen on petroleum, to show that the gravity of the oil from any one field is intimately related to its sulphur content, and to discuss the possi-

* Published by permission of the Director of the U. S. Geological Survey.

† Associate Geologist, U. S. Geological Survey.

bility that oil in its migration will encounter and take up sulphur or oxygen.

Chemical Action of Sulphur and Oxygen on Petroleum

Petroleum chemists have been occupied so largely in working out the complex constitution of petroleum and the processes of refining it, that we know little of the effect on petroleum of any of the substances with which it comes in contact underground. The study of petroleum from the geochemical standpoint has received slight attention except from a few chemists, and it is therefore impossible to trace in detail the natural changes that oil may undergo. Clifford Richardson,[1] however, pointed out many years ago the influence of sulphur in hardening asphalt. The general action of sulphur and oxygen on oil has long been known to chemists, but the reactions involved have apparently never been exhaustively studied and their geochemical significance does not seem to have been appreciated by geologists.

If paraffin, or a paraffin-bearing oil, is digested with sulphur at moderate temperature it becomes black and heavy and finally passes to a substance resembling solid asphalt. Similarly, if a light asphaltic oil is treated with sulphur it also becomes darker and more viscous, finally becoming asphalt. Under laboratory conditions the reaction is accompanied by the evolution of hydrogen sulphide, and in fact an old laboratory method of generating hydrogen sulphide consists in heating paraffin and sulphur in a retort. The sulphur atom, by extracting 2 atoms of hydrogen from the oil, causes a condensation or polymerization of the hydrocarbon molecule, and this change is reflected in the increase of the gravity of the oil itself as it approaches solid asphalt. A simple example of change through the action of sulphur, from the polymethylene series of the general formula C_nH_{2n}, to the heavier aromatic series of the general formula C_nH_{2n-6}, may be written:[2]

$$\begin{array}{llll} \text{Hexamethylene} & + \text{Sulphur} & = \text{Benzene} & + \text{Hydrogen sulphide} \\ C_6H_{12} & + 3S & = C_6H_6 & + 3H_2S \end{array}$$

This type of reaction is taken advantage of in the Dubb[3] process for manufacturing asphalt commercially by heating crude petroleum or petroleum residue with sulphur. A variety of artificial asphalt known as Pittsburgh flux, made by treating with sulphur the residuum of Pennsylvania petroleum, is described as tough and sticky, and melting only at high temperature. According to Richardson[4] the condensation of one of

[1] C. Richardson: On the Nature and Origin of Asphalt. *Journal of the Society of Chemical Industry* (1898), **17**, 28.

[2] H. Köhler: *Die Chemie und Technologie der natürlichen und künstlichen Asphalte*, 112. Brns., 1904.

[3] English patent 3026, Feb. 16, 1892.

[4] C. Richardson: *Journal of the Society of Chemical Industry* (1897), **16**, 125.

the hydrocarbon molecules involved in this process may be represented as follows:

Diphenylmethane	+ Sulphur	= Tetraphenylethylene	+ Hydrogen sulphide
$2(C_6H_5)_2CH_2$	+ $2S$	= $(C_6H_5)_2C:C(C_6H_5)_2$	+ $2H_2S$

Although the dehydrogenation or condensation of the hydrocarbon molecule caused by the formation of H_2S is the change of greatest interest geologically, under most conditions some of the sulphur enters into combination with certain oil constituents to form simple sulphur compounds or complex sulphur derivatives. Nearly all petroleum contains sulphur, and the nature of some of its simpler compounds has been investigated. In the Lima oil of Ohio, Mabery and Smith[5] found normal sulphides of the paraffin series, having the general formula $C_nH_{2n+2}S$ and ranging from C_2H_6S to $C_{12}H_{26}S$. In Canadian petroleum, Mabery and Quayle[6] isolated another series of sulphur compounds of the general formula $C_nH_{2n}S$. Eight members of this series, which were called thiophanes, were described. According to Köhler,[7] such compounds may be formed by the reaction

$$C_nH_{2n+2} + S_2 = C_nH_{2n}S + H_2S$$

which may be regarded as analogous to ordinary substitution by halogens:

$$CH_4 + Br_2 = CH_3Br + HBr$$

Sulphur is readily soluble in petroleum and some of the Texas oil contains both hydrogen sulphide and free sulphur in solution, in addition to the compounds of sulphur. According to Mabery the proportion of free sulphur in the Humble crude approaches the saturation point, so that sulphur frequently separates out of this oil during transportation.[8] Apparently the solution of sulphur in the heavier hydrocarbons of petroleum is immediate, even at ordinary temperature,[9] and in Peckham's

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[5] C. F. Mabery, and A. W. Smith: The Sulphur Compounds in Ohio Petroleum, *American Chemical Journal* (1891), **13**, 233.

[6] C. F. Mabery, and W. O. Quayle: The Sulphur Compounds and Unsaturated Hydrocarbons in Canadian Petreum,ol *Proceedings of the American Academy of Arts and Sciences* (1905), **41**, 89.

[7] *Op. cit.*, 74.

[8] C. F. Mabery: Relations of the Chemical Composition of Petroleum to Its Genesis and Geologic Occurrence, *Economic Geology* (1916), **11**, 520. Also F. C. Thiele: Free Sulphur in Petroleum from Beaumont, *Journal of the Society of Chemical Industry* (1902), **21**, 1271.

[9] The solubility of sulphur in oil naturally varies according to the series of hydrocarbons involved. Thus, the Edeleanu process for refining petroleum by liquefied sulphur dioxide depends on the fact that the latter is taken up by aromatic hydrocarbons even at very low temperatures, but by the paraffins only at higher temperatures. L. Edeleanu: *Trans.* (1914), **50**, 809–827.

opinion the reaction between sulphur and the hydrocarbons, whereby the latter are rendered heavier and more asphaltic, also proceeds at ordinary temperature.[10] The continuous evolution of large quantities of H_2S from the asphalt lake at Trinidad may be regarded as supporting this statement. Finally, according to Endemann,[11] heavy hydrocarbons react with pyrite just as with free sulphur, half of the sulphur in the pyrite molecule (FeS_2) being given off as H_2S and the rest remaining as a lower sulphide of iron.

Oxygen also exercises a pronounced effect on petroleum and the general character of its action is commonly known. When oil is exposed to the air for a long time it becomes dark, heavy and viscous, and finally passes to asphalt. This change is due chiefly to the evaporation of the more volatile constituents, but partly to oxidation. Thus, if hot air is passed through oil for several hours the oil becomes black and asphaltic.[12] The action of oxygen is similar to that of sulphur; some of the oxygen may be taken into combination with the oil to form complex acids or phenols, and part of it abstracts hydrogen from the hydrocarbon molecule to form water. This reaction is involved in the Byerly process for preparing asphalt by slowly distilling petroleum while passing air through it.[13] Oxygen, like sulphur, may also cause direct polymerization; Hausmann and Pilat[14] note that a naphthene may be transformed into an aromatic hydrocarbon by the action of oxygen at 140° C. and suggest that a similar change may be accomplished underground by some such agent as manganese dioxide.

Oxygen is usually not determined in the analysis of oil and comparatively little is known of the structure of the oxygen compounds. However, naphthenic acids, which are the oxygen derivatives of the naphthenes, having the general formula $C_nH_{2n-2}O_2$, are known to exist in California oils. According to Ostrejko,[15] if Russian oil, similar to the Californian, is exposed to the air, especially in sunlight, these acids form at ordinary temperature, with resulting turbidity and darkening of the oil. Hydroxyl derivatives of the nature of phenols have been found in

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[10] S. F. and H. E. Peckham: On the Sulphur Content of Bitumens, *Journal of the Society of Chemical Industry* (1897), **16**, 996–997.

[11] H. Endemann: *Journal of the Society of Chemical Industry* (1897), **16**, 426.

[12] W. P. Jenney: On the Formation of Solid Oxidized Hydrocarbons Resembling Natural Asphalts by the Action of Air on Refined Petroleum, *American Chemist* (1875), **5**, 359.

[13] C. F. Mabery, and J. H. Byerly: The Artificial Production of Asphalt from Petroleum, *American Chemical Journal* (1896), **18**, 141.

[14] J. Hausmann, and S. Pilat: Studien über die Oxydation der Petrolkohlenwasserstoffe, *Congrès International du Pétrole, Comptes rendus*, Sess. 3 (1907), 378.

[15] R. A. Ostrejko: Influence of Sunlight and Air on Petroleum Products (Abstract), *Journal of the Society of Chemical Industry* (1896), **26**, 345 and 645.

California and other oils,[16] and formic and oxalic acids are reported in petroleum from the Grosny district, Russia, the quantity increasing with the gravity of the oil.[17]

The ease with which a petroleum takes up oxygen naturally varies according to the prevailing series of the hydrocarbons in it. For example, the terpenes, which have the general formula C_nH_{2n-4}, and which are important constituents of some California petroleum, have a well-known tendency to oxidize and to polymerize, and doubtless contribute to the viscosity and resinification of the heavy oils.[18] "However, hydrogen is given up more or less readily by all hydrocarbons in contact with air; even the gasolines soon show color during distillation, and the constituents with higher boiling points more readily become dark and thick. On account of this tendency at high temperatures to lose hydrogen, even the lighter hydrocarbons in varieties of petroleum that contain no asphaltic members are converted during distillation, as ordinarily conducted in the refinery, into the heavier forms that compose the residuums and asphaltic tars. In some crude oils, such for example as the Russian, which is composed largely of the naphthenes, the sudden inflow of air into the hot vapor during distillation in vacuum causes a violent explosion."[19]

Relation of Gravity to Sulphur Content of Oil from California Fields

As shown in the preceding section, there is no question as to the ability of sulphur and oxygen to render petroleum heavier and more asphaltic, although many of the reactions involved are not well understood. If the action of these agents on petroleum in nature has really been widespread, it is reasonable to expect that variation in the gravity of the oil from any one field would be accompanied by variation in the sulphur and oxygen content, as shown by analysis. It is evident, however, that there can be no exact relation, for the particular part of the sulphur or oxygen that contributes chiefly to increasing the gravity of the oil is the part given off and lost as H_2S or H_2O; and the size of this part in relation to that remaining in the oil as sulphur or oxygen compounds is apparently indeterminate. In other words, only the *traces* of the reactive substances can be found by analysis; the portions of those substances that have actually been effective have been lost. Another factor that might militate against an exact relation is the tendency of the sulphur to concentrate in the

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[16] C. F. Mabery: The Composition of American Petroleum, *Journal of the American Chemical Society* (1906), **28**, 426.

[17] N. Schidkoff: Acid Content of Grosny Petroleum and Derivatives (Abstract), *Journal of the Society of Chemical Industry* (1899), **18**, 360.

[18] F. W. Bushong: Chemical Composition of Petroleum, *Kansas Geological Survey* (1908), **9**, 306.

[19] C. F. Mabery: *Economic Geology* (1916), **11**, 518–519.

heavier fractions of the oil, so that if any evaporation had taken place the sulphur content would become abnormally high. Finally, inasmuch as oxygen is not determined in the industrial analysis of petroleum, the data are not available for studying variation in gravity in relation to oxygen content, and as all gravity variation in all oils can not be ascribed to sulphur alone the relation between sulphur and gravity cannot be uniformly exact. In view of these considerations, the relation between the two shown in the accompanying diagrams is surprisingly close.

Figs. 1, 2, and 3 show the relation between the gravity of the oil and the per cent. of sulphur in more than 300 samples from the Coalinga, Midway-Sunset, and Santa Maria fields, California. These fields were selected partly because their product exhibits a considerable range in gravity and sulphur content and should therefore afford a fair test, and partly because of the large number of analyses available. Most of the analyses were made by the United States Bureau of Mines, and all of the published analyses of oil from the fields mentioned that have been made by the Bureau of Mines are plotted in the figures.[20] A few other analyses made in various laboratories have also been included; these are distinguished in the diagrams by circles.[21] According to the method of analysis in the Bureau of Mines the specific gravity of the oil is determined by means of the Westphal balance, or in some cases when the sample is very small by means of the Nicoll picnometer. Sulphur is determined by carefully washing out with distilled water the contents of the Berthelot combustion bomb, after the combustion of the oil, and precipitating with barium chloride the sulphuric acid so dissolved. As the analyses were, of course, not made with the special purpose of showing relationship between gravity and sulphur, it is possible that in some cases these determinations may be slightly inaccurate; but the large number of analyses involved, and the fact that the great majority of them were made according to a uniform method, reduce to a minimum the chance of erroneous deductions.

Fig. 1 represents 171 analyses of oil from the Coalinga field. All of the Coalinga petroleum is low in sulphur, but the relation between sulphur and gravity as shown in the figure is remarkably delicate. The product of the Coalinga field includes two rather distinct varieties of

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[20] The analyses are taken from the following reports. Coalinga field: *U. S. Geological Survey, Bulletin* 398 (1910), 264–272; *U. S. Bureau of Mines, Bulletin* 19 (1912), 20–21; *U. S. Bureau of Mines, Technical Paper* 74 (1914), 34–35.

Midway-Sunset field: *U. S. Bureau of Mines, Bulletin* 19 (1912), 24–27, and *Technical Paper* 74 (1915), 36–37.

Santa-Maria field: *U. S. Bureau of Mines, Technical Paper* 74 (1915), 32–33.

[21] These analyses are cited by D. T. Day: Production of Petroleum, *U. S. Geological Survey, Mineral Resources* (1913), 1126–1139.

Analyses of Santa Maria oil are cited in *U. S. Geological Survey, Bulletin* 322 (1907), 115–118.

petroleum, which are believed to have had their source in different formations.[22] The great bulk of the field's production apparently originated in an Oligocene formation, consisting largely of the remains of diatoms and other organisms, and has since migrated into the overlying Miocene beds in which it is now found. This oil is black, asphaltic in character, carries 0.30 to 0.80 per cent. of sulphur and generally ranges in gravity between

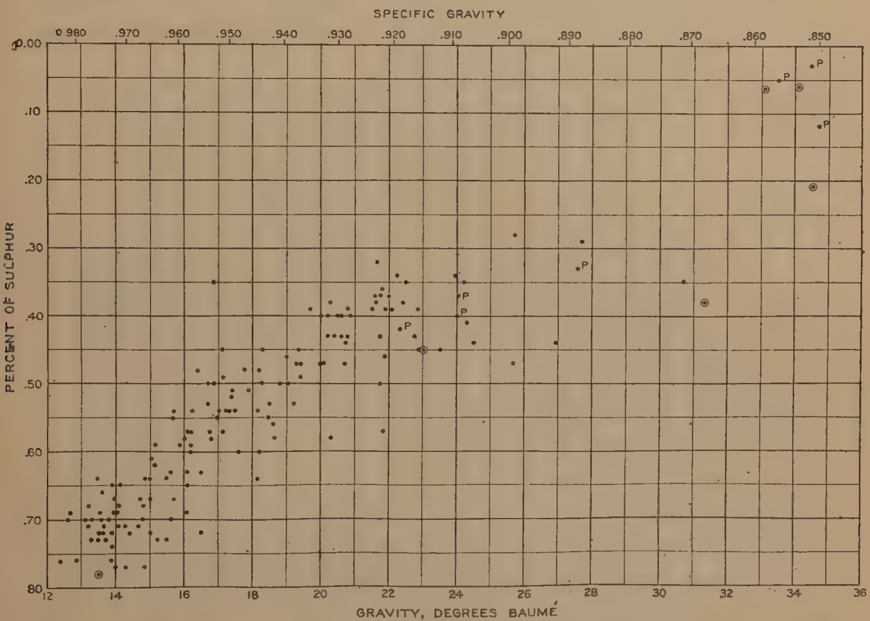

FIG. 1.—RELATION OF GRAVITY TO PER CENT. OF SULPHUR IN 171 SAMPLES OF OIL FROM THE COALINGA FIELD.

Analyses represented by small solid circles made by U. S. Bureau of Mines, the others by various chemists. In analyses marked P paraffin wax was reported.

13° and 30° Bé.; it is represented in Fig. 1 by the great majority of the analyses plotted. The other type of oil apparently originated in another and older diatomaceous shale formation; most of it still remains in that formation, though some has migrated into the overlying Eocene beds. The oil that is in place is greenish in color, contains some paraffin wax, is very low in sulphur and usually approximates 33° Bé. in gravity; it is represented by a few analyses in the upper right-hand corner of the diagram. The oil that has migrated into the Eocene strata also contains

[22] R. Arnold, and R. Anderson: Geology and Oil Resources of the Coalinga District, California. *U. S. Geological Survey, Bulletin* 398 (1910), 182–189.

R. Anderson, and R. W. Pack: Geology and Oil Resources of the West Border of the San Joaquin Valley, North of Coalinga, *U. S. Geological Survey, Bulletin* 603 (1915), 122–162.

paraffin wax, but is much darker in color and distinctly heavier than what is thought to be the same oil in place. The marked difference between the two main types of oil is usually ascribed to difference in age and probably in the character of the organic remains from which they were derived. This view is doubtless partly correct, but it is probable that some of the difference is due to the fact that the light oil is in place, whereas the heavy has migrated into upper formations and in the course of its movement may well have encountered alterative agents.

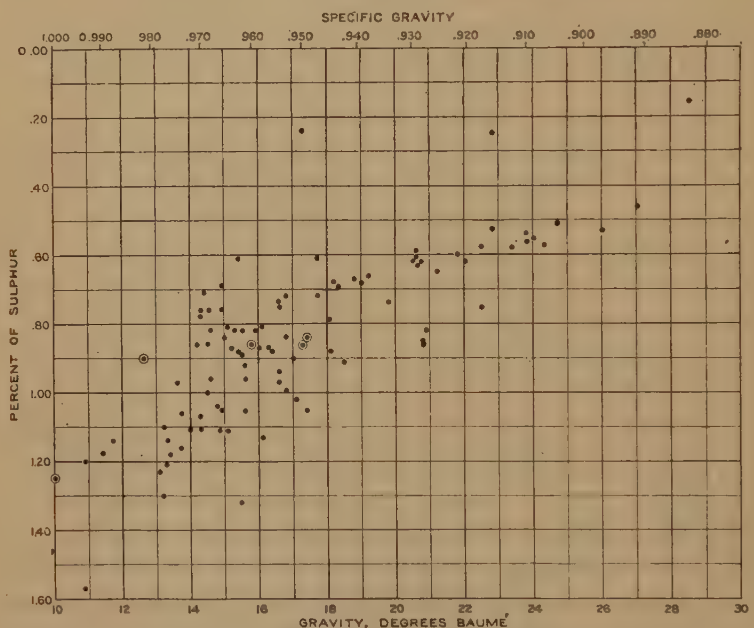

Fig. 2.—Relation of Gravity to Per Cent. of Sulphur in 100 Samples of Oil from the Midway-Sunset Field.

Analyses represented by small solid circles made by U. S. Bureau of Mines, the others by various chemists.

Fig. 2 represents a hundred analyses of oil from the Midway-Sunset field. They exhibit nearly as close a relation between gravity and sulphur as the Coalinga oils, but it is noteworthy that the average per cent. of sulphur is considerably higher. In other words, a typical Coalinga oil carrying a given per cent. of sulphur is considerably heavier than a typical Midway-Sunset oil carrying the same per cent. All of the oil from the Midway-Sunset field is of the same general type, being dark brown to black in color and free from paraffin. It is believed to have originated in diatomaceous shale beds of Miocene age and to have migrated into suitable reservoirs in the overlying formations.[23]

[23] R. Arnold, and H. R. Johnson: The McKittrick-Sunset Oil Region, California, *U. S. Geological Survey, Bulletin* 406 (1910).

Fig. 3 shows 47 analyses of oil from the Santa Maria field. The relation between sulphur and gravity in these oils appears to be less close than in those already discussed, though this may be due in part to the much smaller number of analyses available. The Santa Maria oils are decidedly higher in sulphur than the foregoing, even the lightest oils containing a larger per cent. of sulphur than the heaviest oils in the Coalinga field. The Santa Maria oil is believed to have originated in a diatomaceous shale formation.[24] Some of it occurs in fractured zones of flinty shale in the lower part of the diatomaceous beds and some of it has apparently migrated into sand lenses in the underlying formation.

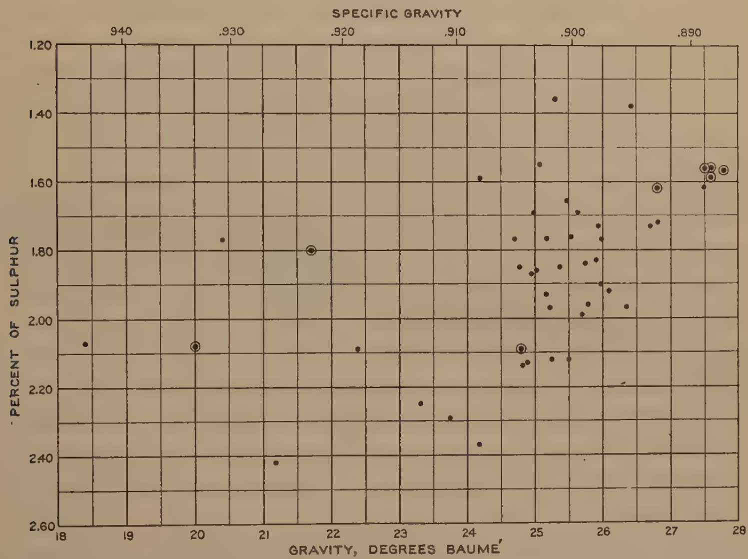

FIG. 3.—RELATION OF GRAVITY TO PER CENT. OF SULPHUR IN 47 SAMPLES OF OIL FROM THE SANTA MARIA FIELD.

These include all analyses published by the U. S. Geological Survey and Bureau of Mines except two, one of which has a gravity of 34.6° B. and 0.60 per cent. of sulphur, and the other a gravity of 15.4° B. and 3.75 per cent. of sulphur.

It will be noted that in all three fields there are a few samples of oil in which the sulphur-gravity relation departs more or less widely from the field average. It is not unreasonable to suppose that these departures represent variation in the unknown factor, viz., the oxygen content. Heavy oils abnormally low in sulphur may owe their gravity to excessive

[24] R. Arnold, and R. Anderson: Geology and Oil Resources of the Santa Maria Oil District, California, *U. S. Geological Survey, Bulletin* 322 (1907).

oxidation, and light oils that seem abnormally high in sulphur may be free from oxygen and affected only by reaction with sulphur. The great majority, which fall along a regular curve, have probably been affected by both oxygen and sulphur. All of the Coalinga and Midway-Sunset oils that depart most widely from the field average seem to occur in the fairly shallow territory within a mile or so of the outcrop of the sand; but whether this has any special significance is not clear.

Possible Sources of Sulphur and Oxygen

That sulphur and oxygen may be important agents in determining the character of petroleum is evident from the wide distribution of these common elements in available form. A few of the possible sources will be briefly considered.

Under ordinary conditions oxygen is probably the less important, though as it is usually not determined in the analysis of oil the extent of its action in nature is conjectural. However, all meteoric waters carry oxygen in solution and it is probable that these waters in places percolate down to the oil measures. Moreover, as Hausmann and Pilat have suggested, oxygen may be contributed locally by oxidizing agents such as manganese dioxide.[25] The most striking evidence of the action of oxygen on California oil is of course seen at the outcrop of the oil sands, where the oil has oozed out on the surface, and, through the evaporation of the lighter constituents and the oxidation of the heavier, has been transformed into asphalt.

There are several possible sources of sulphur, though in the California fields sulphide waters are probably the most important. The surface and shallow groundwaters in the California fields carry large amounts of sodium, calcium, and magnesium sulphate, and outside of the oil fields the deeper waters are also strongly sulphatic in character. The waters in and near the oil measures, however, are almost or quite sulphate-free, and are usually solutions of carbonates and chlorides.[26] Between the sulphate surface waters and the sulphate-free waters associated with the oil, every gradation may be found; and near the horizon at which sulphate begins to decrease and carbonate to increase, the waters usually contain hydrogen sulphide (see Fig. 4). As sulphate is abundant in the shallower waters everywhere along the California coast ranges, whereas sulphide is found only in the oil fields and near the oil and gas, it is reasonable to suppose that the sulphide has been formed through the reduction

[25] In their geochemical studies of the action of silver solutions on ore-forming minerals, Palmer and Bastin point out that certain solutions may act as potent oxidizing agents below the groundwater level. *Trans.* (1913), **45**, 224.

[26] G. S. Rogers: The Chemical Relations of Oil-field in Waters San Joaquin Valley, California, *U. S. Geological Survey, Bulletin* 653 (1917).

of sulphate by the hydrocarbons. The reaction supposed to be involved is usually written:

$$CaSO_4 + CH_4 = CaS + CO_2 + 2H_2O = CaCO_3 + H_2S + H_2O.$$

Although the field evidence in favor of some such reaction is strong, it must be admitted that it has apparently never been experimentally proved.[27] In any event the reaction as written can be considered only as a condensed representation of the type of change that takes place, the intermediate stages in the decomposition of the hydrocarbons on the one hand and of the sulphate on the other being as yet unknown.

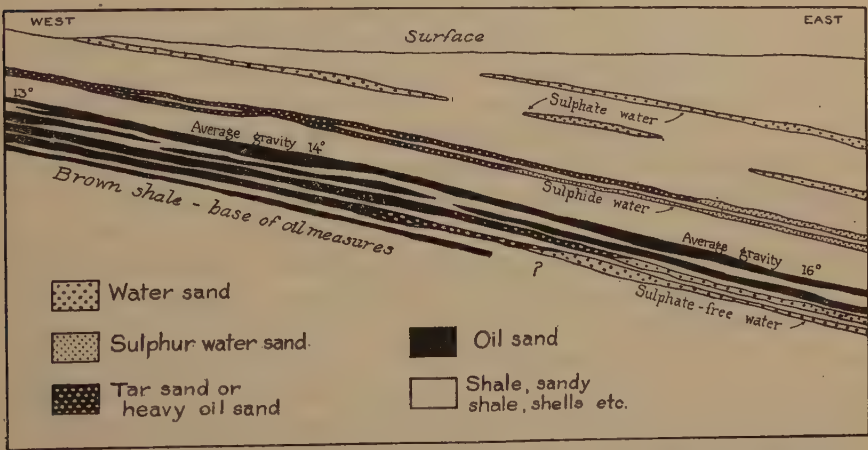

Fig. 4.—Generalized Section through Western Part of Coalinga Field, Showing Replacement of Oil by Water Down the Dip, in the Tar Sand Zone and in the Lower Part of the Producing Oil Zone. The Oil in General Becomes Heavier Toward the Outcrop, but in the Sands Carrying Both Water and Oil a Belt of Heavy or Tarry Oil is Reported to Separate the Two.

In some regions gypsum may be disseminated through the strata near the oil measures and if taken into solution and carried to the oil may be reduced to sulphide. This is, of course, essentially the same as the reduction of sulphate surface waters. Gypsum in the anhydrous condition, however, is a very stable compound, and even with an active reducing agent such as carbon monoxide, a temperature of about 700° C. is required for its reduction.[28]

Whether the reduction of sulphate by hydrocarbons takes place or

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[27] Suggestive experiments are however described by K. Kharitschkoff: The Waters in Petroleum Wells, *Petroleum Review* (1913), **29**, 368.

[28] H. O. Hofman, and W. Mostowitsch: The Reduction of Calcium Sulphate by Carbon Monoxide and Carbon, and the Oxidation of Calcium Sulphide, *Trans.* (1910), **41**, 763.

not, it is certain that many of the oil-field waters carry hydrogen sulphide or alkaline sulphide in amounts ranging up to more than 300 parts per million. The tendency of alkaline sulphide to become free H_2S, and the tendency of this gas to oxidize to sulphur, are well known. In this connection the following personal communication from Clifford Richardson is of interest:

"Some years ago I collected in a sealed tube 200 or 300 c.c. of a natural gas in Trinidad which contained hydrogen sulphide. This was allowed to stand for about 10 years without observation, but at the end of that time it was found that the sulphur of the H_2S was deposited on the walls of the tube in colorless crystals."

The oxidation of hydrogen sulphide proceeds even under very feebly oxidizing conditions, as on the floor of the ocean, and it doubtless takes place even in deeply buried strata.[29] In the light of certain other corroborative evidence,[30] it seems probable that considerable amounts of hydrogen sulphide are oxidized to sulphur, which is precipitated. As the strata directly above the oil measures have not been tested for sulphur, this supposition can not be definitely proved, but it is significant that small deposits of disseminated sulphur are not uncommon along the western edges of the Coalinga and Midway-Sunset fields. Moreover, a commercial deposit of sulphur has been found near the southern end of the Sunset field, in the same formation that contains the oil measures in the field nearby. An interesting feature of this sulphur, to which the writer's attention was first directed by E. A. Starke of the Standard Oil Co., is its intimate mixture with hydrocarbon material, which seems to constitute 20 per cent. or more of the amorphous substance.

It may be added that the waters associated with the oil in many regions are known to be free from sulphate.[31] In many fields there are strong chloride waters which doubtless represent the sea water entrapped in the sediments when they were laid down, and which therefore never contained a very large concentration of sulphate; but in some fields the low chloride and the high carbonate indicate that the waters are in part altered meteoric waters from which considerable quantities of sulphate have been removed. In certain Tertiary and Cretaceous fields in which the normal surface waters are strongly sulphate in character the reduction

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[29] J. Y. Buchanan: On the Occurrence of Sulphur in Marine Muds and Nodules and Its Bearing on Their Mode of Formation, *Royal Society of Edinburgh, Proceedings*, (1890–91), **18**, 17.

F. W. Clarke: The Data of Geochemistry, third edition. *U. S. Geological Survey, Bulletin* 616 (1916), 514.

[30] G. S. Rogers: *Op. cit.*, 100.

[31] C. Engler, and H. Höfer: *Das Erdol*, Band 2 (1909), 28.

A. Potilitzin: *Deutsche chemische Gesellschaft Berichte* (1882), **15**, 3099.

E. H. Pascoe: *Memoirs, Geological Survey of India* (1912), **40**, *pt.* 1, 221.

K. V. Kharitschkoff: *Chemisches centralblatt* (1907), 295.

of the sulphate by the stages outlined above may afford abundant supplies of sulphur to react with the oil. Hence, as meteoric waters carry oxygen and also salts that eventually yield sulphur, it is probable that in many regions waters are the chief agents in the alteration of the oil. The apparent increase in the gravity of oil that has been associated with certain types of water is recognized by many practical oil men (see Fig. 4).

As pyrite is said to react with and yield sulphur to petroleum it is probable that in some localities the action of both pyrite and its less stable isomer, marcasite, have been important. These minerals have been encountered in many wells in the California fields. They probably formed in part during the deposition of the sediments, through the reducing action of organic matter on iron sulphate solutions; but they may also have originated later through the direct action of hydrogen sulphide on chalybeate waters. J. J. Hern, a California oil man of wide experience, told the writer that in wells in which large quantities of iron sulphide are found the oil below is likely to be abnormally warm. If this observation is well founded, it may be significant as indicating chemical reaction between the sulphide and the oil.

In some regions sulphur is doubtless derived from still other sources. Hydrogen sulphide and sulphur dioxide are common components of volcanic emanations, and the former is found in many thermal springs that are supposed to represent the last stages of igneous activity. Much of the Mexican oil, which is heavy, asphaltic, and high in sulphur, is found near igneous intrusions and may well have been affected by the sulphurous gases that doubtless accompanied them. The oil in the salt domes of the Gulf Coast is somewhat similar in character and has evidently been altered by sulphur, but the origin of the sulphur in this case is related to that of the salt domes themselves, and has never been satisfactorily explained. Again, it has long been known that certain bacteria have the property of generating hydrogen sulphide through the reduction of sulphate solutions. Some of these bacteria are anaerobic, being able to exist in the absence of air, and their action has been repeatedly observed in ocean water;[32] but whether they can exist and function in deeply

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[32] A. Van Delden: Beitrag zur Kenntnis der Sulfatreduktion durch Bakterien, *Centralblatt Bakteriologie*, Band 11, Abt. 2 (1903), 92–94, 113–119.

A. Lebedinzeff: Vorläufige Mitteilung über den chemischen Untersuchungen des Schwarzen und Asowischen Meeres in Sommer, 1891, *Soc. Naturalistes à Odessa Trav.* (1891), **16**, fasc. 2, 149. Abstract in *Royal Geographical Society Proceedings*, New ser. (1892), **14**, 461.

N. Zelinsky: Sulphydric Fermentation in the Black Sea, *Russian Chemical Society Journal* (1894), **25**, 298–303. Abstract in *Chemical Society Journal* (1894), **66**, pt. 2, 200.

N. Andrussoff: Physical Exploration in the Black Sea, *Royal Geographical Society Geographical Journal* (1893), **1**, 49.

buried strata is open to question. Finally, there are oils, like those of the Appalachian fields, that have apparently never been subjected to the action of sulphur; and others, like the Trenton limestone oil of Ohio, that are generally supposed to owe their sulphur to the character of the organic remains from which they were formed.

Sulphur and Gravity in Relation to the Migration of the Oil

In the preceding pages the writer has endeavored to show that the alteration of petroleum by sulphur and oxygen in nature is chemically and geologically possible, and the specific field evidence tending to show that such alteration has actually taken place may now be considered. Arnold and Anderson, in their exhaustive report on the Coalinga district,[33] make the significant statement that two of the important factors influencing the gravity of the petroleum are the extent to which it is, or has been, associated with water, and the extent to which it has migrated, either upward, or along a particular bed, or downward. In the present writer's opinion, the increase in gravity incident to association with water and to migration is due not to the action of H_2O as such nor to mere movement, but rather to the salts and gases carried by the H_2O and to the reagents that the oil encountered in the course of its movement.

It is evident that in any one field the action of both oxygen and sulphur on the oil will be more or less localized, since ordinarily the quantity of these substances available is insignificant as compared with that of the petroleum. The free chemical energy of reacting substances and the time of contact are potent factors in chemical change, and should be recognized in accounting for variations in the properties of oil that has migrated. An oil that is the first to traverse a given course comes into contact with reacting substances at their highest potential, and therefore becomes changed more radically than does the oil that follows it at the same rate. As soon as these reacting substances have become exhausted, then oil may pass them unaffected. Again, oil that moves with extreme slowness and remains a very long time in contact with reacting substances may undergo changes just as marked as though it had moved more rapidly in a new channel. In general, therefore, that portion of the oil which migrated first or farthest will be the most altered. Moreover, the oil nearest the surface or nearest the outcrop of the oil-bearing zone may be further altered by fresh supplies of descending sulphate waters. Therefore, the oil around the edges—particularly the upper edges—of the main body should as a rule be the most altered or, in other words, the heaviest and most asphaltic.

This reasoning is well borne out by the variations in the gravity of the oil in the Midway-Sunset and Coalinga fields. In the deeper portion of

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[33] *U. S. Geological Survey, Bulletin* 398 (1910), 186.

the Midway-Sunset field, for example, the specific gravity of the oil ranges between 20° and 30° Bé., but as the outcrop is approached the oil becomes heavier and most of the wells nearest the outcrop produce oil of about 12° to 14° Bé. gravity. A part of this difference may be due to the escape of the more volatile constituents of the oil in the zone along the outcrop, but variation in gravity several miles away from the outcrop can hardly be explained in this way. As shown in Fig. 4, conditions in the Coalinga field are similar, the average gravity increasing from about 16° to about 13° Bé. in less than 2 miles. Certain producing oil sands in the lower part of the oil measures become water-bearing down the dip, however, and it is significant that the oil and water in these sands are apparently separated by a belt of tar or abnormally heavy oil. In the western portions of the Coalinga, Midway, and Sunset fields, the producing oil sands are overlain at a distance of several hundred feet by the tar sand zone, which contains sands partly impregnated with heavy, viscous tar. Some of the sands in this zone carry sulphur water and most of the tar sands become water-bearing farther down the dip. So far as the writer knows, this very heavy tar has never been analyzed, but the analyses of several samples of oil of 11° Bé. gravity show about 1.15 per cent. of sulphur, and it may be presumed that the tar carries at least as much as this. In a general way the tar sand zone marks the farthest limit of migration of the oil. It has been observed, furthermore, that oil which has migrated for some distance into sands that lie in angular unconformity with the main oil zone generally becomes heavier with distance from the main body and finally passes to tar. Finally, the marked difference in character between the light Coalinga oil that has migrated only a short distance, if at all, and the heavy Coalinga oil that is believed to have migrated from one formation to another, may be due in part at least to the alteration of the heavy oil in the course of its movement. Purely on the basis of field evidence, therefore, it would appear that among the important factors influencing the gravity of the oil are the distance that the oil has migrated and the extent to which it is or has been subject to contact with waters, especially meteoric (sulphate) waters.

There seems little doubt, therefore, that the local variations in the character of the oil are in part, at least, due to the action of sulphur and probably also of oxygen. Sulphate waters descending from the surface are to a large extent altered in the zone of tar sands, and the tar itself is thereby rendered still more asphaltic. To some extent, therefore, the tar sands may be conceived as protecting the main body of oil in the sands below. The same process goes on near the outcrop, although how far descending meteoric waters have affected the main body of the oil since it attained its present position is a matter of conjecture. It seems more reasonable to suppose that the oil which migrated first and farthest was

considerably altered by the water that had previously occupied the sands, and that most of its alteration took place before it had come to rest in its present position.

Bearing of the Reactions Outlined on the Migration and Origin of Oil in General

Although the action of sulphur and oxygen thus seems to have been important in determining the local character of California oils, the writer realizes that conditions in the Californian fields are exceptional. There are few regions in which commercial accumulations of petroleum occur so near the outcrop of the containing sands and in which, therefore, the oil is so likely to be affected by meteoric waters. Carbonate waters similar to those of the California oil fields are described from the Russian, Galician, and Burma fields and have doubtless played a similar rôle in the alteration of the petroleum in those areas. In most regions, however, meteoric waters do not appear to have had access to the oil measures, and the oil-field waters are strongly salty. The very fact that the oil itself can accumulate and remain within a small area under high pressure for long periods of time ordinarily implies that the underground circulation is decidedly restricted. Hence, if the beds were originally laid down in salt water it is reasonable to expect that this water under most conditions would be retained with the oil, and other water excluded; and as sea water is relatively low in sulphate its action on the oil would not be very pronounced. It is only when oil is brought into contact with a type of water carrying reactive agents—either through the migration of the oil into a formation saturated with such water, or through the percolation of the water to the oil zone—that the action of water on oil is likely to be localized and therefore apparent.

There is a tendency among petroleum geologists to assume that many oil pools represent the accumulation of the oil originally disseminated over a wide area. The enormous deposits in some small districts are indeed difficult to explain except as due to concentration. If the supposed course of the oil is along one sand, or laterally in one formation of reasonably homogeneous character, the hypothesis may be acceptable; but it should be recognized that if the oil has traversed beds of diverse texture and containing reactive substances it is very likely to have undergone recognizable change. Contact with sulphur and oxygen may result in the loss of hydrogen and the formation of unsaturated and asphaltic compounds; subsequent passage through shale may result in the fractionation of such oil by diffusion, and the removal of the heavy fractions, as suggested by the experiments of Day.[34] Hypotheses involving the migra-

[34] D. T. Day, J. E. Gilpin, and M. P. Cram: The Fractionation of Crude Petroleum by Capillary Diffusion, *U. S. Geological Survey, Bulletin* 365 (1908).

J. E. Gilpin, and O. E. Bransky: The Diffusion of Crude Petroleum through Fullers Earth, *U. S. Geological Survey, Bulletin* 475 (1911).

tion of apparently unchanged oil over long distances should be accepted with caution, for petroleum is one of the most complex and delicate of natural substances.

The reactions outlined have a bearing also on theories of the origin of petroleum. Thus, they appear to furnish further evidence against the inorganic hypothesis, for in view of the pronounced changes induced in the California oils by a relatively short migration it is difficult to understand how petroleum could have ascended from the interior of the earth. Some of the Mexican oils, which are unusually heavy and high in sulphur, have been cited, because of their apparent connection with igneous intrusions, as evidence in favor of inorganic origin, but their peculiar character is simply what would be expected in any oil that has been locally exposed to the action of gaseous emanations, such as hydrogen sulphide.

If the organic origin of petroleum is accepted, it is evident that the old idea, that sulphur in petroleum indicates derivation from animal remains, is not necessarily valid, for any oil may take up sulphur from extraneous sources. On the other hand, there is no reason to suppose, since contact with sulphur increases the gravity of oil, that the earlier stages in the evolution of petroleum are represented by the light sulphur-free oils and the older or higher ranks by the heavier oils. David White[35] has shown that the character of petroleum depends largely on its degree of dynamic alteration, the light oils being in general the oldest and most altered and the heavy oils the youngest and least altered. This is indicated by the fact that the gravity of oil, regionally considered, varies inversely with the degree of metamorphism of the coal in neighboring strata. It is generally recognized, furthermore, that the character of the original organic material has a bearing on the composition of the oil derived from it, though the precise relation has not yet been ascertained. Day's work has indicated that another important factor is probably the character of the rock—whether coarse or fine, loose or indurated—in which the petroleum is found or through which it has moved. These are generally the broad and fundamental factors; the action of sulphur, oxygen, or other agents is usually local and in a measure incidental, though it must none the less be taken into account.

Discussion

C. W. Washburne, New York, N. Y. (communication to the Secretary*).—It has long been known that sulphur and oxygen react upon crude oils, removing hydrogen and thereby creating unsaturated hydrocarbons which unite with each other, producing heavier forms. Dr. Rogers now

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[35] David White: Some Relations in Origin between Coal and Petroleum. *Washington Academy of Science Journal* (March 19, 1915), **5**, No. 6, 189–212.

* Received Aug. 8, 1917.

finds that the percentage of sulphur in the oils of several fields in California is roughly proportional to the specific gravity of the oil. This additional proof is most welcome. It is rather surprising, in that the impression had prevailed that practically all the reacting sulphur escaped from the oil as hydrogen sulphide, and did not remain in combination with the oil. It would be interesting to know what part of the sulphur plotted by Dr. Rogers is in solution as free sulphur and hydrogen sulphide, and what part is in more complex compounds, such as thiophanes, etc.

Dr. Rogers considers sulphur more important than oxygen in making crude oils heavier. This is debatable. If the main source of the sulphur lies in the reduction of sulphates, it is evident that for each atom of sulphur there are four atoms of oxygen to work on oil. The complete absence of sulphates from oil-field waters indicates their complete reduction by oil. *Vice versa,* it indicates that the dissolved sulphates oxidize oil until they are destroyed.

Stagnant oil resting in oil pools is not greatly oxidized except at its junction with the basal water. I am glad to confirm Dr. Rogers' observation on this point. On the edges of the Healdton Field, Okla., and in other places, I have noted the prevalence of heavier oil next to the marginal water.

In migration, however, oil doubtless comes into contact with much water. Since the volume of altered sea water in the rocks is several thousand times the volume of the oil, it is evident that, during migration, the oil may have come in contact with a great quantity of sulphates.

Red formations always contain comparatively heavy oil. This is true of red formations of our Western States, of Argentina, of Egypt, and of the Congo, the only places where I know of oil in red rocks. The red pigment, the gypsum, which usually is present, and all active compounds that can exist in red formations are necessarily oxidizers of crude oil. The rock is turned green in contact with the oil, proving the reduction of the iron oxides in the rock and the oxidation of the oil. Gypsum, so far as I know, is associated only with heavy oil. In general, there is reason to believe that oxygen exerts a very important rôle in increasing the specific gravity of crude oils.

Oil, being the best reducing agent in Nature, always is in an oxidizing environment. Oxidation is especially effective during migration. The action of oxygen and sulphur on oils, making them heavier, is, in my opinion, the principal geological alteration of crude oil. I believe, in general, that exothermic processes must prevail in nature and that crude oil must get heavier with the passage of time. Its molecules cannot split into lighter forms, except under unusual temperatures, such as may prevail during igneous intrusion. It is probable that all crude oils are descended from lighter parent oils. Under any plausible theory of origin, it is probable that crude oils were very light liquids when first formed. Dr.

Rogers calls attention to one of the important agents that helped to alter the original light oils into the crude oils we find today.

Clifford Richardson, New York, N. Y. (communication to the Secretary*).—The contention advanced by the author that there is a variation in the specific gravity of certain petroleums bearing a close relation to the percentage of sulphur which they contain, would seem to be a natural inference since the action of sulphur on hydrocarbons results in condensation with elimination of hydrogen as hydrogen sulphide. Naturally the greater the percentage of sulphur the greater the possibility of condensation which can or has been accomplished and the greater the density of the oil.

The sulphur present in petroleum may be derived from various sources. In the well-known Beaumont, Texas, oil it was recognized as occurring, at least partly, in a free state and could be crystallized therefrom by passing the petroleum through fuller's earth, thus removing the heaviest hydrocarbons and their derivatives from the oil, and leaving only those in which sulphur is less soluble.

In other petroleums sulphur must exist as derivatives of the hydrocarbons composing the oil.

It is a question, which cannot be definitely settled, as to how far the sulphur in petroleum in any particular instance may be derived from hydrogen sulphide existing as such in the natural gas in which the oil finds its origin, and how far from sulphur derived from sources external to those in which the natural gas or oil has originated. It has been suggested that the origin of the sulphur derivatives of the hydrocarbons in petroleum is to be attributed to the reduction of sulphates of thermal origin by organic matter. A further possibility lies in the fact that in the condensation of natural gas to liquid hydrocarbons, due to the relation of films of the gas to surfaces of solids, hydrogen would be eliminated in an active state and might reduce sulphates from some external source. The rôle which the relations of surface and films may bear to the origin of petroleum is an important one, and has been considered by the writer in a paper on The Nature and Origin of Petroleum and Asphalt.[1] In any case, it seems to the writer that there is no necessity for advancing the theories which have been proposed for either the origin of petroleum or the presence of sulphur therein. The percentage of sulphur in petroleums, as is well known, varies to a great extent. One per cent. was considered a large amount in the petroleums known prior to the development of the Mexican and Trinidad fields. As these oils, however, carry over 2 per cent. of sulphur they represent a new type. They are also differentiated among themselves by the fact that the sulphur is in a

* Received, Aug. 30, 1917.

[1] *Metallurgical and Chemical Engineering* (Jan. 1, 1917), **16,** No. 1.

different state of combination in the two oils. In the Trinidad sulphur derivatives are found in the lighter distillates, while in the Mexican they remain, to a large extent, in the residuals, pointing to the fact that the manner in which sulphur occurs in petroleum is variable.

In view of the facts that have been stated, it seems impossible for the writer to follow entirely the conclusions of the author of the paper, or of those who have looked upon petroleum as being of animal or vegetable origin. The great simplicity of the theory of its origin in the relations of surfaces of solids and films of gas and the subsequent changes in the petroleum under the environment to which it is subjected in nature, recommends itself highly and does away with much that has been hitherto inexplicable. The theory of the writer has been elaborated in the article previously referred to and will be given further consideration in a paper which is in preparation.

Charles F. Mabery, Cleveland, O. (communication to the Secretary*).—This interesting paper recalls a conversation with Dr. Rogers, two years ago, in connection with some experiments I then had in progress and still unfinished on the action of sulphur on the individual hydrocarbons. I was then much interested in the outline of prospective study that Dr. Rogers had in mind during his visit to the California oil fields and am greatly pleased to learn of his valuable results. The California fields are doubtless especially adapted for such a comparison of specific gravity with sulphur content, a relation that appears generally in other localities, but with some exceptions. For instance, oils of the type of the long-known Meccal oil, and the Mahone oil, both Ohio oils, and both heavy, specific gravity, 0.87, contain no sulphur. The oils are excellent examples of the view advanced by David White, and accepted by the author, that the nature of petroleum depends to a considerable extent on the degree of dynamic alteration and on the degree of metamorphism of the neighboring coal. It is evidently not possible to draw a sharp line of demarcation with reference to specific gravity and sulphur content, since such oils as those of the Trenton limestone in Ohio are not high in specific gravity, but all carry sulphur. In some localities there can be no doubt that sulphur has had much to do with changing the nature of the crude oil from its original condition, since such an action is a direct result of contact of crude oil with sulphur. The best example of such a change is found in the Texas and Louisiana fields where such contact is direct with solution of sulphur and evolution of hydrogen sulphide in large amounts. But as explained in this paper, deposits of sulphur are not always necessary to account for its influence. Under reducing conditions sulphate waters may yield sulphur, and such strongly reactive conditions always exist near the petroleum areas. The odor of

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* Received Aug. 28, 1917.

hydrogen sulphide frequently observed may have its origin directly or indirectly from this source.

As is well stated in this paper, it may be said in general, that by reason of the volatility and instability of the hydrocarbons of which petroleum is composed, the potent influences in the formation of many of the heavier varieties were evaporation and the chemical action of oxygen and sulphur. All but the heaviest constituents of petroleum evaporate rapidly on exposure to the atmosphere. Hydrocarbons as heavy as $C_{18}H_{38}$ evaporate completely in corked containers. I have lost hydrocarbons on standing a year or more by evaporation through cork. Some years ago, I placed a large amount of a light Pennsylvania crude in a draft; in a month it lost all of its volatile constituents, leaving a thick mass of paraffine. The best example of the influence of oxygen on hydrocarbons that has come under my observation is described in a paper soon to appear in the *Journal of the American Chemical Society.* Some unsaturated hydrocarbons freshly distilled *in vacuo*, colorless, in corked containers soon began to show color on the surface, and the color gradually extended downward through the entire mass of the oil. The uncolored oils contained no oxygen, but after coloration the same oils showed on analysis a considerable percentage. There can be no question that such influences as those mentioned above have been active in the formation of the solid bitumens such as gilsonite. The atmosphere is extremely pervasive through even semi-permeable obstructions, and no doubt it finds way to petroleum deposits through not too dense shale covers.

Any study of petroleum as to its origin, present storage or condition, manipulation in pumping or refining, cannot avoid reference to its components, and it is greatly to be regretted that so little is known of its ultimate composition. Laboratory study has reached its limit in the accumulation of fundamental data. Investigations based on the manipulation of larger amounts of crude oil—ton lots—with adequate funds and expert service for the complete separation of its constituents should yield results of immense value to science and to the petroleum industry.

The Practical Value of Oil and Gas Bureaus

BY W. G. MATTESON,* B. S., E. M., E. MET., HOUSTON, TEX.

(St. Louis Meeting, October, 1917)

THE Oklahoma legislature recently passed a bill providing for "the creation of an oil and gas department under the jurisdiction of the Corporation Commission, authorizing the Corporation Commission to appoint a chief oil and gas conservation agent, and conferring exclusive jurisdiction on the Corporation Commission in reference to the conservation of oil and gas, and the inspection of gasoline and oil, the product of crude petroleum, and repealing all acts or parts of acts in conflict therewith and declaring an emergency." Some of the functions of the new bureau include the receiving and filing of all well logs, and the direction and supervision of plugging all abandoned oil and gas wells, under rules prescribed by the Corporation Commission.

The two leading petroleum-producing States of the country, Oklahoma and California, have now established oil and gas bureaus with a staff of inspectors to see that the important rules relating to the conservation of oil and gas are thoroughly enforced. The economic significance and the value of well logs in the conservation work has been recognized, and the necessary legal action taken to insure and procure them. William B. Heroy, of the Government Land Classification Board, in a personal communication to the writer, says that the drilling of wells is also regulated by public authority in the Roswell artesian basin in southeastern New Mexico, where a permit must be obtained for the drilling of any well and a certified log of the well recorded with a county official. The method is said to give fairly satisfactory results.

In a recent article[1] a National instead of a State Bureau was suggested for collecting and filing well logs. In discussing the paper, L. L. Hutchison characterized such a plan as impractical because the average driller or contractor will often juggle or guess at the log and will not keep the record sufficiently close or accurate, making it necessary to maintain a special man at each well for the sole purpose of obtaining

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[1] W. G. Matteson: The Need and Advantages of a National Bureau of Well-log Statistics, *Trans.* (1917), **56**, 881.

such data correctly, a procedure that is expensive. A number of geologists and producers will take issue with this contention. Moreover, the excellent logs of every well in California, obtained at little or no expense, form the best refutation and answer to such an objection. It should be further borne in mind, as suggested previously, that a log need not be accurate in a strict scientific sense, to be of great practical value when projected on the scale of 1 in. = 100 ft. Where a number of logs are obtained from the same field, any glaring inaccuracies resulting from an improper record are generally detected when checked against the average of several adjacent logs. Such inaccurate logs can therefore be rejected.

To emphasize further the practical value of well logs from a geological standpoint, the writer knows of a company that has saved and made several millions of dollars in one pool by the judicious application of such data. This company industriously gathered well records for a considerable period and projected them on a scale of 1 in. = 100 ft. In this way, the structure of the pool was completely and quite accurately defined, the purchase of much high-priced but poor acreage avoided, while acreage that was considered questionable was shown to contain some of the best wells in the region. All the large producing oil companies have recognized the practical importance of well logs, and many now maintain special subsurface departments whose chief functions are the recording and plotting of such data.

As for the difficulty of getting the head driller or contractor to keep an accurate record of the well, the writer has questioned a number of operators who say that such difficulty is greatly exaggerated. The driller generally shows a tendency to coöperate in every way and to obtain that which is requested. In some regions, it might be necessary to educate the driller to the point of recognizing the value of accurate logs. In this connection, it is gratifying to note the vast improvement during the last 2 to 4 years in this matter. Where there were 10 accurate records 5 years ago, there are a thousand today. The "oil smeller" or geologist is no longer viewed with suspicion or contempt by the field men; he has come into his own. If such results can be secured under very unfavorable conditions, there seems to be no logical reason why a more determined and organized effort, supported by public authority and directed by a standard and efficient bureau, should not produce the results desired. Aside from the geological importance, if water troubles are to be prevented, if desirable and satisfactory data are to be obtained on the life and rate of decline of wells, if accurate estimates of oil reserves are to be obtained, well logs are vital and an efficient oil and gas bureau becomes a practical necessity.

Despite the advantages that a National Bureau of Well-log Statistics, or a National Oil and Gas Bureau, might possess over a State organization of similar design, the establishment of such a bureau would constitute

undoubtedly an infringement of State rights.[2] Herein lies the main obstacle to the plan for a National organization and there seems to be no way at present to overcome it. The matter, therefore, must await the initiative of each State, and it is to be hoped that the importance of such an organization will be so recognized that the bureau will not be made ineffective by the appointment, for political reasons, of incompetent and inexperienced officials.

The new Oklahoma Oil and Gas Bureau has unlimited possibilities, since its powers are not expressly defined by any definite and prescribed statute, as they are in California. The Oklahoma bureau is under the direct jurisdiction and supervision of the Corporation Commission, a body which has the power to hear and to sit in judgment on any case or complaint pertaining to oil and gas and their conservation, and to prescribe and cause to be enforced any rule or provision which it deems necessary to remedy an undesirable condition. The development of this bureau will therefore be studied with interest. Meanwhile it would be gratifying to see similar State bureaus established in Texas, Louisiana, Pennsylvania, West Virginia, Ohio, Illinois, Wyoming, Kentucky, and all other States where extensive drilling operations are now in progress.

Discussion

E. G. Woodruff, Houston, Tex. (communication to the Secretary*). —Mr. Matteson has presented to us an idea worthy of our careful consideration. We have real troubles in the petroleum industry and as the supply of crude is exhausted our problems become more complicated and more numerous. Therefore, we who are engaged in field work are forced to consider all possible means which promise assistance in the location of prospective pools, and the economic development of pools already located. In our country we are accustomed to turn to the Government for relief in all our troubles. Do we want an Oil and Gas Bureau doctoring us? Will such a bureau do us practical good?

The principal point Mr. Matteson suggests is that such a bureau should be authorized to receive and file all well logs. In analyzing such function, I am led to the following conclusions:

If well logs are simply to be filed they would be useless. If they are to be filed and the files opened to the public, what benefit will be derived and can such action be taken justly? There is no doubt that if the record of all wells drilled in the United States were available the petroleum industry would be saved an immense amount of needless expense. Accumulating conditions would be better understood and prospecting more intelligently directed.

[2] M. L. Requa: Discussion of the Need and Advantages of a National Bureau of Well-log Statistics, *Trans.* (1917), **56**, 884.

* Received July 24, 1917.

Our other consideration is: can the oil companies be required to open well records to the public justly? I think all will agree that if any one should have the advantage of a well record it should be the company that drilled the well. If the oil companies are forced to file such record and open the files to the public, the company drilling the well cannot derive the exclusive benefit from the money expended. Would it be just for a small company to be compelled to reveal its records of a well it had recently completed, to its strong competitor who holds near or even adjacent leases? Would it have been just to the successful companies in the Gulf Coast country to have been forced to display their hard-earned knowledge of 15 years to the host of younger competitors who have recently flocked to the Gulf Coast, attracted by the success of those who have labored so long, and learned of the conditions controlling the fields, through the hazards of the industry?

The men of these newer companies are just as intelligent as the representatives of the old companies. The chief protection for their hard-earned advantage to these old companies is their secrecy about their operations. If there is a Federal record bureau and its data are open to the public, such protection is withdrawn. For the fact is, many records are given out and exchanged. This practice should continue, but the owner of the record should be the judge of the advisability of giving out the record. Please understand that I am not opposed to well records, I am heartily in favor of them, but I feel that it would be unjust to force companies to deliver them to the public.

We want our competitor's information, but he is forced to retain it as confidential because we do want it to take advantage of the information he has obtained.

The action proposed by Mr. Matteson might be defended for two reasons, namely, for scientific study and for the complete and economic development of the petroleum resources of the country. I believe that the scientific purpose is largely accomplished already by the companies placing their records at the disposal of Government scientists.

All will agree that certain treatment of wells should be required to conserve and properly develop the fields; to see that our heritage is not wasted. This can be done by the conservation commissions who should have access to sufficient record to carry on their work efficiently. Our need for data to solve our prospecting problems often leads us to extremes in our desire for records, but when laws require a company to display its records for use by competitors, the equity of the case is to be seriously doubted.

I do believe in the Federal study and interpretation of data, but I think enough organizations exist and that their power is sufficient without forcing the companies to deliver copies of their records.

All would be glad to see the public heritage properly conserved. If

the data were filed such purpose might be conserved to a limited extent, but the great saving would be in the expenditure of individual capital, and the Government cannot be expected to exercise paternal care to so great an extent. Therefore, the filing of record data is not justified for this reason.

I conclude, therefore, that the proposed bureau is unnecessary because there are Government instruments enough to do the necessary work, and that the purpose mainly urged in the paper would be either useless or unjust.

I. N. KNAPP, Ardmore, Pa. (communication to the Secretary*).—I believe that the author of this paper proves, by the examples he gives, the very opposite to what he advocates. I also believe that there is no practical need of creating National or State oil and gas bureaus, so far as collecting and filing of all well logs is concerned.

To emphasize this belief I quote him as follows:

> "All the large producing oil companies have recognized the practical importance of well logs and many now maintain special sub-surface departments whose chief functions are the recording and plotting of such data."

Also, he states that he "knows of a company that has saved and made several millions of dollars in one pool by the judicious application of such data."

It is my belief that these quotations most clearly indicate that there is no call for legislation to create bureaus to gather all well logs as proposed. Why tax the general public to carry on work that he shows to be practically covered by private enterprise?

I also believe that it is a proper function of the National Government gradually to map the whole area of the United States and have its general geology studied by experts, and bulletins of typical districts written and the results made public, thus forming a basis for private enterprise to develop geological detail for commercial purposes.

Many bulletins now published by the U. S. Geological Survey are of scientific and practical interest and give well logs in great detail. Such work should have more generous support than has been given it in the past both by the public, the State and the National Government.

To discuss another point in the paper, I will again make quotation from it, as follows: "Where a number of logs are obtained from the same field, any glaring inaccuracies resulting from an improper record are generally detected when checked against the average of several adjacent logs. Such inaccurate logs can therefore be rejected."

Now why should a log be rejected as inaccurate because it does not check with other logs in the same field?

Let me illustrate. A very conservative manager of a large company,

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* Received Aug. 17, 1917.

with whom I was acquainted, in looking over his maps discovered that there was room for another well between some good ones. He made a location about in the center of a plot having the form of an equilateral triangle, with a good well at each apex about 450 ft. distant from the central location made. From the logs of the three good wells he considered that he had a sure shot for an oil well, so he put in a field tank, ran his oil line, and brought up a shackle rod all before the well was drilled in. It proved to be a dry hole, no oil sand being found, although the three adjacent producing wells had between 20 and 30 ft. of good paying sand. I could cite other similar instances of dry holes among good producers.

The author indicates, in what I have quoted, that he would consider the log of the well described as improper and inaccurate, and proposes to reject such from record because it would not check with the average of several adjacent wells.

If the proposed bureau for collecting and filing well logs cut out such records as did not happen to suit the fancy of some bureaucrat, what sort of a muddle would the bureau soon get into? If an oil man in responsible charge of property did this it would properly be characterized as juggling the record.

I must dissent very emphatically from any such summary rejection of well logs from records as the author proposes.

I quite believe that such a bureau is impractical, and that it would require special men in the field for the sole purpose of supervising the making of well logs, if such data were to be collected with reasonable accuracy.

I believe every well log to be of real value for public record should carry with it the elevation of the top of the well casing from some datum point as sea level, also a reasonably exact land location with respect to property or other lines. This requires instrument work and measurements that practically no driller is capable of doing.

Again, in classifying the material penetrated, the driller or man in charge describes it as the sample recovered looks to him, and very likely in a way that would mean nothing to a geologist.

Let me illustrate. I personally took 112 samples in drilling a 2000-ft. well and sent them to the U. S. Geological Survey at Washington, D. C.

These formed the basis for writing *Professional Paper* 90-H. Sample No. 100 concentrated from mud on the bit pulled from 1841 ft. was described by the geologist in this paper as follows: "Coarse sand with chunks of gray calcareous sandstone, gray calcareous shaly clay, a few quartz pebbles up to 7⁄16 in. in length and fragments of shells; contains Hamulus onyx Morton, Ostrea cretacea Morton, and undetermined pelecypod cast."

My log of this horizon (15 ft. thick) made from this same sample and guided by the actual observation of the drilling, is printed in a parallel column in the same paper, as follows: "Layer of shale, conglomerate, limestone, sandstone; excessively hard, thin layers of pyrites; some layers of pyrites drilled at rate of 1 in. per hour with a Sharp & Hughes bit in good order."

The geologist in his description makes a feature of the fossils, and these I did not mention. On the other hand, I made a feature of the pyrites, which the geologist did not mention. Both were unquestionably present in the sample under observation.

I believe that it is apparent that if all well logs are to be made for public record in a way that will mean anything, it will be necessary to establish standards and have the work supervised by specialists.

It has always been a pleasure to me to furnish logs of wells to State or National surveys or bureaus that were thought to be of general interest, and quite a few of them have been published. That this practice is general is shown by the number of well logs, of which I contributed a few, published in *Bulletin* No. 429, U. S. Geological Survey. This bulletin clearly shows that it is no trouble for a reputable geologist to go out and get all kinds of oil-field information of scientific interest for publication. The "oil-smelling" kind of geologist referred to by the author will find any kind of information hard to get. But one does not have to go to the trouble and expense to hunt out practical up-to-date oil-field information. Maps of all the active oil fields and pools can be had, showing the location of all wells and their status brought right up to date. Such maps show the general trend of the development and give depths to the various sands.

The trade papers give frequent oil-field reviews and publish the location of important wells and give depths and other log data. They also give a monthly summary showing each individual well completed and in progress.

All this up-to-date information is furnished by private enterprise and can be had at a very reasonable figure; there is no secrecy about it. If such information were handled by oil and gas bureaus, it would take, judging by the usual course of events, 1 to 3 or more years to get it in shape for free distribution.

To keep up the oil and gas production of the United States requires the completion of a well every 20 min. 365 days in the year. I have no data to indicate how many other wells are drilled, possibly in all 40,000 in a year. It would be some job to keep proper track of all well logs.

Our present Chairman (A. F. Lucas) of the Institute's Oil and Gas Committee read a paper at the Richmond Meeting on Feb. 20, 1901, describing the famous Lucas gusher that he had brought in 41 days previously (Jan. 10, 1901). A log of the well was attached as a postscript to the paper.

You will invariably find that the men and companies that spend their own time and good money in drilling wells are quite willing to publish their discoveries and give their well logs to the world.

I have been credibly informed that geologists of the very highest standing advised against drilling for oil at Spindletop, Texas, and perhaps justly so. They said there was no precedent for finding oil in such recent formations.

Logs of various wells were quoted, notably the deep Galveston well, to prove that it was hopeless to look for oil along the Gulf Coast. Also, there was no known oil production in the whole world under the geological conditions there present.

However, our Chairman persisted in his efforts notwithstanding there had been three previous attempts at drilling to solve the mystery of Spindletop. He succeeded and brought in one of the world's great oil wells. He proved that his conception of the geology of the salt mound formation was right and earned for himself laurels that can never fade.

Spindletop has since produced over 40,000,000 bbl. of oil and the salt mounds of the Gulf Coast over 200,000,000 bbl.

The author refers to "the excellent logs of every well in California." This is rather a broad statement. I remember that, when I had occasion to make examination of a lot of well logs in several of the California fields some 10 or 12 years ago, I certainly failed to find the happy condition of affairs indicated by the author.

Many States have established bureaus or surveys to aid in the development of their mineral resources. It is, I think, fair to say that many of these have, for a time, under the direction of able men, done most excellent work considering the limited means placed at their disposal. But there has been a lack of continuity in their operation and some degenerate into resting places for "deserving" politicians.

I am utterly opposed to the formation of any State oil and gas bureaus to keep all well-log statistics.

The author himself is doubtful of the success of State bureaus for this purpose for he hopes "the bureau will not be made ineffective by the appointment, for political reasons, of incompetent and inexperienced officials."

The Chairman (A. F. Lucas, Washington, D. C.).—The activities of the Petroleum and Gas Committee for the half year have brought gratifying results, as exemplified by 14 technical papers, brim full of novel and attractive issues in which we are all keenly interested. I beg to extend my hearty thanks to all members and authors who have responded to my appeal, and to all those who have endeavored to meet the growing demand which has resulted in the augmented production

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[1] See *Trans.* (1901), **31**, 362–374.

of this commodity which is so essential for the Nation's welfare, comfort, and safety.

As you well know, within the last few years, petroleum and its products have been used for numerous and novel purposes, so that we have become accustomed to their use in thousands of activities, and pleasures as well, but we should also note that we have been wasting and misusing these wonderful products, apparently believing that their sources are inexhaustible.

Our country is indeed fortunate in its prodigal production of oil and gas, when compared with others less favored, but nevertheless we have arrived at a point where we ought to give its use more serious thought and consideration.

The maximum production, as given by the statistics of J. D. Northrop of the U. S. Geological Survey for last year, seems to have reached the enormous figure of 300,767,158 bbl., but in spite of this large production, the time has come when all our operators, chemists, and refiners are called to exercise their best ingenuity and technical knowledge, in order to continue increasing intensified production.

Owing to the war that we are now engaged in, our Government and those of our Allies will soon be compelled to make far greater demand than heretofore, and our efforts for increased production must necessarily follow, while those of the "Joy-riders" must proportionately decrease.

Let us therefore sound a warning, which coming from us may perhaps be better heeded, and caution the spendthrift joy-riders, and, indeed, all good patriotic citizens, men and women, to be less prodigal in the use of gasoline, and restrict the abuse of this most beneficent product. By so doing, rich and poor can contribute their quota toward shortening and winning the greatest war of all times.

I. N. Knapp, Ardmore, Pa.—I believe it would be a good thing if all mineral production in the United States were supervised and regulated by the Federal Government without any regard whatever to the doctrine of "State rights." This doctrine has been a great stumbling block to the proper development of our country. My idea is that the Government should compel the operators in each district to get together and make rules for the practical operation of their own business, subject to amendment or approval by the Government. These would then be the rules under which the operator would work and they would not affect property rights in connection with the fee of the land or in leasing.

This idea is in line with the first development of the West where miners made their own mining laws. In the oil and gas business, the operators gradually learn how to operate each particular pool in the best manner, and they are best fitted to direct what should be done in that pool with regard to drilling, casing, and plugging of wells, their distance from property lines, and the like.

I have had considerable experience with State regulation as carried out by the officials who have been appointed by the State. Such regulations have proved to be of no practical worth and have brought contempt on this method of regulation. I could cite many instances showing what a farce State regulation has been.

L. L. Hutchison, Tulsa, Okla.—I am not opposed to regulation that regulates. However, I am opposed to the mere piling up of records to be kept by official mandate without any system. Those of you who are owners of oil wells have found out how nearly impossible it is to make the prescribed reports.

As to the Oklahoma Corporation Commission's rules and regulations, at present the Oklahoma Bureau of Mines, which has been the officially constituted bureau with which drill records and logs were filed, and for years has had charge, according to the laws of Oklahoma, of the plugging of the wells, is now in a legal controversy with the Corporation Commission as to who is authorized to plug the wells. When we drill a dry hole down there, or pull a well, we don't know whether to call on the Bureau of Mines or the Oklahoma Corporation Commission. I agree with Mr. Knapp, that the men connected with these State bureaus are inefficient. Conditions could be much improved if the States would put their mining bureaus on a Civil Service basis, say, and require their employees to pass a practical examination before they are sent into the oil fields to tell old operators how to handle their business.

William Kennedy, Fort Worth, Texas (written discussion*).—In a former paper,[1] Mr. Matteson advocates the formation of a State or National bureau, in which the logs or records of all wells drilled should be filed for reference. In this new paper, he is pointing out the practical value of such a bureau and advocating a National one.

While I would endorse the value of such a bureau as an immense assistance to geologists, especially those working in such regions as western and southern Oklahoma, northern Texas, as well as the coastal regions of Louisiana, Texas, and eastern Mexico, regions in which it is extremely difficult and often impossible to determine any structure on the surface, yet I would hesitate, for many other practical reasons, to say that the filing of such records, especially where they may be open to public inspection, would be an unmixed blessing. Many of the records of wells are so carelessly kept that unless the geologist had a more or less intimate knowledge of what might be expected within the region he was examining, the log or record would be apt to mislead him. This carelessness is not all due to the small promoter; a great many of the large companies are equally at fault. Nor is this want of accuracy

* Received Sept. 10, 1917.

[1] *Trans.* (1917), **56**, 881.

in the records to be blamed altogether upon the contractors or drillers. In contract work, the general order is to make hole as swiftly as possible, irrespective of what materials may be passed through, and the requirements of a complete, well kept log or record are of secondary importance. The depth to production is the point desired, and to reach that point in the shortest possible time the whole energy of the driller is bent.

Neither the contractor nor his drillers are mineralogists, and the record of a well often shows considerable variation from the actual state of affairs. The driller records what he thinks he has. I have had several experiences of this kind and have examined a great many well records showing the same character of information. Not long ago, in watching the drilling of a well, the driller reported a blue shale, but as all the other wells in the field found a soft blue limestone about that depth, it looked strange that this limestone should be missing here. Upon examination of the sample of cuttings kept, it was foundth at the driller's "blue shale" was the limestone expected. In another well, the driller recorded having passed through 300 ft. of limestone and entered granite, which was drilled into some 300 ft. Here again, the cuttings showed that there were only a few feet of granite at the bottom of the hole and the remainder of the 300 ft. of driller's "granite" was a hard red sandstone, made up, largely, of granitic grains. As this was an important well in demonstrating how far east of the main granitic outcrops and at what depth we might expect to find granite, an error of this kind would be of serious consequence in developing the geology of the region.

If records of this character are filed in any bureau, of what use are they? Of course, the companies interested in the drilling can eliminate such errors by requiring the drillers to save samples of the cuttings, to be afterward examined by a geologist, whose report shall be used to correct the driller's records and make a true log. This keeping of samples of the cuttings has been adopted by a few of the more progressive companies. In all drilling carried on by the oil interests of the Southern Pacific Co. in California, Texas, and Louisiana, as well as eastern Mexico, this showing of the cutting is demanded, and the same requirements are made by the Lone Star Gas Co. in its drilling contracts. It may also be said that the Southern Pacific drilling operations are almost wholly under the direct supervision of the geologists, and by this means errors of determination are corrected at once and much useless drilling is avoided.

By this method an accurate log of any well can be obtained at a very small expense and thus eliminate the objection raised by Mr. Hutchison, that it would be necessary to keep a man at each well to obtain an accurate record. It would also have the effect of preventing the contractor or driller from "juggling" or guessing at the log, if either had any desire to do so. However, I do not think that many of the contractors or drillers

are guilty of such practices as deliberately faking their logs. I am more inclined to attribute their errors to their ignorance of the real nature of the material drilled through. This could also be overcome if some one of the company's geological staff were occasionally to visit the wells being drilled and explain to the drillers the difference between the several materials found. This information, I know from experience, would be appreciated by both driller and contractor and be of more than passing benefit to the geologist. Most drillers notice a great many conditions in the work not visible in any examination of the samples kept, and often in cases where he is not required to keep samples, a driller will save some material he considers abnormal or peculiar in that field. This he is ready to show, and all drillers are willing to tell their experiences and discuss them with the proper parties, so that the geologist often obtains a great deal of valuable information regarding the region in which his company is operating, which he otherwise never could get. I have heard it said frequently that there is a strong antagonism between the driller and the geologist, but I have never found any evidence of this antagonism, through a good many years of experience.

This system of geological visits, however, does not appear to be the general practice. With one or two companies this is an established rule, but with the great majority, after the well is located the work is turned over to some field man, who has no more knowledge than has the driller of the character of the material passed through—often not half as much—and the geologist hears no more about the well until after it is finished. He may get the log, but it is burdened with all the inaccuracies of the driller. Such a record as this is often dangerous and leads to the expenditure of many thousand dollars which might have been saved if the log of the initial well in a new territory had been accurately recorded.

Well kept and accurate logs of all wells are of undoubted value, not only to the owners but to all others operating within the same region. Every company recognizes this, although all of them do not put their ideas into practice. It does not, however, necessarily require a State or National bureau to collect these logs. In some respects, a State bureau may be made useful, but in others it is detrimental. In its useful capacity its authority may be exercised in seeing that all wells are carefully and properly drilled and cased, thereby preventing a careless company from damaging the property of its neighbors through the admission of water into the productive sands, and, in the event of the abandonment of a well, by seeing that the abandoned well is properly plugged before the casings are withdrawn. In other words, the bureau should have the power to prevent careless operators from working injury to any district by their carelessness. The bureau should also have the power to see that no unnecessary waste of either oil or gas, and, in many localities, of water, should be permitted.

In order to carry out these duties, the bureau must be provided with accurate logs of all wells, not so much perhaps of every change in the character of the formation, as the depth and thickness of all sands, and their conditions, whether carrying water, gas, or oil, or dry, and the character of the formations overlying these sands and in contact with them, the size of all casings and the depths at which they are set with the manner of their seating; or, in other words, a complete and accurate record of how the well is mounted. Without this information no bureau or company can tell what to do with any well. These difficulties are often encountered when a company takes an old well and attempts to work it over. Frequently the original owners of the well are not accessible, and it would be of the utmost importance if some bureau possessed available information of this kind.

I do not know under what conditions the State of California operates its bureau, but the State of Oklahoma has a bureau such as Mr. Matteson advocates. It is charged with the duty of seeing all abandoned wells properly plugged so that no water may enter into the other sands, and has a number of other duties, some of which are of undoubted advantage to operators, but others which, to say the least, appear to be of very doubtful value. In Texas there appears to be only one condition imposed by law, and that is the plugging of abandoned wells. The duty of seeing this done is placed on the County Judge, but does not appear to be observed to any great extent, and the sight of abandoned wells flowing salt water, even after the casings have been pulled, is quite common in all fields.

Admitting the great value of accurate well logs and their great use in any scheme for the most efficient means of conserving gas and oil, and, in many portions of the country, water, there are yet some very serious drawbacks to the formation of such bureaus as Mr. Matteson advocates. One of these is the loose and dangerous manner in which this information may be used. In both of Mr. Matteson's papers he emphasizes the great utility these logs would be to a geologist making an examination of a difficult portion of the country, provided a well had at some time previous to the examination been drilled, if the log of this well were available. From this it may be inferred that Mr. Matteson argues that this log should be filed with the bureau and thus become public property to be inspected by any one interested. It would be manifestly unjust to compel the drillers of that well to give up information that may have cost them many thousands of dollars to obtain. This would be compelling a man by law to give up what in all probability a courteous request would bring. He tries to make another point when he says, "Aside from the geological importance, if water troubles are to be prevented, if desirable and satisfactory data are to be obtained on the life and rate of decline of wells, if accurate estimates of oil reserves are to be obtained, well logs are vital and an efficient oil and gas bureau

becomes a practical necessity." So far as water troubles are concerned, when these assailed the Kern river field in California, the operators did not sit down and cry for a bureau to help them out; they simply got together and had the whole conditions of the field investigated, and in the light of that investigation set to work and overcame the water troubles themselves.

Another objection, particularly in wildcat territory, to making the logs of wells public property and available to any one interested, is the unscrupulous uses to which the information obtained is often put. These logs get into hands of irresponsible lease dealers and are "doctored" so as to show the presence of both oil and gas at depths where neither were seen in the well, and the leases are sold at high prices; fake companies start up and sell stock upon statements said to be verified by such logs. But there is no use elaborating upon this; anyone familiar with the literature of some of these companies, and the wild scramble following the opening of Spindletop and some of the other fields throughout the Coast country, will readily understand the conditions following the publicity given to the logs of the various wells drilled throughout that country, and the enormous loss of money involved.

Frankly speaking, I am not the least inclined to favor Mr. Matteson's idea of filing logs of wells in any place where either the geologist or outside parties have free access to them. It is true, this would be an immense advantage to the geologist, but there is also the risk of the information being used by unscrupulous parties to gouge the public. It is true that the Oklahoma Commission says this information is purely for the use of the Commission and not for public use.

So far as the formation of an oil or gas bureau is concerned, that may be all right so long as it confines its operations to the proper treatment of wells and the conservation of their products, but unfortunately such a bureau often goes outside of these functions and attempts to compel operators to do other things. As for gathering data regarding wells and their production, we already have at least three very efficient gatherers of such statistics and do not need another. Another objection to the establishment of any such bureau as that advocated by Mr. Matteson is that the officials are usually chosen from political life, as a general thing have no practical knowledge of either the oil or gas business, and adopt many regulations that are contrary to the best practice of either business.

What I want to see is the introduction of methods whereby we may obtain accurate logs of all wells drilled, and the introduction of such methods lies with the operating companies themselves and not with any bureau, no matter how formed. The geologist can do much to bring this about, but as he is not omnipotent he must have the aid of the men at the head of the various organizations throughout the oil and gas world.

Chairman Lucas.—Like Mr. Kennedy, I have a strongly adverse personal feeling about the matter of filing records. When I drilled the first great gusher in Texas, I labored hard, single-handed, without help from anyone, until finally I was rewarded by a most wonderful and sudden result. Within a week, 23 train loads of well-drilling machinery were coming from every part of the United States, principally West Virginia and Pennsylvania, cable outfits mainly, as at that time it was very hard to obtain a rotary rig and drillers capable of running it. The new arrivals all wanted the log of my well, and appointed a committee to ask me for it. I had kept that log jealously, because it was my own property and I had no other protection. I knew that I was surrounded by inimical conditions, and as I did not have all the land on the Dome that I felt I was fairly entitled to, it was natural for me to wish, for the time being, to keep the log secret. However, feeling confident that they could never use the cable-drilling system on Spindletop, I published the log that day, but warned them of the difficulties awaiting them.

They went ahead, however, and in about a week, 17 rigs were hung. We had about 250 ft. of quicksand through which no cable drill could pass until, a year afterward, the pressure beneath the quicksand and the gas were exhausted; then, of course, they could use the cable drills. In consequence, a great many of those drillers went home to their old jobs, but some of them applied themselves to mastering the rotary rigs, by working as helpers, and in time they have become experts. Some of them are probably now working in Oklahoma.

I. N. Knapp.—When drilling with the rotary, you can tell instantly when the character of the ground changes, for example, from hard to soft, but otherwise you do not know what you have. Drilling at 2000 ft. (609 m.) with a 4-inch (101.6-mm.) rotary drill pipe in an 8-in. (203.2-mm.) hole, the circulating mud pump running at a fair speed, it takes between 20 and 30 min. for cuttings from the bit to reach the surface and show the quality of the ground penetrated.

Before I attempted rotary drilling, I had had considerable experience with cable tools. When I started to use the rotary, I hired drillers with rotary experience, and with them I had my own cable-tool men, who had been some time in my employ. That these men could "make hole" was shown by the fact that the first well attempted in unconsolidated material reached a depth of over 2400 ft., but I failed to learn the actual character of the material penetrated.

After many failures, I concluded that coring would be necessary to ascertain the exact character of the strata that I had penetrated, which carried gas in quantity, and, I believed, included sand and shale each of considerable thickness. I purchased a core barrel similar to those which have been successfully used in coring for coal and slate, but it proved a

failure in unconsolidated material. From the experience thus gained, I designed a hollow bit and core barrel which proved a complete success in getting cores of soft material. I found that the gas-bearing streak mentioned consisted of very thin alternate layers of sand and shale, none over 1 in. thick. In previous drilling in this horizon the overflow had been carefully watched, and showed at times a predominance of the shale through which the drill would advance about 10 ft. and then encounter sand for about the same distance. This is to be explained by the jigging action of the pump on the mud circulation, which would separate and aggregate the various materials according to their specific gravities as they rose to the surface. At that time there were also differences in the rate of drilling which seemed to indicate the penetration of a stratum of considerable thickness.

Sometimes the driller would say, "We are in a nest of boulders" and by the way the drill pipe was jumping and the drive chain was slapping around one would think that such a diagnosis was correct. On pulling out, you would find the fish-tail bit worn square across, with no evidence of boulders. I have had my core barrel run in at such times and found we were drilling in a dry clay carrying a very sharp sand. A dull bit had probably worn the bottom of the hole into irregularity, causing the drill pipe to jump and the chain to slap around. I had before observed under like conditions that a sharp bit would cause the difficulty to disappear.

My son, Captain Arthur Knapp, who helped me design the successful core barrel mentioned above, has since spent two years in the Russian Baku fields. He says that all the large operating companies there are very careful in taking samples from their wells. They use a tool something like a casing splitter, which is run to the bottom. This tool then throws out a suitable scraper which gouges out a sample from the wall of the well when pulled up a foot or two. When lowered again the scraper pushes back out of the way and the tool is brought out with a sample of the real stuff just at the bottom of the hole.

While watching rotary drilling I have noted wells which proved good oil producers but did not show a drop of oil while being drilled. The depth at which to expect the top of the oil sand was known; when the driller felt a change in formation at about that depth, he would watch for particles of oil sand in the overflow and by tasting them could judge whether he had oil or not.

I have always had feet marked on my rotary grief pipe and by careful measurement of the drill-pipe joints, as they go in, I can always tell the exact depth of the bit below the rotary jaws. It is impossible to know the depth with equal accuracy while drilling with cable tools; in the latter case, when the exact depth is required, the tools have to be withdrawn and a steel measuring line run in.

H. M. Ami, Washington, D. C.—Twenty-nine years ago I had charge of drilling by the Canadian Geological Survey. The drillers were very desirous that the companies that were engaged in drilling for gas, oil, and water should send their materials to the Department in order to have them examined and recorded. For the most part, such materials were sent in a very intelligent fashion, although occasionally we received samples which were obviously not correct. For instance, in the drillings around Montreal city, where we find imbedded intrusive masses, we received records of coal encountered by the drillings at depths of 800, 900, and over 1000 ft. Probably these drills were set up too close to certain power plants, and coal and other extraneous matter got into the samples.

W. Van Der Gracht, Tulsa, Okla.—I had been directing the Geological Survey of the Netherlands for some 12 years before coming to this country, and during that time had extensive experience with well logs. Mere filing of such logs for the use of the public, without further comment or interpretation, I consider worse than useless. I have been conducting explorations for coal deposits under the extensive Tertiary plains of the northwestern European Continent, and for this purpose had to gain knowledge about the pre-Tertiary floor of these regions, where rock outcrops do not exist for hundreds of miles, and the logs of wells are the only information to go by. This pre-Tertiary floor is extensively cut up by faults into a chaos of blocks. On the highest ones, coal or valuable potassium salts may be reached within workable depths, but on the sunken blocks, the pre-Tertiary floor is thrown below all practicable depths. These conditions had all to be learned from well logs. I gathered and considered thousands of them, and thus may say that I got some experience in handling this sort of information. That experience, however, taught me that by far the great majority are totally incorrect; some in a bona fide way, which may become useful after careful study and interpretation; many, however, are absolutely useless, because they were faked by careless drillers who neglected to keep samples or to make the necessary periodical observations.

The difficulty is greatest with rotary or circulator-drilled wells, where the mud flush greatly changes the samples, for instance, making it impossible to distinguish between sands and only slightly sandy clays. Also, the depths at which fossils occur in such open holes are very uncertain. At one time it was highly important for us to determine the exact depth of a Miocene key-horizon. I found most characteristic Miocene shells, in numerous well samples, distributed over a depth from 400 to 2000 ft., whence it seemed that there was a Miocene, 1600 ft. thick, beginning at a depth of 400 ft. Nothing of the kind! The Miocene was only 500 ft. thick, and the fossils found lower down had simply

dropped from the wall of the hole (even from behind casing), had descended as far as 1100 ft. and become mixed with samples of Palaeocene clays, which contained nothing but very friable fossils, leaving no recognizable fragments in the samples. All this was cleared up later, when I used a wonderfully satisfactory coring device, which I had designed, in order to get true sections and to clear up the embarrassing muddle. I thus secured cores of 75 per cent. of the strata; in fact, of everything which was not actual quicksand. Thus the many unexplainable puzzles in the old well logs were finally all cleared up.

If we have encountered such difficulties with well logs, how can you expect the public to get useful, instead of highly dangerous and misleading, information from logs procured at random, out of a mere official file? It is worse than useless.

The only thing to do is to support competent U. S. Survey or State Geologists, to compile well logs of a certain field, interpret them, and offer their conclusions to the public, *after careful study and discrimination.* Every sane-minded operator will gladly coöperate with them and contribute his logs to such work. In fact, much splendid work has been done already along this line, and is being done privately by companies maintaining a geological department.

That much can be achieved in this way, I know personally by experience. I was able by such means to unravel the mystery of the pre-Tertiary floor under the Netherlands very satisfactorily and to discover over 150,000 acres of coal-bearing lands, with an estimated coal resource of 4½ billion tons. Consequently, I have nothing whatever against carefully filing and compiling well logs, provided only that these are interpreted, after considerable study and discrimination, by a geologist, who then publishes his conclusions.

Dorsey Hager, Tulsa, Okla.—One reason why there has not been greater coöperation between the geologist and the driller is that so few technical men have gone into the oil fields as practical workers. If our young men, when they were about to leave the universities, would go into the field, work with the drillers, and get an intimate knowledge of field conditions, then they could talk intelligently to the drillers and to the operators. Most young men will not do this, with the result that today there is a mere handful of geologists and engineers in the United States—in California there are more than in Oklahoma—who have any idea of practical field conditions. Until the time comes when our college-trained men go into the field to serve an apprenticeship, as many engineers do, the fullest coöperation will not be attained between the driller and the operator, on the one hand, and the geologist on the other.

W. E. Wrather, Wichita Falls, Tex. (written discussion*).—The establishment of State bureaus for collecting and filing well records appears to me to be of very doubtful value. The main objection is that it is manifestly unjust to compel a company to divulge private information which has cost considerable sums of money to obtain, and thereby make it possible for competitors to take such information and use it to the detriment of the rightful owner. In most cases, a geologist will succeed in securing the well records he desires, provided he makes an earnest enough effort, but the company in possession of such information should undoubtedly have the right of granting or withholding it at will. Fortunately, with each passing year, the tendency of producing companies to withhold information which may have general geologic value is lessening, and today it is doubtful whether companies who maintain the secrecy of former years in regard to such matters really gain anything by so doing. Coöperation in the exchange of well records very fortunately prevails throughout the Mid-continent and Southwestern oil fields.

There are certain duties which a State bureau may profitably perform, such as the prevention of waste of oil, gas, and water for drilling; the supervision of setting casing and plugging abandoned wells to prevent the flooding of oil or valuable fresh-water sands with salt water. No operator should be allowed to injure the property of adjacent owners by his carelessness, and to safeguard the rights of others it is proper that certain information, relating principally to the sands, their content, depth at which casings are seated, and the manner of their seating, should be turned over to the bureau; and such bureaus should have the authority to demand the delivery of information within the scope of their jurisdiction, or penalize producers for refusing to furnish it.

It will be observed that the duties properly assignable to State bureaus are more mechanical or technological than purely geological. The principal objection today relates not so much to the accessibility of well records as to their faulty and imperfect character, and, personally, I am more concerned about improving the standard of well records than in gaining access to a mass of data of questionable accuracy, to which I may not justly be entitled.

The great fault of recording well data in the past lay in the fact that this duty was left entirely with the driller, who usually knew nothing whatever about geology or ordinary rock classification. Furthermore, until recently, the only use to which well records were put, by the operating department, was to show the depth, thickness, and character of the sands, and to indicate convenient casing seats; also the character of the formation as affecting the probable rate and cost of drilling. It is

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* Received Nov. 30, 1917.

often the task of the geologist today to work over well records made according to this standard and naturally they are lacking in the particulars he is most anxious to obtain. The drillers trained under the "old school" are today making the most of the well records, and as many of them are men of limited education, they fail to see readily the utility of making any "new fangled" changes in the method of keeping such data.

To secure better well records is a question of educating drillers and of closer coöperation between the geological and operating departments. The education of the driller in this respect is to be largely accomplished by an exchange of views with the geologist and it must be admitted that thus far there has been a lack of team work between the two. The geologist, who often knows next to nothing about the practical operation of a drilling rig, assumes a critical and sometimes a supercilious attitude toward the driller, who knows next to nothing of geology. The failure to get together is too often the fault of the geologist, who has never had the opportunity of getting acquainted with drilling crews as a class, or to secure any first-hand information about oil-field operations. The geologist fresh from a technical school, garbed in regulation khaki suit, riding trousers and puttees, comes on the derrick floor and asks a lot of questions, involving the use of geological terms which shoot entirely over the head of the average driller. At once an antagonism springs up which is hard to overcome.

The driller's ideas of formation are largely formulated by the manner in which the bit is scored; by the rebound of the tools in the hole, as evidenced by the spring of the cable; or in the case of rotary tools, by the smooth rotation of the drill pipe in soft-cutting shale, compared to its jumping and jerking in hard rock. Sound is often indicative, particularly with a rotary rig, as the pipe "growls" when drilling in certain formation. Color, which means much to a geologist, is too often entirely disregarded.

To the geologist untutored in such matters, these criteria mean nothing. He wants to see cuttings, to which the driller often objects, as it takes time to run the bailer, and when work is done by contract, according to the driller's views, time spent in useless recovery, washing, and proper labeling of cuttings is money lost for the owner of the tools.

The best method of overcoming antagonism between the geologist and the driller is to train the geologist to an extent in oil-field operations. He should be willing and anxious to serve an apprenticeship in the field, dressing tools, or helping on a rotary rig; and it would be desirable also for him to work in other capacities under the order of the field superintendent. Under this training his critical attitude would soon disappear. The time of such apprenticeship should depend entirely upon the personal qualifications and adaptability of the individual. He will soon learn

to "speak the language" of the field force and thus be in a better position to convert them to ideas of geology, and he will obtain an education of great value to him in future years.

Furthermore, he should remember that in the next several years the demand for geologists to do surface mapping as a guide to wildcat operations will decrease decidedly. The rapid acceptance of geology by the oil producer during the past 3 or 4 years has created an abnormal demand for men to do surface mapping, and this has resulted in profitable employment of men rather poorly equipped to do general oil geology. The technical schools have responded with courses in oil geology and there will soon be a slackening in demand, which will hinder the new graduates of these schools from finding employment in the kind of work for which they have prepared. A geologist with a technical or mechanical bent, will, with experience, soon develop into a very desirable drilling and production superintendent, and with the days of declining oil production in the United States, which seem to have already set in, methods of necessary conservation and intensive production will most readily be originated by men with broader and more technical educations. The oil industry can use an unlimited number of such men, and many geologists who take the proper attitude, and are willing to take the hard knocks in acquiring practical field experience, will readily find a profitable and desirable berth in this class of work, when he would have difficulty in securing a position on the geological staff. During the next few years, geology will inevitably be diverted from its present capacity of searching for new fields to a general supervision of drilling and producing operations, and the geologist who prepares for the coming of this day will find himself in a fortunate position.

W. G. Matteson (written discussion*).—The discussion on the practicality of oil and gas bureaus has been of value in that it has presented in detail the advantages and objections to which the functions of such bureaus are subject. On the constructive side, it is conceded that the protection of neighboring wells against encroachment of water, loss of gas pressure, etc., by supervising the effective shutting in of gas and water sands, the plugging of abandoned wells, and regulating the shooting of productive sands, is desirable; also that the careful compilation of well logs would result in the accumulation of valuable data *provided accurate logs* could always be secured. Opposed to these advantages is the contention that the majority of well logs are inaccurate and therefore worthless, due to the ignorance of the driller as to correct nomenclature; that making a log public may often be unfair to the operator who has spent a large sum to obtain information which he might consider espe-

* Received Feb. 9, 1918.

cially valuable to him; and, finally, that political appointments of inexperienced, incompetent men destroy the efficiency of such a bureau.

Admitting the importance of the above objections, none of them is so serious that it cannot be easily overcome. If it is true that logs are inaccurate as a rule, why not take immediate steps to eliminate this undesirable feature? Producers, engineers, and geologists have been talking and lamenting about inaccurate logs for the last several years, and yet how much concentrated conscientious effort have they made to overcome this defect? It is easy to recognize and condemn an existing evil; to apply an effective remedy demands initiative of a different character. A concern which can afford to gamble $20,000 to $50,000 in a test can also afford the small amount of extra time and labor required to obtain an accurate log of that test, and corporations which are drilling scores of wells yearly, representing an outlay of hundreds of thousands of dollars, can ill afford a number of inaccurate and worthless well records when the expenditure of $5,000 to $15,000 yearly for a few competent geologists would remedy the evil. Such geologists could be assigned to cover certain districts daily wherein the company is operating, obtain the cuttings from the various wells and make a proper classification and record. In the case of the small, independent operator, such an expense could be avoided by the collection of cuttings, as suggested by Mr. Kennedy, which could then be sent to the bureau, where an accurate classification and record could be made by a bureau geologist.

As to the publication of such logs, it is only right that the operator should be protected. The writer has previously[1] suggested that laws could be framed so that every operator may be safeguarded until he wishes to release the log; or, after two or three years from the data of filling, the log might automatically become open to public inspection. The most secretive operator could hardly offer conscientious objection to such a ruling since the elapse of time will give him sufficient opportunity to derive full benefit from his information.

It is not true, as has been claimed, that a geologist can generally get all desired logs if he seeks them diligently and diplomatically, because often no logs of tests have been kept and, in other instances, some companies refuse the information on general principles, although what they would lose by releasing such records is difficult to perceive at times. A bureau provides a general and definite place for the preservation of this information and the enforced filing of logs insures a definite record of all wildcat wells far in advance of production. How often has the geologist gone into a territory to find that, several years ago, one or two wells had been drilled; and, on locating the driller or operator after considerable expenditure of time and energy, has met with the reply, "Oh, I never kept any

[1] *Trans.* (1917), 56, 881.

record of that well. Believe we got a sand . . . ., etc." Meanwhile, this same operator has released his acreage, has no further intention of prospecting that locality, and hence no object in withholding information. Yet, because no law compelled him to keep and file an accurate log, information of possibly great value to some new investigator is lost, probably resulting in the squandering of more capital which might have been applied to better advantages in some other region.

Having learned that the Roumanian Goverment had established a national bureau for the purpose of conserving their oil resources, the writer made inquiry of one of the Standard's superintendents, who had recently returned from that country, concerning the methods of regulation prescribed by that bureau. A bureau engineer is assigned to a district. These engineers exercise the most stringent precautions and insist on absolute adherence to the law. For instance, after a water sand is cased off, the well must stand 24 hr., after which the bailer is run to ascertain whether the water has been completely shut in. This procedure must be repeated until the engineer is satisfied and gives his permission to continue drilling. Graphic logs, with a cross-section of the well and with casing record drawn to a prescribed standard scale, must be filed with the bureau on the completion of each well. These logs, arranged so as to give a graphic representation of different sections through the field, are suspended in a room specifically designed for that purpose, which is always open to the public. Companies offered strenuous opposition to these methods at first, but experience has so demonstrated the value of the regulations that they are now commended by all. The publication of the logs, which has been one of the main objections raised in the present discussion, caused little concern, each company taking the view that the ruling was equally fair to all, and finding it possible also to protect themselves satisfactorily.

As for the political appointments of incompetent, inefficient men, is this argument supported by fact? Are we not harping on theoretical possibilities? There has been no cry of the operators against the ability and efficiency of the deputies of the California State Bureau and, in case of the Oklahoma Bureau, some of its members have resigned to accept very advantageous offers as field superintendents of reliable operating companies. This reveals the type of man who has been employed in many instances; but admitting the possibility of incompetent appointments, there are various ways and means to prevent or avoid them.

A principle which is recognized as desirable and correct should not be condemned and discarded before practical application because of a few objections to be overcome. The value of accurate well logs is unchallenged; the need of a sort of general clearing house, known to all, where such logs may be filed, to be released to investigators after a cer-

tain elapsed time, is evident. The creation of a State or National bureau with the necessary powers prescribed by law supplies this need, and it has been pointed out how every conscientious objection to such a bureau may be eliminated. And only when criticism and a continued state of forbearance and inactivity are superseded by constructive action and coöperation among the leaders in the petroleum industry, will distinct progress be made toward the solution of this problem, so pertinent to the conservation of capital and resources.

A Review of the Exploration at Belle Isle, Louisiana

BY A. F. LUCAS,* WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

Introduction

BELLE ISLE, located in the low sea marshes near Atchafalaya Bay, is the southeasternmost of the famous Five Salt Islands of Louisiana. Rising about 80 ft. (24 m.) above the level of the surrounding marsh and being some 800 acres (323 ha.) in extent, it constitutes a striking landmark on the low Gulf coast, and since the eighteenth century, when it played an important part in the operation of the buccaneers, it has been intermittently frequented by man. In 1897, the writer discovered the great mass of salt that underlies the island and shortly afterward published a brief account of the geology of Belle Isle and the neighboring salt islands.[1] In that paper he described principally the salt, but pointed out also the occurrence of petroleum and of sulphur in the strata above the salt. Since that time the island has been extensively explored by the drill, and the developments up to 1908 have been described by Veatch[2] and Harris,[3] and Knapp.[4]

The writer's early conclusions as to the probable presence of sulphur in considerable quantity have been confirmed by the later drilling, some of the wells having found quantities of gas containing hydrogen sulphide and a number of sulphur-bearing strata as well. Under the stimulus of the present keen demand for sulphur for war purposes, a syndicate of New York capitalists was recently formed to explore the sulphur resources of the island in more detail. Under the writer's direction, six wells were drilled for this syndicate during the winter of 1916–17. The present paper presents briefly the results of this work and sums up the present knowledge of the geology of this highly interesting locality.

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* Consulting Engineer.

[1] A. F. Lucas: Rock Salt in Louisiana. *Trans.* (1899), **29**, 462–474.

[2] A. C. Veatch: *Louisiana Geological Survey Report* (1899), 221–229.

[3] G. D. Harris: *Louisiana Geological Survey, Bulletin* No. 7 (1907), 18–26; *U. S. Geological Survey, Bulletin No.* 429 (1910), 43–48.

[4] I. N. Knapp: *Journal of the Franklin Institute* (November and December, 1912), **174**, 447, 639.

History of Development

The first well drilled was near the old landing, in November, 1896. It found no salt at 590 ft. (179.8 m.) (Lucas 1), but the second well, (Lucas 2), located in the center of the northern section of the island, reached salt at the depth of 335 ft. (102 m.). No. 3, now in the canal, and No. 4 reached salt at 135 and 325 ft. (41 and 99 m.), respectively. No. 3 found oil of a canary yellow color 37° Bé. at a depth of about 125 ft. (38 m.), and No. 4, south of it, found some oil with considerable gas at 335 ft. (102 m.), where it stopped in salt.

The Gulf company shortly afterward put down 13 wells to determine the contour of the salt surface, and in August, 1898, started a shaft close to the site of their prospect well No. 11, which found incipient salt at 103 ft. (31.4 m.) and solid salt at 140 ft. (42.6 m.). A shaft was carried to a depth of about 390 ft. (118.8 m.), at which level an entry was driven in the salt to the southwest and then to the south-southwest, for a distance of 340 ft. (103.6 m.). This point must have been practically at the edge of the salt mass, for when it was reached the working face caved and water and sand rushed in, filling the entry and shaft to sea level within a few hours. A second shaft was soon started, but the difficulty of penetrating a considerable thickness of quicksand at 210 ft. (64 m.) led to its abandonment a few feet short of reaching the salt body. An attempt was also made to extract the salt by pumping it out as brine. In this manner several thousand tons of salt were obtained, but, owing to the fact that the salt in this part of the island is much impregnated with oil and gas, it was found difficult to crystallize it, and it was also left with rusty stains. The unconsolidated character of the overburden also permitted the escape of brine to the surface, which, in spots, threatened to cave; hence it soon became evident that this process would result in reducing the whole island to sea level and salt-mining operations were therefore abandoned.

Harris[5] says that the Gulf company put down another series of 10 test wells, designated A to J. He gives the logs of several of these wells, one of which (well I) is of considerable geologic interest, but unfortunately neglects to give their exact locations. The locations of wells H, I, and J on the accompanying map (Fig. 1) are therefore only approximate. I quote the log of well I.

Notes on Well I, Belle Isle (Location No. 12 of Knapp's Notes)

	Feet
Hard rock, one-third sulphur	203
Salt	375
Salt, limestone, and oil sand	430
Quicksand	450

[5] G. D. Harris: *Louisiana Geological Survey, Bulletin* No. 7 (1907), 18–26.

Notes on Well I, Belle Isle (Location No. 12 of Knapp's Notes).—(Continued)

	Feet
Sulphur and hard shale	485
Sulphur rock	503
Sand rock with showings of gas and oil	550
Salt	585
Hard material, no salt	655
Sand rock, showings of gas and oil	675
Improving showings of gas and oil	700
Flint rock of the hardest kind	735
Salt	840
Hard material	856
Material softer, good showing	1010
Gas pressure; brown sand rock and blue shale containing oil; gas flame 25 ft. high	1165
Loose brown oil sand and slate	1202
Bottom of 4-inch pipe	1210
Gas blowout; oil showing improving	1230
Increase in gas	1375
Diamond drill installed	1430
Rock salt	1455
Iron sand; probable depth from sample left in bottle	1550
Gas blew water out of well; oil showing on slush tank	1575
Salt	1625
Strong gas pressure; better oil showing	1945
Salt, with showing of gas and oil to bottom	2035
Total depth	2359

Partial Section of New Orleans Mining and Milling Co.'s Well No. 3 (Location No. 10 of Knapp's notes)

	Feet
First oil showing	1396
Oil showing increasing	1506
Big gas showing; good oil showing	1584
Small gas showing	1786
Small gas showing	1930
Gas and oil in sand	2190
Last 80 ft. give pronounced oil showing	2270
Biggest pyrite showing; good oil showing	2389
Total depth	2411

No salt reported in this well.

In 1906, the New Orleans Mining & Milling Co., having taken over the property, drilled two wells near the north end of the island, in an effort to find oil. Although the deeper of these wells reached 2411 ft. (734.9 m.) and is less than 1000 ft. (304.8 m.) north of the old salt mine shaft, it found no salt, indicating that the northern face of the dome is very abrupt. This well reported good shows of gas and oil between 1396 and 2389 ft. (425.5 and 728 m.), but apparently no commercial accumulations were encountered.

Section of Knapp Well No. 1, Belle Isle

	Feet
Shale of varying hardness; at 75–80 ft. went through old mine timber; at 100 ft. pyrites showed	0– 114
Whitish limestone	115– 122
Shale	122– 128
Gravelly shale	128– 133
Hard lime	133– 142
Oil sand; oil at 143 ft.	142– 145
Shale and specks of sulphur	145– 159
Hard streak of lime; gas from 150 to 160 ft.	159– 160
Shale, showing oil	160– 186
Blue clay; *Rangia cuneata*, fish bones, and lignite at 186–189 ft.	186– 187
Rotten limestone	187– 200
Probably salt	200– 211
Salt of varied character	211– 912
Salt with strong oil odor and taste to 700 ft.	
Salt, impure, with anhydrite, 418–767 ft.	
Puff of gas, 838 ft.	
Anhydrite, 844, 864 ft.	
Salt, varied	912–1790
Slight bubbling of gas at 1090 ft.	
Gas enough to give flame 2 or 3 ft. above 10-in. pipe at 1348 ft.; this gas has a strong odor of crude oil; does not react for H_2S with acetate of lead; came from salt dirty with black shale; similar gas at 1423 ft.; both the puffs of gas lasted for a couple of days and then died out.	
Clear, clean salt at 1425–1790 ft.	
Greenish shale; mixed salt, gypsum incrusting salt, also black sand at 1800 ft.	1790–1800
Impure gypsum; gas held in salt crystals, which give a crackling noise when dissolving	1800–1806
Gas sand	1806–1810
Impure gassy salt	1810–1840
Soft gypsum	1840–1850
Impure salt showing a little gas and oil	1850–1904
Probably impure salt; overflow frothing with gas and showing oil	1906–1965
White salt	1965–2030
White salt with gas and oil showing; dark amber oil at 2100 ft.	2030–2112
Bits of pyrite and lime	2112–2114
Opaque salt with puffs of gas; fine amber oil obtained by straining scum at 2190 ft.	2114–2190
Salt, varied, clear and white or opaque and dirty with gas; oil obtained by straining froth at 2500 ft.	2190–2520
Salt, particles of galena and pyrites of iron and copper	2520–2606
Salt, anhydrite, and sulphur	2606–2628
Salt, impure and crystal; varying hardness; good oil showing; puff of gas at 2722 ft. threw water up 20 ft.	2628–2740
Black sand—magnetite	2740–2745
Gas throwing water up 40 ft. in derrick	2783
Dark-red oil in salt	2900
Anhydrite	2940–3005
Bottom of well very hard—anhydrite or gypsum	3005–3171–3171

Notes on Knapp Well No. 2, Belle Isle

	Feet
Sand, gravel, clay, streaks of limestone to sulphur rock	0–290
Gypsum and sulphur rock	290–390
Sulphur, gypsum, and anhydrite, very porous	390–880
Salt	880–890

Remarks: Average sulphur, 7.5 per cent.; best sample, 35 per cent.; slight showing of oil at 390 ft., where porous rock began. Well was 430 ft. southwest of first and about 100 ft. southeast of the little blacksmith shop, in flat back of salt works.

Log of Knapp Well No. 3, Belle Isle

	Feet
Stiff clays	0–127
Streaks of sand, clay shale, gravel, wood, and wood fiber	127–460
Lime rock	460–475
Gypseous material, clay, and salt	475–550
Impure salt	550–620
Clear salt	620–800

Not a particle of oil or gas showed; nor any sulphur.

In 1907–08, I. N. Knapp drilled three wells on the island, the northernmost of which, No. 1, close to shaft No. 1 entered salt at about 140 ft. (42.6 m.) and continued in it to 3050 ft. (929.6 m.), though in the last 600 ft. (182.9 m.) several occurrences of limestone, sand, anhydrite, sulphur, and oil were noted. This well also found a canary yellow oil of 37° Bé gravity at less than 125 ft. (38 m.) It encountered heavy puffs of gas at many horizons in the salt, and throughout the depth of 1500 to 3171 ft. (457 to 966 m.) found dark red oil of 37° Bé in salt and stopped in hard bottom. For the last 300 ft. (91 m.), this well labored very hard on account of heavy gas pressure, and was abandoned at the depth of 3171 ft.

In 1916–17, the writer drilled six wells on the island for the purpose of ascertaining the extent of the sulphur deposits known to overlie the salt in places. This work has demonstrated that the sulphur-bearing rock is very impure and is confined chiefly to a narrow belt across the northern part of the island, though it has also brought to light some very interesting information on the configuration and extent of the salt surface. It became evident, however, that as a potential source of sulphur Belle Isle may be eliminated.

Topography

Belle Isle consists of a ridge about ¾ mile long, extending in a northeast-southwest direction, and rising to a height of 80 ft. above the surrounding low marsh (see Fig. 1). The ridge is cut by a pronounced saddle which practically isolates a small hill at the northeastern extremity, known locally as Bald Hill. This depression is of especial interest, for it appears to be a reflection of the contour of the salt mass itself.

Geology

The surface of the island is covered by unconsolidated recent sediments, the surficial layer being in general a clay. This is commonly underlain by a thick mass of sand, but the beds are highly lenticular and in some localities the sand is irregularly interbedded with lenses of clay, gravel, and shale. Limestone with the pores and cracks filled with sulphur is reported in certain wells; also gypsum and anhydrite. At the northern end of the island the surface deposits are inclined at an angle considerably more than that of the surface of the underlying salt.

The most detailed section extant of the strata above the salt is that measured by Veatch[6] at the old salt-mine shaft and reproduced below. The occurrence of clay containing barite and metallic sulphides is of especial interest, and it is also noteworthy that the upper surface of the salt mass is not sharply defined but grades up through impure salt to "clay with large salt crystals."

Section of Shaft No. 1, by A. C. Veatch

No.	Depths	(Elevation above tide 7 ft. (2.1 m.)	Feet	Inches
1	0 – 4	Clay	4	0
2	4 – 13	Hard sand	9	0
3	13 – 30	Blue clay	17	0
4	30 – 40	Blue clay and sand	10	0
5	40 – 63	Hard clay and gravel	23	0
6	63 – 68	Blue clay with crystalline masses, from the size of marble to a man's head, of barite, galena, sphalerite, pyrite, and chalcopyrite	5	0
7	68 – 95	Blue clay and shells	27	0
8	95 –96½	Rock. Impure black limestone; and barite	1	6
9	96½–103	Blue clay with masses of barite near the base	6	6
10	103 –116	Dark-colored clay with large salt crystals	14	0
11	116 –117	Dark-colored clay with oil	1	0
12	117 –142	Salt with dark-colored clay	25	0
13	142 –162	Discolored salt	35	0
14	162 –163	White limestone	0	8
15	163 –175*	Dirty salt becoming white.		

* This was the depth of the shaft at the time of Mr. Veatch's departure, May 19, 1899.

Further data on the character of the material above the salt are given in the following logs of the six wells drilled by the writer, and of the Knapp No. 2 well.[7]

[6] C. A. Veatch: *Louisiana Geological Survey Report* (1899), 221–229.
[7] *Op. cit.*

Logs of Syndicate Wells Drilled at Belle Isle, Louisiana, December, 1916, to March, 1917

Well No. 1

From	To (In Feet)	Thickness	Strata
0	13	13	Surface muck and clay
13	102	89	Clays and shales with a little sand
102	153	51	Sanded red and blue clays and shales
153	224	71	Gray sand
224	254	30	Hard gray sand
254	260	6	Very hard gray sand (probably sandstone)
260	289	29	Soft gray sand
289	291	2	Hard shale
291	292	1	Gravel
292	330	38	Fine gray sand
330	355	25	Gravel
355	375	20	Coarse sand
375	389	14	Coarse sand (very hard)
389	419	30	Hard blue rock
419	438	19	Sand
438	441	3	Rock
441	457	16	Sand
457	465	8	Shale
465	481	16	Sandstone at first turning to limestone

Well No. 2

From	To (In Feet)	Thickness	Strata
0	20	20	Surface muck and clay
20	38	18	Fine gray sand
38	56	18	Banded red and blue clays
56	80	24	Fine gray sand
80	127	47	Clay and shale
127	135	8	Sandstone
135	210	75	Sand
210	225	15	Hard sand
225	246	21	Clay and shale
246	281	35	Gray sand with a little oil
281	286	5	Limestone with some gypsum
286	341	55	Sulphur rock with pores and cracks filled with sulphur

Logs of Syndicate Wells Drilled at Belle Isle, Louisiana, December, 1916, *to March,* 1917.—(*Continued*)

Well No. 3

From	To (In Feet)	Thickness	Strata
0	80.00	80.00	Surface muck and clay with a little sand
80	163.00	83.00	Blue and red banded clays and shales
163	173.00	10.00	Soft sand
173	224.00	51.00	Hard blue clay (gumbo)
224	280.00	56.00	Sand
280	295.00	15.00	Rock
295	310.00	15.00	Sand
310	375.00	65.00	Gravel
375	380.00	5.00	Hard clay
380	386 00	6.00	Gravel
386	391.00	5.00	Rock
391	396.00	5.00	Sulphur rock
396	399.00	3.00	Clay
399	404.00	5.00	Gravel
404	445.00	41.00	Sand
445	566.25	121.25	Gravel and rock salt

Well No. 4

From	To (In Feet)	Thickness	Strata
0	10	10	Surface muck and clay
10	85	75	Clay
85	198	113	Banded blue and red clay and shale
118	225	27	Hard blue clay
225	405	180	Blue sand
405	415	10	Sandstone (?)
415	465	50	Gray sand with some gravel
465	489	24	Gravel and sand (chert pebbles)
489	494	5	Rock
494	499	5	Sulphur rock
499	513	14	Gravel mostly chert
513	525	12	Rock
525	565	40	Sand, shale and gravel
565	715	150	Gypsum with sulphur (last 35 ft. sulphur sparingly).

Logs of Syndicate Wells Drilled at Belle Isle, Louisiana, December, 1916, *to March,* 1917.—*(Continued)*

Well No. 5

From	To (In Feet)	Thickness	Strata
0	6	6	Surface muck and clay
6	179	173	Fine sand
179	315	136	Sand, gravel, and shale (some gas in shale)
315	319	4	Shale
319	329	10	Hard clay
329	355	26	Shale (some gas)
355	393	38	Gravel and hard clay
395	430	37	Gravel
430	446	16	Shale
446	490	44	Lime
490	534	44	Salt

Well No. 6

From	To (In Feet)	Thickness	Strata
0	4	4	Surface muck and clay
4	415	410	Gray sand (medium fineness)
415	460	55	Lime and gypsum, with some sulphur
460	505	45	Lime, harder and showing little sulphur
505	545	40	Salt

Syndicate well No. 2 was lost at a depth of 341 ft. (103.9 m.), but as it was drilled close to Knapp No. 2, which penetrated to 890 ft. (271.3 m.), it may be disregarded, and Knapp No. 2 used as a criterion. Knapp No. 2 reports 590 ft. (179.8 m.) of gypsum, anhydrite, and sulphur immediately above the salt, and Lucas No. 2 passed through 60 ft. (18.3 m.) of sulphur rock at the same horizon. These two wells must have encountered the sulphur deposit at its maximum thickness, for Syndicate wells 3 and 4 found only 5 ft. (1.5 m.) of sulphur rock and Nos. 1, 5 and 6 found no commercial sulphur rock at all. Knapp well No. 1 found a little sulphur at two horizons and the Gulf well I reports a number of sulphur-bearing strata. As shown by the later drilling, however, the sulphur encountered in these wells is either in very thin layers or in local lenses of no commercial importance. The thick deposit in Knapp well No. 2, which Mr. Knapp says averages 7.5 per cent. sulphur, is evidently of very limited extent.

The relation of the sulphur to the salt, and the available information as to the contour of the salt surface itself, are shown by the tabulated well data in the accompanying summary of logs.

Summary of Logs of Wells Drilled at Belle Isle, 1896–1917

Well Name	No.	Elevation +	Oil Showings At	Sulphur Rock At	Top of Salt	Total Depth	Reference
Lucas........	1	5	....	....	......	590	*a*
Lucas........	2	13	Traces	276	335	410	*a*
Lucas........	3	7	110–125	*d*110–125	135	170	*a*
Lucas........	4	5	135–275	*d*135–275	325	.....	*a*
Gulf shaft....	1	7	116–117	None	117	326	*b*
Gulf shaft....	2	5	None?	16.0–190	210	196	*b*
Gulf shaft....	H	5	None	340–391	....	420	*b*
Gulf shaft....	I	20	303, 940, 985, 1086, 1105, 1212, 1365, 1370	256 to 285, 870, 1206	1,545	2,450	*b*
Gulf shaft....	J	3	None	None	....	600	*b*
New Orleans Mining Co.	2	..	....	....	....	1,740	*c*
New Orleans Mining Co.	3	..	1396, 1506, 1584, 2190, 2270, 2389	None?	....	2,411	*c*
Knapp.......	1	..	143, 160, 1850 to 1965, 2100, 2190, 2500, 2722, 2900	*d*145 to 159, 2606 to 2628	150	3,171	*c*
Knapp.......	2	..	390	290 to 880	880	890	*c*
Knapp.......	3	..	None	None	550	800	*c*
Syndicate....	1	..	None	None	....	481	
Syndicate....	2	..	246–281	286–341	....	341	
Syndicate....	3	..	None	391–396	445	566	
Syndicate....	4	..	320–499	494–499, *d*565–715	....	715	
Syndicate....	5	..	None	None	490	534	
Syndicate....	6	..	None	*d*415–505	505	545	

a A. C. Veatch: *Louisiana Geological Survey, Report* (1899), 221–229.
b G. D. Harris: *Louisiana Geological Survey, Bulletin* No. 7 (1907), 18–26.
c G. D. Harris, *U. S. Geological Survey, Bulletin* No. 429 (1910), 43–48.
d Sulphur present in traces only.

These data are shown graphically in the accompanying cross-sections (Fig. 2). It is evident, in the first place, that the salt dome extends much farther to the southeast than has hitherto been supposed, and that it probably underlies all of Belle Isle Lake at a depth of less than 800 ft. (243 m.). This eastern and southeastern portion of the salt body forms, with respect to the highest part of the latter, a kind of terrace or shoulder, dipping only from 490 to 505 ft. (149.4 to 153.9 m.) in a distance of nearly 2000 ft. (609 m.) (between Syndicate wells 5 and 6). Beneath the ridge and hills of Belle Isle the salt is not nearer the surface; on the contrary, the nearest salt was found in the northern part of the island in the flat area where the first shaft was dug, where salt was found at 113 ft. (34.5

m.). To the north of this shaft, or at the base of the outlying hill forming the northern extremity of the island, the salt pitches off with extreme abruptness, from 113 ft. at the shaft (No. 1) to 1545 ft. (471 m.) in well I.

The most interesting feature, however, is the deep hollow, or saddle, which crosses the island transversely near its northern extremity, and which is apparently reflected in the topographic depression practically isolating the hill near which the old shaft was sunk. As shown by the

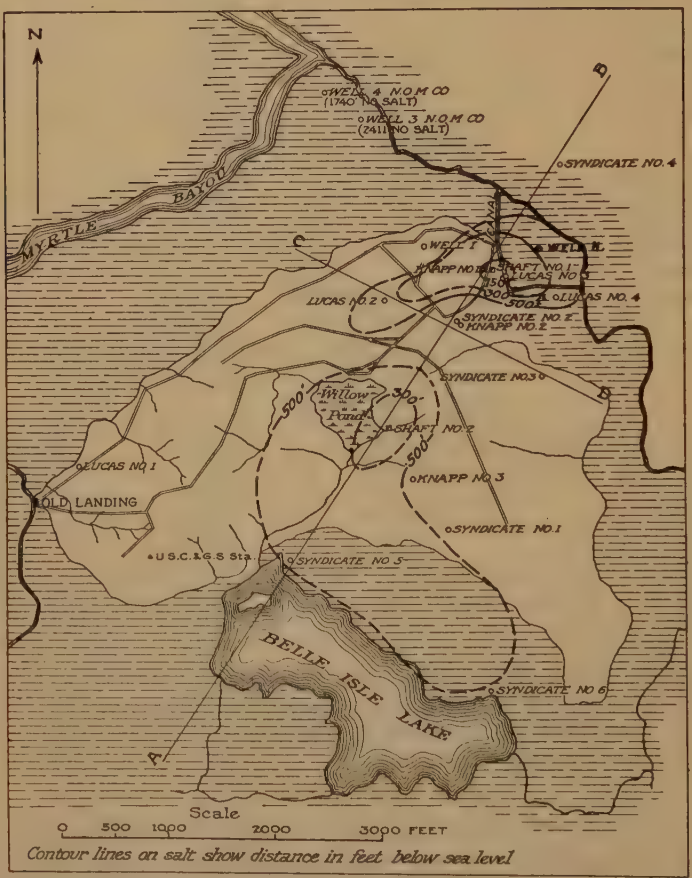

FIG. 1.—MAP SHOWING LOCATIONS OF RECENT WELLS AT BELLE ISLE, LA.

record of the Knapp No. 2 well, this hollow is at least 880 ft. (268.3 m.) deep. On either side of it, as shown by the two shaft sections in Fig. 1, the salt rises nearly to the surface, being found at a depth of only 113 ft. in shaft No. 1, and 210 ft. (64 m.) in shaft No. 2. It is only in this saddle, or notch in the salt surface, that a considerable thickness of the sulphur rock was found. The Knapp No. 2 well found 590 ft. (179.8 m.) of sulphur rock with the salt at 880 ft., and the Lucas No. 1, which is

somewhat higher up on the northwest slope of the saddle, found 60 ft. (18.3 m.) of sulphur rock with salt at 335 ft. (102 m.).

Although slight irregularities in the salt surface have been observed at other salt domes along the Gulf Coast, the writer knows of no case in which the irregularity is nearly as pronounced as that just described. Minor depressions in the salt have been plausibly ascribed to solution by the fresh water that doubtless percolates down to the salt in places, but the saddle or hollow at Belle Isle appears to be of primary rather than of secondary origin. This is suggested by the fact that it is now filled with nearly 600 ft. (182 m.) of hard sulphur rock which was undoubtedly deposited by the escape of sulphur and sulphureted hydrogen of an incipient

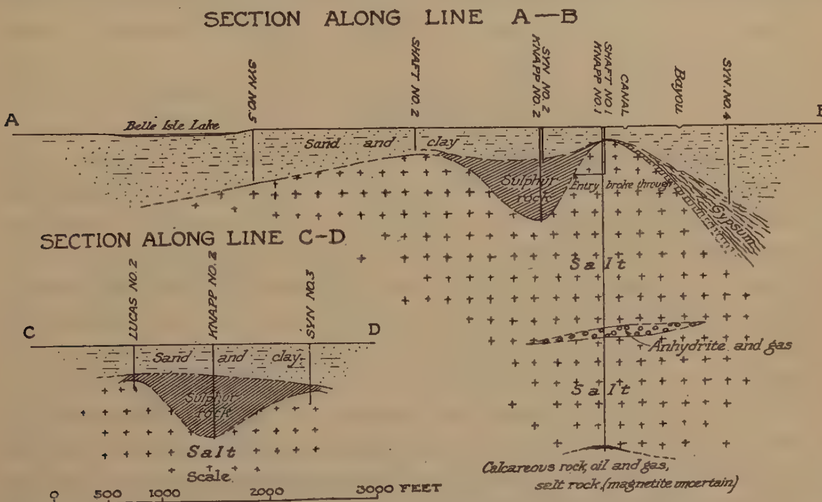

FIG. 2.—CROSS-SECTIONS OF RECENT WELLS SHOWING SULPHUR POCKET AT BELLE ISLE, LA.

volcanic origin, analogous to the fumaroles on Spindletop, Texas, as pointed out by the writer to Prof. R. T. Hill[8] in his description of the Beaumont oil field, and also as pointed out by Eugene Coste.[9] The same agencies undoubtedly give rise to the salt dome itself, and perhaps also to the marked construction and depression in the topographic ridge above. It seems reasonable to suppose that the Belle Isle dome is of compound origin, and that the salt ascended chiefly along two lines of weakness, the main one located in the southern part of the island and the smaller one beneath the hill at the northern end. It is probable that these fissures or lines of weakness unite at no great depth, or that they

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[8] R. T. Hill: The Beaumont Oil Field, with Notes on Other Oil Fields of the Texas Region. *Journal of the Franklin Institute* (August–October, 1902), **154**, 143, 225, 263.

[9] Eugene Coste: The Volcanic Origin of Natural Gas and Petroleum. *Journal of the Canadian Mining Institute* (1903), **6**, 74–128.

represent simply one plane of weakness, along which the escape of the gas was more effectively obstructed beneath the saddle than beneath the two salt knobs. As to the manner in which the salt, with its gas and oil inclusions, attained its present position, at a depth of 1500 to 3171 ft.—whether in solution or as semi-plastic mass—has never been satisfactorily explained. The solution of the problem must await a further understanding of the genetic relations of the oil, gas, sulphur, and salt, and such an understanding should result from thorough analytical and experimental observations or from practical demonstration test borings, or, more likely, from a combination of both. The writer urgently suggests the drilling of a deep well, of say 5000 to 7000 ft. (1524 to 2133 m.) or more, at a point carefully selected above the fissure, in order to arrive at a conclusion more practical than the innumerable hypothetical suggestions now current.

Discussion

I. N. Knapp, Ardmore, Pa. (written discussion*).—Captain Lucas has made an important contribution to our knowledge of Belle Isle, especially as to the outline of the sulphur deposit, and the pronounced irregularity of the salt surface. I agree with his view, that the elevation or mound that constitutes Belle Isle may consist of several salt domes formed at different periods.

It may be interesting to note the high gas pressure encountered in pockets in the salt by my No. 1 Belle Isle well. While drilling a dry hole with cable tools, we sometimes struck pockets of gas which would float the tools, so that drilling had to be suspended sometimes for several hours at a time, until the head of gas was exhausted, and this with a string of tools, 5-in. (127 mm.) stem 36 ft. (10.9 m.) long, that weighed complete about 3500 lb. (1587 kg.). At about 1800 ft. (548 m.), we struck the largest gas pocket and the string of tools was thrown up and nearly out of the hole. When it dropped back it made several loops with the ⅞-in. (22-mm.) drilling cable, and gave us a bad fishing job. While drilling with the rotary (we had a combination rig), several gas pockets were encountered that would blow water and oil over the top of the derrick for an hour at a time.

If the salt had been porous, so as to yield the contained oil and gas, No. 1 well undoubtedly would have proved to be a paying one. I feel sure that a paying well will never be found in the Belle Isle salt, although it is quite possible that there may be a sand lense or lenses contiguous to the salt which will yield oil in paying quantity.

When No. 1 well was abandoned, it was giving over 100,000 cu. ft. (2831 cu. m.) of gas and over 3 bbl. of oil per 24 hr. and had reached a

* Received Oct. 9, 1917.

depth of 3171 ft. (966 m.). At the time, it was the deepest well on the Gulf Coast, but much deeper wells are now quite common.

The author suggests drilling a well 5000 to 7000 ft. (1524 to 2133 m.) deep, and thus hopes to solve the mystery of the Belle Isle formation. With the better tools and equipment now to be had, I feel confident that it would require no greater effort to reach a depth of 5000 ft. than it was to drill No. 1 well 3200 ft.

Another interesting matter is the extreme porosity of the sulphur rock found in my No. 2 well. While drilling with the rotary, and using a thick mud, the mud-pump circulation disappeared as soon as we penetrated this rock. By stopping the advance of the drill and pumping in sawdust and horse manure, the walls of the well could be puddled off, but as soon as the drill advanced a few feet further, the overflow would once more disappear. Later, we used clear water for drilling and the cuttings went off through the walls of the well. There was no evidence of caverns, but the porosity must have been very great.

I used the gas from No. 1 well in an unsuccessful attempt to jet the water from No. 2, which probably delivered 10,000 bbl. in 24 hr.; the water was highly impregnated with sulphureted hydrogen, and there was some showing of a dark thick oil. No. 2 well indicated a typical salt-mound formation; that is, from the surface downward, we penetrated sand, gravel, and clay, with streaks of limestone, down to the highly porous sulphur rock, and then the salt. Unfortunately the pores of the sulphur rock were filled with water instead of oil.

As I have never noted a published analysis of Belle Isle oil, it may be of interest to present the one accompanying this discussion. The sample was taken from a shallow well (probably less than 500 ft. deep) in January, 1903, on my first visit to Belle Isle. At that time the salt company was still operating.

In my No. 1 well, I found the same light straw-colored oil at 143 ft. At increasing depths more oil was found, but the color darkened to amber, and at 2900 ft. it was dark red.

About six months ago, a small sample of crude oil was given me, which was said to have dripped from a natural-gas line. The well yielding this gas was about 2600 ft. deep and located on the Lirette Plantation some 20 miles below Houma and 50 miles east of Belle Isle. My chemist reported that this oil was very similar in color, odor (smelled like furniture polish), gravity, and paraffin contents, to the deep Belle Isle oil. This is certainly remarkable. I have had perhaps 20 gas analyses made from surface gas escapements, and in every case the marsh-gas content was high, that is, over 95 per cent. I also had three analyses of gas from depths between 1600 and 1700 ft., and these were over 98 per cent. marsh gas.

Analysis of Crude Oil from Belle Isle, St. Mary's Parish, La.

Color, Canary — Specific gravity, 0.8462 = 35.45° B.
Water, Trace — Sulphur, 0.38 per cent. by weight

Fraction Number	Temp. of Distillation	Per Cent. by Volume	Per Cent. by Weight	Specific Gravity Direct	Specific Gravity Baumé	Description of Fraction	Resumé
1	200° F.						
2	200–250						
3	302						
4	325–350	0.3	13.78	0.7933	46.50	Colorless	
5	350–400	14.4					
6	400–450	16.3	15.64	0.8121	42.40	Light apple green	Burning oil,
7	450–500	15.5	15.32	0.8364	37.40	Light canary	302–572° F.,
8	500–550	12.6	12.66	0.8506	34.60	Canary	63.4 per cent.
9	550–572	4.3	4.36	0.8581	33.15	to	
10	572–600	5.5	5.60	0.8621	32.40	Deep canary	
11	600–650	9.5	9.73	0.8673	31.40	Deep canary	Paraffin oils,
12	650–700	17.2	17.56	0.8640	32.05	Deep canary	572–780° F.,
13	700–780	4.7	5.00	0.9007	25.45	Deep canary	36.9 per cent.
Coke....	....	.....	0.60				
Total....	....	100.3	100.25				

Remarks: This is a very remarkable paraffin-base crude oil, very light in color, low in coke residue. Paraffin separates from last fraction very much like refined oil. April 13, 1903. C. C. Tutwiler, Chemist.

On page 48 of U. S. Geological Survey *Bulletin* 429, is the following erroneous statement regarding Belle Isle oil: "It is said to be of paraffin base and to become nearly solid on one day's exposure to the sun. The same is true of the oil coming in small quantities from the Knapp well No. 1." Dr. Ueberloddue, a German petroleum expert, is given as the authority for this statement. As a matter of fact, no such thing ever happens. My recollection is that Dr. Ueberloddue and his party arrived at Belle Isle late in the evening and were obliged to leave early the next day to catch the train they wished to take in order to keep up with their itinerary. We went out soon after daylight to look at the old salt works and my No. 1 well. It so happened that the temperature was then close to the freezing point. The oil from the well had spread on a pond about 150 ft. in diameter to a depth of several inches and at this temperature the surface had a crust of paraffin about like medium soft butter. Of course, this crust would melt when the sun came out and would remain perfectly liquid for months unless the temperature was again reduced. Dr. Ueberloddue got the wrong impression as to the cause of the paraffin hardening. I believe that undue haste in making examinations of this kind is often the cause of grievous errors.

In the section of shaft No. 1, by A. C. Veatch (p. 1039), the material

between 63 and 68 ft. is stated as "blue clay with crystalline masses, from the size of a marble to a man's head, of barite, galena, sphalerite, pyrite and chalcopyrite." In my No. 1 well, between 2500 and 2600 ft., we got the same material. Mr. Veatch was at Belle Isle at the time and recognized the material from the well as almost identical with what he had previously seen in the shaft. It has always been considered very remarkable that the materials mentioned were found in a crystalline salt at that depth.

J. A. Udden, Austin, Tex.—I would like to ask Dr. Dumble whether he ever learned of the occurrence of any of these metallic minerals in association with the salt dome uplifts and the salines in northern and eastern Texas.

E. T. Dumble, Houston, Tex.—I cannot say as to the salt domes, but in some of the upper Tertiary deposits in eastern Texas such minerals have been found, chalcopyrites mainly, but also a number of fragments of galena.

W. Van Der Gracht.—Little kernels and specks of galena, zinc, iron pyrites, copper pyrites, sometimes traces of millerite, are not at all rare.

W. E. Wrather, Wichita Falls, Tex.—Mr. C. L. Baker, formerly connected with the Rio Bravo Oil Co., of Houston, Tex., called my attention to an occurrence of oolitic barite, discovered in wells in the Saratoga oil field, Hardin County, Texas. Specimens were carefully studied by Mr. E. S. Moore and reported on in a recent publication.[1] The oolites in this instance were about the size of small shot.

E. T. Dumble.—The concretions are sometimes annular; sometimes they are like small peas or shot; sometimes they are highly polished and sometimes they have no polish. They occur anywhere from 800 to 1000 ft. deep in the southern part of the Saratoga field, in the Batson field, and in other oilfields in Texas. I do not know exactly in what connection they occur; they just come up with the drillings.

F. B. Plummer, Tulsa, Okla.—To me the existence of these sulphide minerals in connection with the salt domes is not to be wondered at. A great deal of iron and sulphur accumulates around these domes, just as the oil accumulates. There is also more or less heat around these salt domes from the exothermal reaction of the water with the salt and particularly with the cap-rock which gives an alkaline reaction. Those conditions are favorable for chemical precipitation. The action of sulphate waters and perhaps sulphureted gases on mineral matter buried in the sedimentaries would naturally yield sulphide minerals, such as galena, sphalerite, and pyrite. Great quantities of pyrite occur in some domes, particularly around Iberia, and the only surprise to me is that there is not more of it.

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[1] *Bulletin of the Geological Society of America* (1914), **25**, 77–79.

A Feasible Plan for Gaging Individual Wells

BY ROSWELL H. JOHNSON,* B. S., M. S., AND W. E. BERNARD, PITTSBURGH, PA.

(St. Louis Meeting, October, 1917)

To know the rate of decline of oil wells is very important, yet ordinarily we are prevented from getting this rate because the oil from several wells is put into one or a few tanks as soon as the wells have been connected to the power. The object of doing this is economy of tanks, pipes, labor, and accounting.

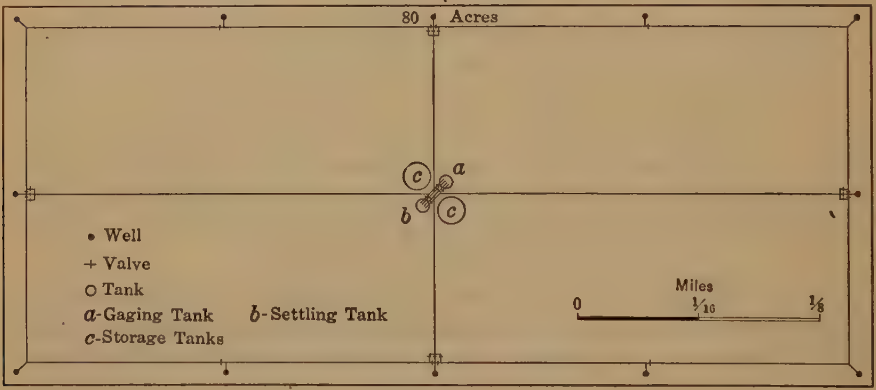

FIG. 1.

But we are making a great sacrifice for this small economy. The advantages of the individual records are that we would know better:

1. When the well has ceased to be profitable.

2. When a well has gotten into bad condition.

3. The real record of the yield per acre and the rate of decline of the wells. The lease total is too mixed as to age of the wells for existing records to have much significance. To obtain such individual well records and more accurate logs are the great desiderata of satisfactory management of oil properties.

4. Which wells show the slowest decline—information which is valuable in locating new wells or purchasing adjoining leases.

The truth of this statement is evident, when we reflect upon the large number of purchases of producing property. Intelligent appraisal is of the utmost advantage and it is impossible without individual well-pro-

* Professor of Oil and Gas Production, University of Pittsburgh.

duction figures. Such figures are valuable not only for understanding the lease in question, but for making comparisons.

In contemplating the wide variation in the estimation of values, one is impressed with the importance of ascertaining the data that are so fundamental to appraising.

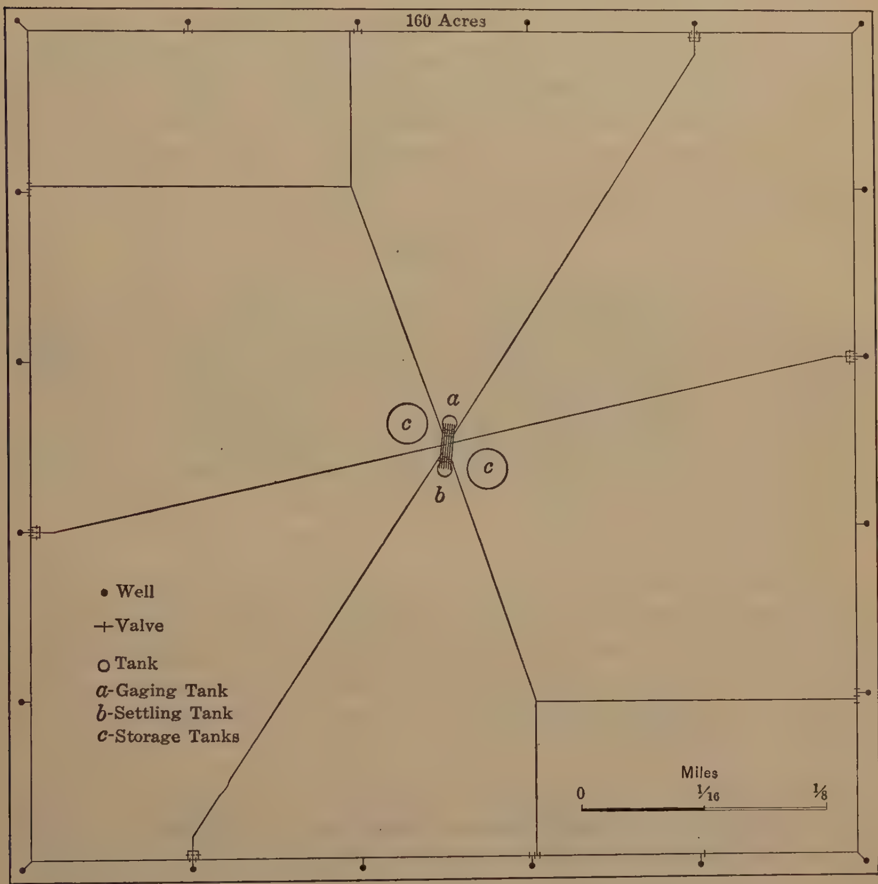

FIG. 2.

Many who would concede the general argument so far, still consider individual gaging too expensive. To reduce this expense we have devised a plan, whereby an individual gaging is taken of each well in rotation. Each well produces into the special measuring tank one day and then into the general receiving tank until its turn comes in rotation for another individual gage. Such records will be frequent enough to give the advantages listed.

The pipe required is far less than would be imagined owing to a scheme of valve manipulation we have devised. Fig. 1 and 2 show the arrange-

ments on leases of 80 and 160 acres respectively. On 10- and 40-acre leases separate radial lines are used.

The cost of the additional materials required, exclusive of the gaging tank, teaming and labor, is at present prices $50, $56 and $20 per well for a 160-, 80- and 40-acre lease, respectively, as compared with the most economical possible arrangement.

Should this expense for the 80- and 160-acre leases be thought prohibitive for routine use, it is recommended for the 10- and 40- acre leases as in these cases the additional expense is not so great per well. A compromise plan in larger leases would be individual tanks for certain wells, the performance of which would have especial significance, and separate tanking from shallow and deep sands.

Discussion

C. P. Bowie, San Francisco, Cal.—In my work with the U. S. Bureau of Mines, I have been detailed to report on oil storage containers, and in going around the country I have been much interested in the possibilities in the use of concrete for such containers. In talking with a good many people, I have discovered that concrete is being used a good deal for a wide variety of oils ranging from 14° to as high as 40° Bé., especially by some of the western railroads. If any member has had experience in storing gasoline in concrete, I should like to learn of it.

Dorsey Hager, Tulsa, Okla.—Mr. M. J. Munn, formerly with the U. S. Geological Survey, and connected with the Gypsy Oil Co., until a short time ago, has invented a method for the application of concrete to the storage not only of crude oil but of refined products. The Cosden Oil Co., of Tulsa, is now working on that plan, and expects to store refined products in concrete tanks.

The Chairman (I. N. Knapp, Ardmore, Pa.).—I have had considerable experience in building masonry tanks for gas holders. I do not believe it is possible to build a concrete or masonry tank of any kind, with hydraulic cement mortar, which will be absolutely water-tight, unless the bottom and inner wall, after completion, are waterproofed with some impervious material. The same will be true of oil tanks, the bottom and inner face of which must be made impervious to oil.

Masonry tanks are intended to be sunk in the ground. All that the wall of the tank then amounts to is a retaining wall to keep the excavation in shape and to receive an impervious coating. The bottom of a tank, when empty, must be protected by a relief valve from any hydraulic pressure acting from below.

At ordinary prices for sheets, a tank 100 ft. diam. by 20 ft. deep can be built of steel much quicker and cheaper than of masonry. Several

years ago I saw a very large oil storage tank under construction in California, the bottom and sides being concrete. The sides were sloped at about the angle of repose for the material excavated.

If you build a steel oil tank, you can get men for the work who do nothing else, and you are likely to get a satisfactory job; but if you wish to build a concrete or masonry tank it is almost impossible to find men skilled in building absolutely water-tight masonry work. The general idea is that any laborer can mix up concrete in any old way and make a good job, but this is a mistake, for it really requires skilful work to do a good job of concreting or any other class of masonry, particularly where water-tight or oil-tight work is wanted.

E. T. Dumble, Houston, Tex.—I can tell you about the concrete tanks in the Curran field. We have two 500,000-bbl. tanks and two 750,000-bbl. tanks, built 4 years ago, one of which was emptied this summer. There was no penetration into the cement by the oil that had stood in it for 4 years. These tanks are from 400 to 500 ft. in diameter, and 20 ft. in depth; the coating on the bottom is 3 to 4 in. thick and on the sides a little over 2 in. They are reinforced with 6 in. mesh, No. 6 wire. They have been perfectly satisfactory and there was very little evaporation from them. There are a great many of such tanks in use in the California field.

Geosynclines and Petroliferous Deposits

(A Contribution to the Study of the Relations between Earth Movements and Hydrocarbon Accumulations)

BY MARCEL R. DALY, SEATTLE, WASH.

(St. Louis Meeting, October, 1917)

In a preceding paper[1] the writer has pointed out some apparent relationship between the distribution, on the surface of the globe, of the known hydrocarbon deposits and the disposition of the principal zones of deformation of the earth's body (geosynclines). He wishes to present some further remarks on this subject.

Petroleum deposits, or their derivatives or descendants, are found through the whole range of sedimentary strata, from the pre-Cambrian to the Quaternary. For instance, the pre-Beltian (Shuswap) series of the Canadian Cordillera are interbedded with limestones which are sometimes rich in carbonaceous matter (Sicamous limestone); and the Beltian system of the same region contains layers of argillites and dolomitic limestones which are equally high in carbon content.[2] This carbon may be interpreted as the last remnant of hydrocarbons previously contained in the rocks. On the other hand, the actual formation of hydrocarbons under conditions that would eventually permit their deposition in modern sediments is conceded by some geologists.[3] Between these two extreme limits, bitumens are found indifferently in all sedimentaries. They constitute a continuous series, some kind of a large family, whose individual members may be entitled to a community of origin and probably represent the different stages of evolution of the primitive, parent matter. The kinship between petroleum and such substances as asphalt, ozocerite, manjak, grahamite, albertite, etc., is well known; and E. H. Cunninggam-Craig has recently shown how intimately shale fields and oil fields are connected.[4] For this reason, no distinction is made in the present

[1] The Diastrophic Theory. *Trans.* (1917), **56**, 733.

[2] *Geological Survey of Canada, Guide Book* No. 8, 124 and 134.

[3] Sir Boverton Redwood: *Petroleum* (1913), **1**, 133–134; Dalton: *Economic Geology* (1909), **4**, 620; E. T. Dumble: *Trans.* (1914), **48**, 526.

[4] E. H. Cunningham-Craig: Kerogen and Kerogen Shales. *Journal of the Institution of Petroleum Technologists* (1916), **2**, 238–273. "Kerogen" is a term denoting the substance or substances contained in Scottish oil-shales from which oil is obtained. According to Craig, kerogen is formed by the inspissation of petroleum and by the adsorption of inspissated petroleum by argillaceous material.

study between the different kinds of hydrocarbons and they are treated as a whole (petroliferous deposits).

James D. Dana has applied the term *geosyncline* to the great earth troughs that have taken place in regions of excessive deposition, and that he considers preparatory to the formation of mountain ranges.[5] Subsequent writers, and especially Prof. Emile Haug, of Paris, have greatly extended the meaning of the term as well as the idea itself. Haug considers the geosynclines as zones of weakness and mobility of the earth between two stable masses (continental areas);[6] whereas T. C. Chamberlin regards "the crumpled tracts as lying on the border of great segments of the earth that acted essentially as units."[7] From a mechanical standpoint, the idea is fundamentally the same. Considered in this light, a geosyncline is no more simply the expression of a local deformation: it belongs to a general system of distortion which is world wide (network). The term geosyncline is used in this paper with this extended sense, but without any special inference as to the real nature of the intervening areas or earth segments, which is immaterial for the present discussion. Individually, geosynclines are to be considered "the sites of subsequent foldings of the strata, or, conversely, the regions of folded mountains are the sites of former geosynclines."[8] A geosyncline may be simple or compound, and is more generally compound.

It seems difficult to call in question the fact that the location as well as the trend of these world-wide zones of mobility have somewhat varied in the course of ages. For instance, Marcel Bertrand has shown[9] that three successive zones of distortion have developed in northern Europe during Paleozoic and pre-Paleozoic times, progressing from North to South and giving rise to three distinct mountain chains, which would have been preceded by as many geosynclinal belts.[10] On the other hand, in some regions, the general strike of the foldings seems to have remained constant during extensive periods. Thus, in British Columbia, the

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[5] J. D. DANA: *Manual of Geology*, 4th Ed., 380, 385.

[6] Emile Haug: Les Géosynclinaux et les Aires Continentales. Contribution à l'étude des transgressions et des régressions marines. *Bulletin de la Société Géologique de France*, 3 Ser. (1900), **28**, 617–711; also *Traité de Géologie* (1908), **1**.

[7] T. C. Chamberlin and R. D. Salisbury: *Geology*, **1**, 543.

[8] A. W. Grabau: *Principles of Stratigraphy*, 902.

[9] *Bulletin de la Société Géologique de France*, 3 Ser. (1887–88), **16**, 576.

[10] M. Bertrand terms the first chain the *Huronian*. It would have extended, in pre-Cambrian times, from the Lake region of North America through the Highlands of Scotland, Norway, Sweden, Finland, and reached the extremity of Asia. Later, toward the end of the Silurian, a second chain (the *Caledonian* chain of Suess), would have arisen further south, and, finally, toward the end of the Carbonic, a third one, the *Hercynian* chain (Variscic or Armorican of Suess) would have arisen still further south, extending from southern Ireland and French Brittany, through the Vosges, the Ardennes, the Black Forest and the Hartz, to Bohemia. The Appalachian folding, which dates of this same period, would have been a link of this latter network.

Rocky Mountain formations, above the Shuswap (pre-Beltian) series, show a Cordilleran trend throughout; and from the Beltian (pre-Cambrian) to the Permian, the enormous pile of sediments involved, more than 50,000 ft. thick, present no thoroughgoing unconformity. This would indicate, at least for this part of the Pacific Geosyncline of North America, a permanency in the direction of the strains extending through the greater part of the known history of the earth. But even there a change may be detected if we trace the movements further back; for the underlying Shuswap (pre-Beltian) strikes N. 70° E., nearly at right angles to the prevalent strike of the later formations.[11]

Thus, in order to examine in some detail the mutual relations between geosynclinal belts and hydrocarbon (petroliferous) deposits, it would be necessary to define the different successive synclinal networks at the different periods of the earth's history and to compare them individually with the corresponding known petroliferous deposits of the same age. But this is a task which, at present, would be beyond our means, by reason of the lack of data. We are reduced to general comparisons, of widely separated periods of time. However incomplete such an examination may be, it offers, in the opinion of the writer, some conclusions of real import.

By far the great majority of the known petroliferous deposits of commercial value belong to the Mesozoic and to the Tertiary eras; or at least the sediments in which they are found date from these ages. Further, there seems to have been a certain degree of constancy, since the close of the Paleozoic, in the location and general trend of the principal areas of deformation of the globe: most of the Tertiary areas of deformation closely follow the trend of Mesozoic synclinals. Finally, the distribution and disposition of the geosynclinal belts of post-Paleozoic age are better known to us than those of the preceding eras. For these reasons, it has seemed advisable to the writer to investigate first the Mesozoic and post-Mesozoic periods.

The map (Fig. 1) shows the results of such an investigation. This map has been established on a Mercator projection (longitudes E. and W. of Paris). The geosynclines are those of the Mesozoic era, and are drawn according to the views of Prof. E. Haug.[12] The dotted part of the outlines are those which this author considers either uncertain or hypothetical. The Mesozoic and Tertiary petroliferous deposits are represented by black dots; the Paleozoic ones by shaded areas. No attempt

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[11] Reginald A. Daly: Introduction to the Geology of the Cordillera. *Geological Survey of Canada, Guide Book No.* 8, 149 and 153. As R. A. Daly observes, this strike recalls the prevailing E. W. to N. 60° E. strikes in the pre-Cambrian rocks of Lake Superior and eastward thereof in the Canadian shield. It also recalls the Huronian system of M. Bertrand, and may indicate a western extension of this pre-Cambrian synclinal network.

[12] *Bulletin de la Société Géologique de France*, 3 Ser. (1900), **28**.

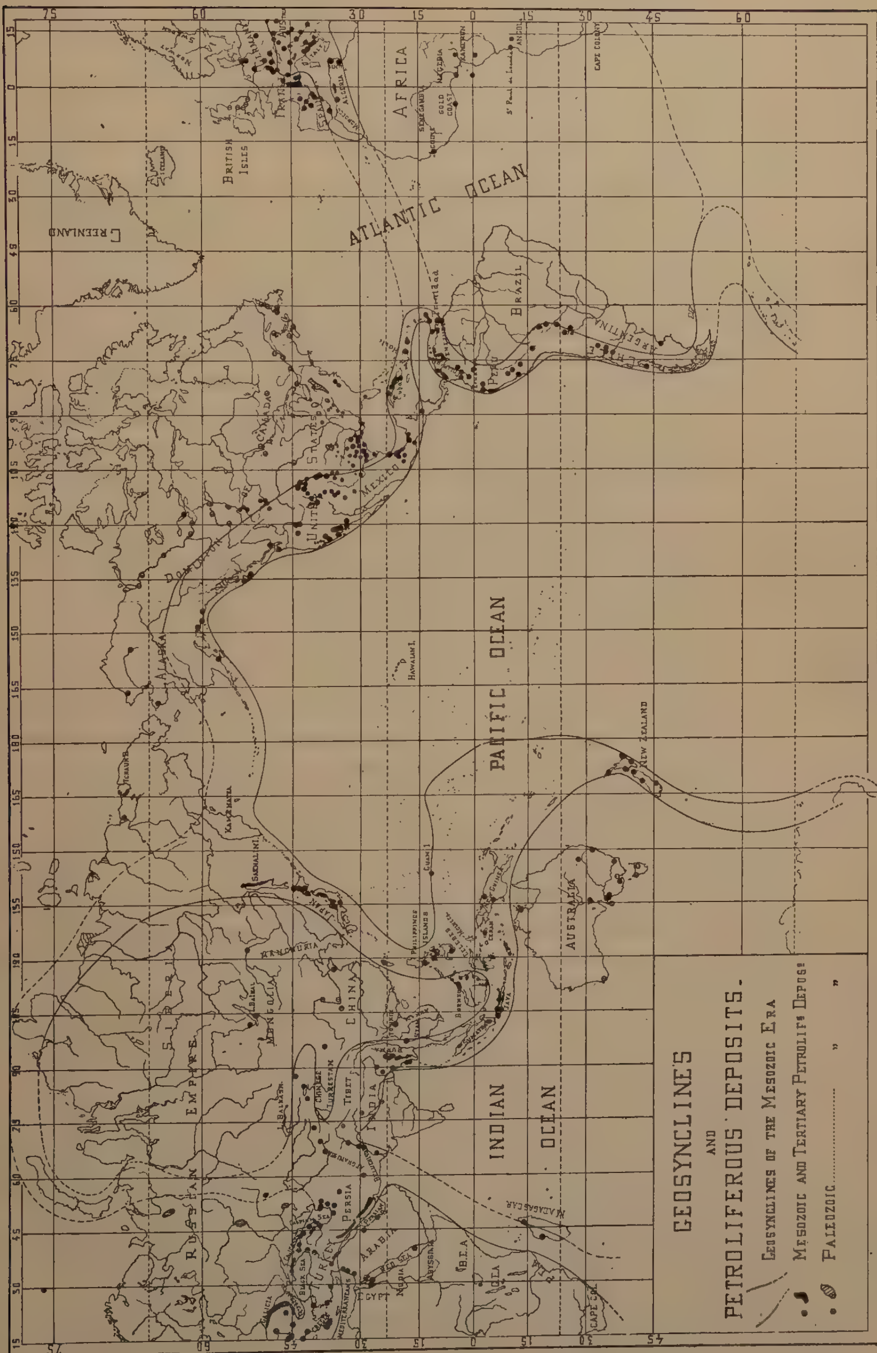
GEOSYNCLINES AND PETROLIFEROUS DEPOSITS.
Geosynclines of the Mesozoic Era
Paleozoic
ATLANTIC OCEAN
PACIFIC OCEAN
INDIAN OCEAN
AFRICA
BRITISH ISLES
AUSTRALIA
NEW ZEALAND

FIG. 1.

has been made to distinguish between the various hydrocarbons nor to set off the respective importance of the deposits themselves. For the location of the deposits, use has been made of the maps published by Sir Boverton Redwood in his treatise on petroleum, and of the publications of the Geological Surveys of the United States, Canada, India, and others available to the writer.

The aim of this map is to present general information rather than to give data correct in every detail. The limits of a geosyncline can hardly be determined with any degree of geometric accuracy. The principal areas of dislocation and folding of a region are defined, but minor folds generally extend at a more or less greater distance outside of their assigned limits. As pointed out by De Launay,[13] "the dislocations of the earth are more and more observed to have taken place not alone in mountainous regions, but even in regions of plains." Further, it is possible that the outlines of some of the geosynclines traced on this map, a part of which are still hypothetical, would have to be corrected. With this reservation in mind, the following observations can be made.

It may be readily seen that the general sequence of the petroliferous deposits closely follows the path of the synclinal network. About 90 per cent. of the deposits are either included between the borders of the synclinal belts or located in their immediate neighborhood. Approximately 10 per cent. only of the total is to be found at some great distance outside. Such are the deposits of the western coast of Africa, of Southern Australia, of the Red Sea, and some isolated and little known or unimportant deposits in Mongolia, Eastern Siberia (Tchaun Peninsula), and Northwestern Alaska.

The relation of the two last groups (Tchaun Pen. and N. W. Alaska) with the geosynclinal branch of the Northern Pacific is uncertain, as the northern limits of this syncline are still in doubt. The other deposits follow secondary lines of deformation, and many of these belong to this class of lines which Suess has termed "disjunctive"; viz., they are breaks that mark out zones of collapse (*effondrements*). Such are the fractures which have determined the curious depression of Lake Baikal, or the remarkable group of faults that extend from Syria, through the valley of the Jordan, the Dead Sea, and the Gulf of Akab, to the cliffs of Abyssinia and hence as far as the neighborhood of Mozambique. These forms are characteristic of disruptive movements having taken place through already consolidated territories, and they seem to imply the break of some fold in the process of formation through grounds that were not plastic enough to allow the completion of the folding. It is remarkable that the hydrocarbon deposits which have been found so far along these secondary lines of distortion are themselves secondary in importance,

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[13] De Launay: *La Science Géologique*, quoted by E. Coste, *Trans.* (1914), **48**, 514.

when compared to many of the deposits that stake out the principal geosynclinal belt.

The result of the above is that petroliferous deposits of Mesozoic and post-Mesozoic ages are essentially distributed on the globe along the lines of dislocation characteristic of the ages, and they become thus apparently connected with the contemporaneous earth movement.

There are many reasons to think that the same conclusion would be reached from the comparison between geosynclines and petroliferous deposits of pre-Mesozoic age. Such a view is strengthened, for instance, by the general disposition of the Paleozoic deposits of North America in regard to the Appalachain belt of deformation, from the basin of the St. Lawrence to the Rocky Mountains, or along the borders of the "Canadian Shield," also by the distribution of the same kind of deposits around the "Russian platform;" and by other facts. This would imply a general law, that we may tentatively express as follows: *The petroliferous deposits of any given period closely follow the principal areas of deformation of this same period.*

This general, apparent relationship between the areas of distortion and the loci of deposition of the hydrocarbons, or between earth movement and petroliferous accumulations, becomes still more precise with the detailed study of certain oil regions. It is to be expected that the simpler the order of deformation, the more distinct the relations will appear. We are thus brought to consider, at first, regions where the structure is easily interpreted, or at least where it has not been obscured by the superposition of a complex series of stresses, varying in age, direction, and intensity. The writer has pointed out such regions as the Eastern Oil Belt of the United States and the belt of the Carpathians in Central Europe, where these conditions are met.[14] In the first instance, the deformation may be interpreted as due to a single dominant thrust; in the second, the thrust would have been repeated, in the same direction, at several distinct periods. The results are represented schematically on Fig. 2 and Fig. 3. Fig. 2 shows the location of the principal oil and gas fields of Pennsylvania and their relation to the Appalachian ridges.[15] Fig. 3 shows the disposition of the principal oil fields of Galicia, and their relation to the trend of the Carpathian range. In the first case, Fig. 2, we have a sequence of simple, broad, and more or less parallel anticlines, merging into monoclines and terraces at the farther end of the distorted area. In the second (Fig. 3), a more complex structure has arisen, due to faults and overthrusts, but it is still anticlinal in its essence and is equally characterized by parallel, concentric lines of

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[14] *Trans.* (1917), **56**, 735.

[15] After C. A. Ashburner and J. P. Leslie: *U. S. Geological Survey* (1900–01), 22*d* *Annual Report*, part 3, 579. The reader is also referred to the general map of the eastern oil and gas fields near the Appalachian Basin, by David B. Reger, *Trans.* (1917), **56**, 858.

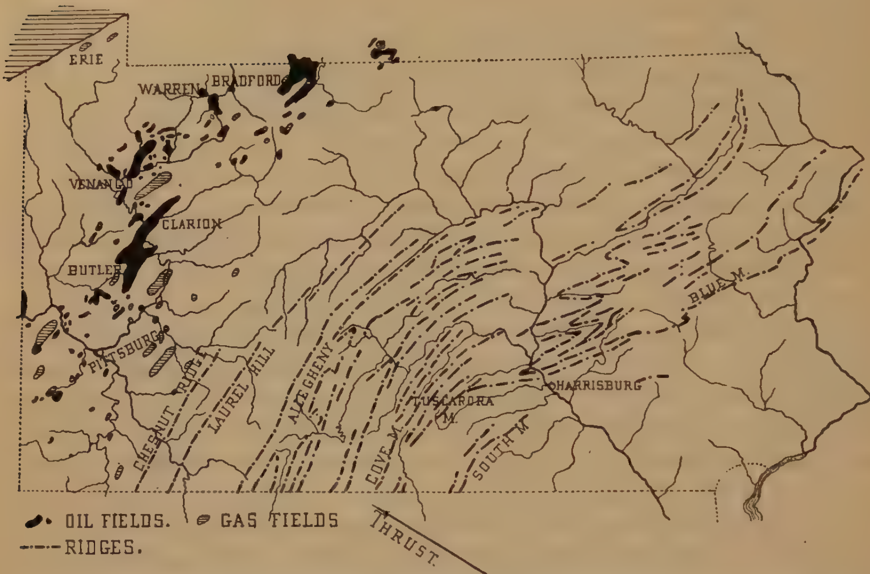

FIG. 2.—DISPOSITION OF THE PRINCIPAL OIL AND GAS FIELDS OF PENNSYLVANIA AND THEIR RELATION TO THE APPALACHIAN RIDGES.

FIG. 3.—DISPOSITION OF THE OIL FIELDS OF GALICIA AND BUKOWINA IN RELATION TO PHYSIOGRAPHIC FEATURES. (AFTER SIR BOVERTON REDWOOD, *Petroleum*, Pl. 2.)

distortion. In either case, the conformity of the trend of the oil deposits with the lines of deformation is impressive, and this unvaried sameness in the effects brings one naturally to infer an identity of causes. If the thrust had, in both instances, at the same time, shoved and folded the strata and concentrated the hydrocarbons along the lines of lesser pressure, the result would not be different. The writer has developed elsewhere the mechanical process through which such a result may have been brought about.[16] It is further to be remarked that a common cause

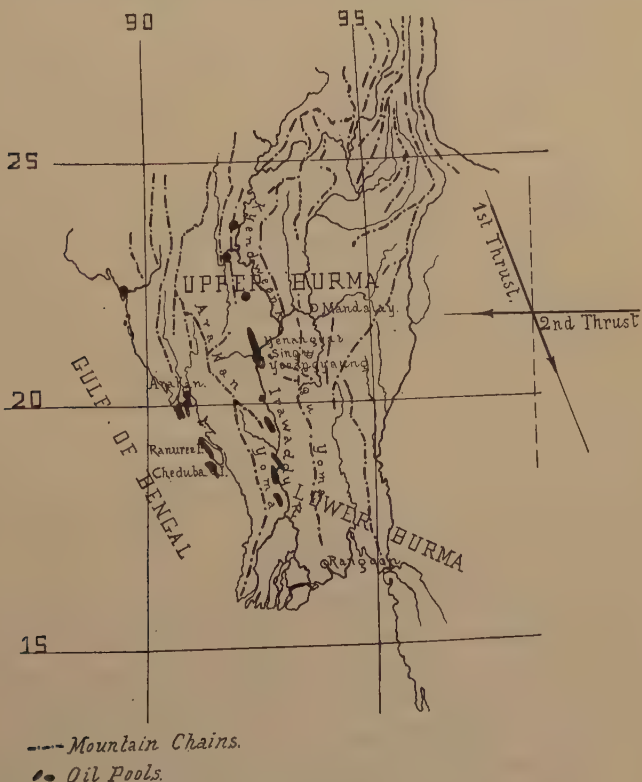

FIG. 4.—OIL FIELDS OF BURMA.

for earth deformation and hydrocarbon accumulation implies the simultaneousness of the two phenomena. Deformation and accumulation would thus be related not only in *space*, but also in *time*. But should this be true, a further conclusion, by no means less important, would follow. If both phenomena, distortion and accumulation, have, so to speak, a common date, then the bulk of the hydrocarbons must have already existed in the strata when the action of the thrust has taken place, or, in other words, the formation of the hydrocarbons must have anteceded the

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[16] *Trans.* (1917), **56,** 735.

earth's distortion resulting from the thrust, and this formation can in no way be connected with fissures and volcanic phenomena of later date.[17]

Let us now consider a somewhat more complex instance. Fig. 4 is a sketch of the disposition of the Burma oil fields, whose location will be found on the map (Fig. 1) near 92° long. E. of Paris and 20° lat. N. The Burma oil fields form two parallel series. The first extends inland along the Kyendween and Lower Irawaddi valleys, between the Pegu-Yoma mountains on the east and the Arakan-Yoma chain on the west; the second follows the eastern shores of the Gulf of Bengal, on the western

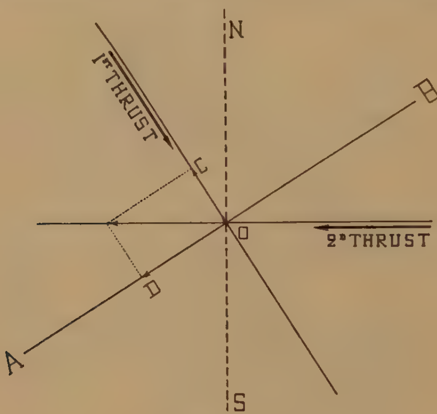

FIG. 5.

side of the Arakan ridge. Both series form a string of isolated deposits, whose course follows the trend of the geosynclinal belt, which runs approximately north-south.

Two flexuring movements have been recognized in this region. An earlier and rather gentle movement caused by a thrust directed from N. 20° W. to S. 20° E., and a later and more important one caused by a thrust directed from east to west. Now, when a series of strata are submitted to two consecutive thrusts coming from two directions inclined at an angle, the first thrust will tend to produce folds normal to its course, like *AB* (Fig. 5). The action of the second thrust is better understood if this force is decomposed into two components: one, *OC*, normal to the fold

[17] This does not imply that during the simultaneous process of distortion and accumulation, or after its completion, some new elements may not have been eventually introduced from below, through crevices or fractures, among the already formed hydrocarbons. This would furnish a key to the presence of helium, for instance, in some natural gas, or may help to explain the peculiar composition of the so-called "connate waters," in other places. But such an action could only be secondary, not primary; local, but not general; and the hydrocarbons, as a whole, could not be genetically related with it. In the same way, it frequently happens that lignites or coals, whose organic origin is undeniable, are locally modified by secondary actions, like dynamometamorphism, or by heat due to the neighborhood of eruptive bodies.

AB already produced, and the other, *OD*, parallel to it. The first component, *OC*, will simply tend to exaggerate the fold *AB*; the second, *OD*, will compress this fold along its axis. The final result will depend on the relative intensity of the two forces that act in succession and on the angle at which they meet. If the two forces are nearly at right angle, which would reduce the component *OC* to a minimum, and if the first thrust is a gentle one, giving rise to large undulations of small relief, while the second is much more powerful, the mean strike lines of final deformation will be in agreement with the second movement. But the resulting folds, instead of being continuous anticlines of great length, will become a succession of shortened anticlines or "elongated domes," separated from each other by depressions corresponding, in a general way, to the synclinal areas of the first flexures. This is precisely what seems to have happened in the Burma region, and the oil fields, the Yenangyung, the Yenangyat-Singu and others, are located on the resulting "domes."[18]

An interesting observation has been made in this connection, that throws some further light upon the relations between deformative movements and hydrocarbon accumulations. ". . . Petroleum," writes E. H. Cunningham-Craig[19] "has never been obtained in paying quantity in any field (of the Burma region) that does not show some traces of the earlier movement, even though these traces are often almost obliterated by the much more powerful later movement. It would seem that there has been a preliminary concentration of the petroleum contents of the strata toward the earlier flexures, which concentration has been greatly increased afterward by the later and greater flexuring." This observation tends to make evident that, in this case, accumulation has taken place in two successive and separate stages, each of these being related to a corresponding distinct period of deformation. During the first stage, a first accumulation of Appalachian type would have been formed, by which the hydrocarbons of the surrounding area would have been concentrated away from the thrust and along the lines of lesser pressure, *i.e.*, along anticlinal folds directed W. 20° S. to E. 20° N. With the intervention of the second thrust, the concentration would have been completed toward the resulting "domes," which, by the laws of mechanics, would represent the final areas of minimum pressure. Thus, once more deformation and accumulation are found to be related in space and in time. Not only would accumulation have taken place where deformation occurred, and only there, but the two phenomena would have been synchronous and the first would have been essentially dependent on the second.

To sum up: It has been shown that petroliferous deposits, when they are considered as a whole, are found to be distributed along the principal

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[18] E. H. Pascoe: The Oil Fields of Burma. *Geological Survey of India, Memoirs* (1912), **40**, 1, Pl. 8, 14, 15 and 19.

[19] *Oil Finding*, 80.

zones of dislocation of the globe, and more, that deposits of a given age follow the trend of the dislocations of this same age. It has been shown also, that in the three particular instances considered (in the Appalachian, the Carpathian, and the Burmese regions), the relations noted between the petroliferous deposits and the general structure of the ground suggest a common cause for both the deformation and the hydrocarbon accumulation. And finally, it has been remarked that this identity of cause would require the simultaneousness of the effects produced, or, in other terms, that deformation and accumulation would have to be understood as synchronous and closely allied phenomena.

If now we remember: that the three instances quoted refer to three of the most important oil regions of the world; that these regions are widely separated, in three different continents (America, Europe, and Asia); and that the deposits themselves date from various ages (from the Paleozoic to the Tertiary); it becomes evident that the constant relations which have been pointed out cannot be interpreted as local or regional, any more than as transient or exceptional. Their permanent character implies a law of a general order.

There are many oil regions in which these same apparent relations are to be found. But there are also many areas in which the mechanism of distortion has been so complicated as to render its interpretation difficult, at least for the present. So that it would be perhaps too soon to try to formulate this probable law in definite terms. But the facts set forth would at least justify the expression of a working hypothesis, that further investigations may amend or supplement. This is that *petroliferous accumulations are generally coincident with diastrophic deformations, synchronous with them, and essentially a result of them.*

There are some other remarks that the inspection of the map (Fig. 1) brings forward. The writer wishes to point to one of these: it is the peculiar position occupied by certain oil regions in regard to the trend of the synclinal belt of deformation, such as the region of Borneo, and the area comprising the northeastern coast of Mexico and the Gulf coastal plain region of Texas and Louisiana, farther north. Both regions are located at points where the belt is bent abruptly, and on the inside or concave portion of the curve. Such regions would probably be submitted to peculiar stresses. Besides flexuring by compression, there would be a tendency to deformation by torsion. Daubrée's experiments have shown that torsion may give rise to a system of conjugated faults, equally inclined on the principal axis of torsion. It is significant that if, on one hand, the lines of fractures, resulting in igneous dikes and intrusions, of the Mexican oil fields,[20] and, on the other hand, the faults of the Gulf coastal plain region (whether ascertained faults or hypothetical faults

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[20] Huntley: *Trans.* (1915), **52**, 302, Fig. 2; and 310, Fig. 6.

according to the views of Harris),[21] are recorded on the map, they present essentially the appearance of a continuous system of the kind. What may have been the influence of such deformation, combined with deformations due to other causes, on the accumulation of the oil, is still to be seen. But that some kind of relation would exist, is probable in the opinion of the writer. Should this point ever be investigated, it would be prudent to remember that the two groups of conjugated lines (right and left) of Daubrée's experiments are not identical in value. When a

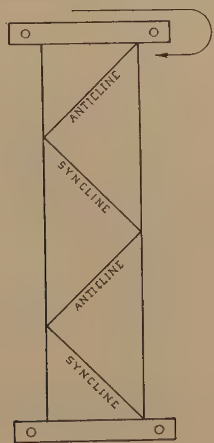

FIG. 6.

flexible sheet, instead of the rigid glass plate used by Daubrée, is submitted to torsion, the motion following the direction of the hands of the clock, the lines of deformation will appear as indicated in Fig. 6. One set is concave (synclinal), the other is convex (anticlinal). If the torsional movement is reversed, the lines will be symmetrically reversed also. Thus, torsion may give rise to flexures; and wherever it produces faults, one set of these would have a tendency to open from above and the other set from below, the maximum relief of stresses being found at the crossing points of the two sets.

DISCUSSION

W. VAN DER GRACHT, Tulsa, Okla. (written discussion*).—I fully agree with Mr. Daly's observation that there is such a coincidence between petroleum deposits and geosynclines as to suggest them to be "closely allied phenomena." However, I accept no "identity of cause." To me the explanation is a simple one.

[21] G. D. Harris: *U. S. Geological Survey Bulletin* No. 429.

* Received Oct. 10, 1917.

Geosynclinal deposition is the first phase in the formation of a mountain chain, and petroliferous deposits are indeed closely allied to zones of mountain folding. In fact, petroleum is a typical occurrence in those beds, which represent the *flysch facies*, to use the Swiss expression. The uplifting of a great mountain chain is a very slow process; it gradually sets in as a slow geo-anticlinal upheaval within a zone of former geo-synclinal deposition. This uplift is immediately attacked by intense erosion as its crest rises out of the geo-synclinal sea, and this erosion causes intensive deposition of lagunary and shallow-water deposits in the adjacent sea, in front of the push, where the bottom continues to sink. As a rule, this deposition keeps up fairly well with the subsidence, maintaining very shallow-water marine conditions alternating with brackish and fresh-water conditions. This goes on during a long time, causing a great thickness of such deposits as shales with numerous sandstones and limestones. This is the typical *flysch facies*, *i.e.*, shallow, quiet-water, marine deposits, alternating with lagunary, sometimes saline, and even fresh-water deposits, the latter often containing numerous coal beds. *Such deposits contain all the large coal-bearing as well as oil-bearing areas of the world.* Wherever there were swamps capable of maintaining heavy vegetation, great bog deposits were laid down, which, after they had been buried by a renewed subsidence, were fossilized into coal beds. Further out, in the shallow sea, often behind sand bars, great masses of plancton accumulated, followed naturally by an abundance of other marine life, which lived on this plancton, and by others which in turn preyed on the first. Thus, a mud that was largely organic was laid down, which gradually was to become an oil shale—the primary container of petroleum.

All this, as I said, happened in the early stage of the formation of a mountain chain. If conditions remained thus, we got a coal field or deposit of oil shale, but no oil field. If, however, the mountain-building forces continued, and the final phase of great folding was consummated, the *flysch* beds were thrown into anticlinal and synclinal folds; the primary oil moved out of the disturbed oil shales and, if an adequately porous container were present, accumulated in the tops of the anticlines, making an oil field. These naturally are found mostly on the front edge of the original geosynclines, because in the heart of the mountains the *flysch* deposits—if ever they were formed—have been destroyed by the overthrusting and sharp folding and subsequent very intensive erosion. Only on the outer edge of the zone of folding, the *flysch* anticlines have had a chance to be preserved as oil containers.

Thus geosynclines and oil fields are indeed "allied phenomena" but there is "no identity of cause." They are really "coincident" with diastrophic deformations and more or less "synchronous," but synchronous only within a certain phase—the formation of bituminous matter in

its primary form is coincident only with the earliest phase of the uplift of a mountain chain, the accumulation with the final phase of acute actual folding.

Thus, to cite a few instances, the Carboniferous oil is mainly Pennsylvanian all over the world, in the *flysch* deposited before the great late Carboniferous, early Permian final folding; the Cordillerian petroleum of America is Upper Cretaceous, again in the *flysch* deposited before the final folding of the Rocky Mountains and the Andes in the early Eocene. These examples could be repeated indefinitely for all mountain chains, and they will more or less clearly explain almost every larger oil field the world over.

Frederick G. Clapp, New York, N. Y. (written discussion*).—We have all agreed for years with Mr. Daly's italicized statement on page 1064, "that petroliferous accumulations are generally coincident with diastrophic deformations, synchronous with them, and essentially a result of them." In other words, oil pools correspond with localities of suitable structure, into which the oil was segregated on account of the doming or other structural processes. This is the structural theory, pure and simple, and does not commit us further than this to the so-called diastrophic theory.

As to the paper itself; why might it not be equally well entitled "*Geanticlines* and Petroliferous Deposits?" Certainly oil occurs on more geanticlines than geosynclines; and the axes mapped on Fig. 1 of the paper are those of geanticlines.

It seems to me, without wishing to criticize the thought and care bestowed on the paper, that the matter has been unnecessarily complicated in its exposition; and that his enthusiasm may have led the author far beyond his better judgment. In studying Fig. 1, for example, we find that in Virginia, Florida, China, Newfoundland, France, the British Isles, and many other localities, petroliferous deposits are represented which are in some cases mythical and in others proved to be of no value, being justified only by their previous appearance on Boverton Redwood's published map. Some great oil fields like those of Oklahoma, Illinois, Louisiana, Pennsylvania, etc., are situated hundreds or thousands of miles from the "geosynclines" (geanticlines) as mapped in other parts of this plate.

The author's statement, of course, is true that in most oil fields two sets of thrusts seem to have taken place. This is especially noticeable in Oklahoma and Wyoming, where what are known as "cross anticlines" are frequent. These may be due to the succession of two or more sets of forces operating in different directions, to torsion, as explained on page 1065, or to some sort of thrust from below, quite different from lateral

* Received Oct. 8, 1917.

causes, as in the Louisiana, Transylvanian, some Mexican, and the Roumanian fields. But to the modern petroleum geologist, cause is of little importance; it is the presence of suitable structure, underlain by porous sands and accompanied by other favorable conditions, that are important.

Marcel R. Daly (written discussion*).—I have read with great interest Mr. W. van der Gracht's discussion of my paper and I fail to see where his theory contradicts mine on any fundamental point. My contention is that petroliferous accumulations are generally coincident with diastrophic deformations, synchronous with them, and essentially a result of them. This implies that the causes which have led to the distortion of the strata, whatever these causes may be (thrusts, torsion, etc.), have also led to the concentration and accumulation of the hydrocarbons existing in the strata at that time. It is in this respect that both sets of phenomena may be said to be related in space and in time and that we may speak of an identity of causes. It is evident that no concentration of the hydrocarbons could take place where these hydrocarbons would not exist. But whenever they do so exist, deformation will tend to concentrate them; and whenever they are found to be concentrated (or accumulated) the cause of their accumulation will have to be traced directly to distortion. I have tried to show, in some previous papers,[1] the probable sequence of such a mechanical process in the simple case of a series of parallel, superposed, horizontal, and homogeneous layers, individually uniform in thickness, but variable in composition and resistance; and I have pointed out that the general results so obtained would harmonize with some striking features recognized in the field. But I have also stated that conditions being much more complicated in nature than in the theoretical instance considered above, a complex superposition of effects would have to be expected as a general rule, though the leading principle would remain the same.

Mr. van der Gracht admits that the motion of the primary oil, when it has been formed, and its accumulation in adequately porous containers, if any, that would be mostly found on the front edge of the original geosynclines, would be due to the action of the mountain-building forces. Our agreement on this point seems to be complete. But Mr. van der Gracht wants also the formation of the bituminous matter in its primary form to be coincident only with the earliest phase of the uplift, *viz.*, after the slow geanticlinal upheaval has been completed, and the accumulation of oil to concur only with the final phase of acute actual folding. It seems to me that this conception covers only a special case. There is no reason

* Received Jan. 29, 1918.

[1] Diastrophic Theory. *Trans.* (1916), **56**, 733, 760. Geosynclines and Petroliferous Deposits is a sequel.

a priori for denying the possibility of the accumulation of the parent matter from which oil proceeds, on the rims of the geosyncline itself, during the period of erosion and deposition that brings about its filling. Mr. van der Gracht's theory is probaby true for the fields of Central Europe, for instance, where the "flysch facies" is so characteristic; but it is difficult to see how it could be applied in some other cases, for instance to the Burma fields, where the oil formations are deltaic.[2] Here, as pointed out by the writer, two successive flexuring movements have been recognized, and it has been shown[3] that a first accumulation of the oil has taken place contemporaneously with the first set of flexures, the direction of which is *normal* to the general geosynclinal trend. The consequence is that the second flexuring movement, precisely the one that would create waves parallel to the geosynclinal axis, must have acted on the already partially accumulated hydrocarbons during its entire duration, not merely at its close.

Mr. van der Gracht seems to consider an oil shale, in a general way, as the primary container of petroleum. I think a distinction is to be made between an oil-bearing shale, where the primary oil may have remained in a state of dissemination, by reason of some interruption in the folding process or even by the lack of it, and an oil shale proper, or "kerogen shale," from which shale oil is extracted, as in the Lothians, New South Wales, and elsewhere. In a recent paper, to which I have had already occasion to refer,[4] Cunningham-Craig has clearly shown that the bituminous matter contained in a "kerogen shale" is the result of the inspissation of an already concentrated petroleum and of its further adsorption by argillaceous material. The "kerogen" phase would thus represent a later phase than the oil phase, and kerogen-shale fields would be the descendants of oil fields and not their forefathers. The distinction is economically important.

Answering Mr. Frederick G. Clapp: The writer feels gratified that Mr. Clapp should agree with his statement that "petroliferous accumulations are coincident with diastrophic deformations, synchronous with them, and essentially a result of them," but unfortunately Mr. Clapp does not give to this statement the same interpretation as the writer himself, when he adds: "In other words, oil pools correspond with localities of suitable structure, into which the oil was segregated on account of the doming or other structural processes." This may answer to the first part of the proposition, which implies the coincidence of accumulation with certain

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[2] "The Pegu Series of Burma, ranging from the Eocene far up into the Miocene according to our subdivisions of Tertiary time, furnishes perhaps the most conclusive evidence of the advance of a delta that has been worked out in any detail" (C. Craig: *Oil Finding*, 63).

[3] See p. 1063.

[4] See p. 1054.

structural forms, but it omits the two factors of synchronization and causality, or at least it throws in doubt their intervention. If the writer's formula is accepted it ought to be accepted with its full meaning, and the logical consequences of these premises would have to be accepted at the same time. It has been the aim of the writer, in his "Diastrophic Theory," to show what these consequences may be.

The paper has been entitled "Geosynclines and Petroliferous Deposits" for the reason that the writer studies in this paper the mutual relations between the loci of deposition of the hydrocarbons and the "zones of weakness and mobility of the earth," geosynclines by definition (p. 1055). The geosynclines of the Mesozoic Era, represented on Fig. 1, have been drawn according to the views of Prof. E. Haug, of Paris, one of our leading European geologists (p. 1056).

The deposits mapped on Fig. 1 do not represent workable oil fields exclusively. The writer has taken great care to specify that, in the paper as well as on the map, no attempt was made to distinguish between the different members of the petroleum family, which are treated as a whole (Petroliferous Deposits), nor to set off the respective importance of the deposits themselves (pp. 1054, 1055, 1058). Hence, the remark on the lack of economic value of some of these deposits does not apply. Economic importance alone is not a sufficient criterion in determining the geological interest of a landmark.

Coming to the question of distance between the great oil fields of Pennsylvania, Illinois, and Oklahoma, and the geosynclines mapped on Fig. 1, the writer refers Mr. Clapp to the legend inscribed on the plate and to the description of the latter (pp. 1056–58). The geosynclines represented are those of the Mesozoic Era, and Paleozoic deposits would have no necessary connection with them. These Paleozoic deposits would be related to a different system or "geosynclinal network," and, for this reason, their distance from any other system is immaterial. This is precisely one of the conclusions represented by the writer (p. 1059, italicized statement). As for the Louisiana field, the writer has himself drawn attention to its peculiar position, and has advanced a hypothesis to explain it (pp. 1064–65).

Finally, the writer begs to take exception to the remark that "to the modern petroleum geologist, cause is of little importance." If the search of causes is of no value, then Newton would have been satisfied that the apple dropped from the tree because it wanted to. I wonder what would have become of our modern art of mining if it had been left, without any further research of causes, in the shape in which Agricola found it in the sixteenth century, when he was writing his treatise "De Re Metallica."

Funnel and Anticlinal-ring Structure Associated with Igneous Intrusions in the Mexican Oil Fields

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(St. Louis Meeting, October, 1917)

Introduction

For a number of years the senior author has been interested in the various geologic and engineering problems involved in the development of the petroliferous districts of northeastern Mexico, having in a previous paper[1] endeavored to present a summarized statement of the information relating to the geology and soil resources of this region.

Of the many interesting geologic phenomena, perhaps those which have been the object of most speculation relate to the igneous intrusions which, in places, have a controlling influence on the accumulation of oil in commercial quantities, a relationship which has been tentatively explained by various observers with as many interestingly different views.

The authors of the present paper aim to record, in the course of a general discussion of the various viewpoints, some additional data, and present further tentative conclusions with a view to rounding out an up-to-date summary of the subject.

The greater part of the area in question lies in the State of Vera Cruz, only a northwest fraction being in the State of San Luis Potosi. Vera Cruz is bounded on the north by the State of Tamaulipas, the northernmost of the Mexican States along the Gulf.

The topography of the States of Tamaulipas and Vera Cruz and of Texas and Louisiana to the north, is controlled by the Gulf coastal plain which in northern Vera Cruz has an average width of about 60 miles, the transitional topography between that of the rugged flanks of the Sierra and the lowlands of the coast being made up of a series of terraces and irregular hills and valleys.

Geology and Structure

The greater part of the mainland of Mexico is made up of Tertiary and later effusive rocks and of sedimentaries of Cretaceous age. The igneous rocks form a belt from 150 to 300 miles wide, spurs of which skirt

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* Asst. Professor, Petroleum Engineering, Stanford University.

[1] Bibliography, No. 18.

the coast of the northwestern State of Sonora and after receding from the coast continue to the southeast, crossing the central part of the republic in a more easterly trend along the general latitude of Mexico City. This igneous belt, which attains its greatest development in the Sierra Tarahumare in Chihuahua, the Sierra Madre in the States of Durango and Zacatecas, and in the central States to the north of Mexico City, forms the high Mesa Central (central plateau) which has an average elevation between 5000 and 8000 ft. (1524 and 2438 m.). North and east of this, along the plateau and down the steep decline to the Gulf, extends the larger of the Cretaceous areas which is folded and contorted along the eastern escarpment of the Sierra Madre Oriental, and reaches the coast as a gentle low-dipping monocline covered in places with Tertiary and a thin veneer of Quaternary formations. The petroliferous zone is situated on this monocline which forms the coastal plain along the Gulf of Mexico, its geologic features being modified locally by spurs of the belt of effusive rocks to the west.

The formations involved in the geology of the oil region, with the tentative classification adopted in the present paper, are:

1. *Lower Cretaceous limestones*, comprising a series with an approximate aggregate thickness of 10,000 ft. (3048 m.).

2. *Upper Cretaceous limestones and shales*, having an estimated thickness of 500 ft.[2] (152.4 m.).

3. *Cretaceous-Eocene shales*, about 3000 ft. (914 m.) thick.[3]

4. *Later Tertiary limestones, sands, and clays*, about 2000 ft. (609 m.) in thickness, of Oligocene and later age.

5. *Igneous intrusions* of late Tertiary or possibly early Quaternary.

6. A thin cover of Quaternary and recent deposits.

A general idea of the areal distribution of the geologic formations and the location of the principal igneous intrusions is given in a map by Geoffrey Jeffreys recently published.[4]

The Igneous Intrusions

General Statement

One of the characteristic features of the Mexican oil fields is the great number of igneous masses which dot the region, sometimes in the form of

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[2] The existence of an unconformity at the base of the San Felipe series, which was hinted at in previous papers, has been further confirmed by the observations of C. W. Washburne. Unconformities very likely exist also at the bases of the Mendez and San Fernando series.

[3] Later investigation of the fossil fauna, found in the sandstones at the top of the Cretaceous-Eocene shales in the vicinity of Alanzan, shows that they bear a very marked relation to the Eocene of California, with which they had been tentatively correlated.

[4] Bibliography, No. 21, p. 302.

conical peaks, breaking the monotony of the level plain. The greater number of these cones and associated dikes occur in the southern part, attaining their greatest development in the Otontepec range which covers an approximate area of 35 square miles (90.65 km.^{2}) and attains a height of about 3500 ft. (1067 m.), decreasing in frequency towards the east and north as the distance from the Sierra Madre increases and disappearing entirely before the Rio Grande is reached.

According to De Golyer,[5] the area of greatest igneous activity, roughly triangular in outline, is defined by a north-south line through Furbero, a second line through Tantoyuca and Ozuluama, and a third line along the front of the Sierra Madre Oriental. Within this area are located the productive fields of Casiano, Potrero del Llano, Dos Bocas, Alamo, Los

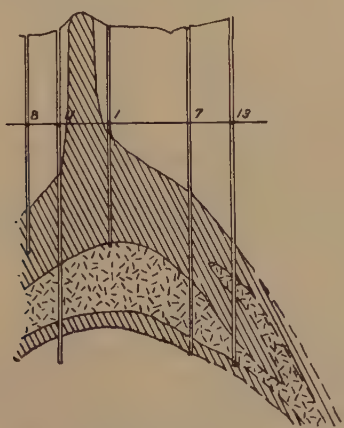

Fig. 1.—Section across a portion of the Furbero sill, showing underground structure. The cross-hatched area above and below the basalt sill indicates the zone of metamorphosed shale. (After De Golyer.)

Naranjos and others, while the most important districts so far discovered outside its boundaries are Ebano, Topila, and Panuco in the northern region between the Tamesi and Panuco Rivers.

Most of the surface indications of petroleum are closely associated with the exposed intrusions, such being the case at La Pez, Chijol, San Geronimo, La Merced, Rancho Abajo, Monte Alto, Los Higueros, Dos Bocas, Casiano, Cervantes, Tres Hermanos, Tinaja, Ojo de Brea, Chapopotillo, Monte Grande, Moralillo, Cerro Azul, Juan Felipe, Las Borrachas, Piedra Labrada, Cerro Viejo, Potrero del Llano, Tierra Amarilla, and Chapopote Nunez. As it is evident that not all of the intrusions reached the surface, some of the surface "showings" of petroleum near which no intrusive masses are visible may be closely associated with the underlying igneous masses.

[5] Bibliography, No. 23, p. 658.

An intermediate step between the exposed and buried intrusions may be represented by outcropping masses of metamorphosed shale, such as the sharp twin peaks of Furbero (see Fig. 1), the main elevation at Cerro Azul, the Cerro de Zaragosa cone, and the peak at Taninul, all of which resemble basaltic cones in outline and likely are representative surface phenomena of the underlying intrusions. Huntley[6] in discussing these topographic features and their association with hidden intrusions, cites as an example Cerro Taninul of which he says:

> "The western one is composed of basalt, while the smaller eastern peak (Taninul) shows only large blocks of a metamorphosed marl containing numerous small siliceous nodules. This peculiar rock is found at a number of places in the vicinity of large intrusions of basalt and has been caused by the circulation of underground currents, probably both hot and more or less mineralized, as pyrite is frequently found nearby."

The senior author has observed a similar phenomenon along the base of Cerro Azul.

Form of Intrusions

It is interesting to record in this connection the different views advanced by various geologists regarding the form of the intrusions: Villarello[7] believes that some of the hills, like that of La Pez, are in reality small cones or craters through which took place the flow of basaltic tuffs which extend a little over the plain; Ordonez[8] and Coste[9] hold the view that the volcanic disturbances were mostly of the explosive type drilling upward through the sedimentaries and opening narrow pipes, these explanations conforming in part with the views of Harker,[10] who states that small masses of molten material, by a process of "overhead-stoping," are able to form almost cylindrical pipes through the intruded rocks.

F. G. Clapp and I. C. White[11] ascribed to the intrusion the form of a more or less irregular cone in a normal position, the vertex of which may or may not reach the surface. This theory, however, is no longer tenable in view of the results of actual experience, as many of the producing wells like those of Ebano are located within "stone's throw" of exposed intrusive plugs, and in other cases the basalt has been penetrated for several feet and the well continued through sedimentaries to the oil reservoir below, conditions which could not obtain were the "normal cone" theory correct. In order to account for this discrepancy, the senior author performed in 1912 a rather crude experiment[12] from which one of the tentative conclusions drawn was that the thickness of the basalt

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[6] Bibliography, No. 21, p. 312.
[7] Bibliography, No. 6, pp. 39–40.
[8] Bibliography, No. 10, p. 1020.
[9] Bibliography, No. 15, p. 504.
[10] Bibliography, No. 8, p. 86.
[11] Bibliography, No. 12, pp. 277–278.
[12] Bibliography, No. 13.

intrusions increases rather than decreases from its deep-seated source to the surface, thus forming an inverted irregular cone, or more nearly a "mushroomlike" plug with projecting irregular annular tongues along the bedding planes of the disturbed intruded beds.

Sills and other forms of intrusions which do not outcrop, like the underlying sill at Furbero, may be formed, as suggested by De Golyer,[13] by an igneous mass ascending in the form of a dike which on encountering a steeply dipping bedding plane of little resistance, is diverted into that plane as a sill or sheet. The statement of Dumble[14] that the basaltic material in the producing fields occurs mostly in the form of sills or intercalated sheets of moderate thickness, would indicate that this is a prevalent rather than an exceptional occurrence.

Although some of the earlier investigators[15] claimed that the occurrence of dikes was practically unknown in the Mexican fields, later work disclosed the fact that dikes are quite common in this region; Jeffreys, who made a thorough study of them, placing considerable stress upon their importance as factors in the accumulation of oil.

Petrology of Intrusions

The material which makes up the intrusions has been generally classified as basalt accompanied, as at La Pez and La Dicha, by volcanic ash and agglomerate.

Ordonez[16] reports some of the pipes made up of basaltic lava, sometimes olivine basalt, and others basaltic tuff; both Clapp[17] and Coste[18] state that the volcanic necks are composed chiefly of olivine basalt, De Golyer[19] who has observed the occurrences in the southern districts stating that:

> "Many of the plugs, such as Cerros Palma Real and Pelon, consist altogether of volcanic breccia composed in part of sedimentary rocks, while others consist of a relatively small chimney of basalt surrounded by a rather wide aureole of brecciated material."

The breccia which is found associated with intrusions is composed largely of fragments of limestones and shales carried in suspension in the semi-viscous igneous mass, metamorphosed and cemented upon the solidification of the molten material. It seems very probable that many of the hills, which are composed in most part of such brecciated rocks under

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13 Bibliography, No. 20, p. 278.
14 Bibliography, No. 19, p. 254.
15 Bibliography, No. 6, p. 40 and Bibliography, No. 17, p. 860.
16 Bibliography, No. 5, p. 247.
17 Bibliography, No. 12, p. 379.
18 Bibliography, No. 15, p. 510.
19 Bibliography, No. 23, p. 659.

varying degrees of alteration, may give place deeper down to the igneous mass which has caused the brecciation and cementation of the fragmentary material.

A microscopic study of the igneous material has been made by Huntley,[20] who published some photomicrographs of thin sections of basalt collected from dikes in the central districts; he reports the presence of labradorite, olivine, augite, and magnetite in a rock which he classes as olivine dolerite.

It will seem, therefore, that lacking a more thorough petrographic study, the intrusive rocks may be classed as belonging to the basalt group consisting for the most part of basalts, dolerites, and diabase, sometimes with vesicular structure and accompanied by breccia, volcanic ash, and agglomerate.

Relationship of Intrusions to Regional Structure

The period of vulcanism in this region was subsequent to that of folding, and the areal distribution of dikes and plugs indicates that the molten material in seeking a relief of pressure arose to the surface, or to higher planes of equalized pressure within the sedimentaries, mainly along fissures or planes of weakness in the overlying beds, the general arrangement of the dikes and the existence of isolated plugs apparently along definite lines suggesting the existence of well-defined fractures; Jeffreys noted that although the intrusive masses occur throughout the coastal plain, there seems to be a very pronounced zone of vulcanism running northeast-southwest, passing from Chicontepec to Tantima, Chapopote, and thence to Dos Bocas.

Huntley[21] ascribes the origin of these fractures and planes of weaknesses to the main intrusion of Sierra Otontepec, and the location of the radiating and tangential dikes in the immediate neighborhood, he claims, may be due to secondary fracturing caused by this large intrusion and its subsequent shrinking. De Golyer[22] advances the explanation that the intrusions took place through faults and fissures developed in the rocks by shearing stresses and strains resulting from conflict in the forces which folded the rocks originally.

It may be noted that the plugs whose location is controlled by structural lines of weaknesses present some analogy to the saline domes in the oil fields of the Texas Louisiana coastal plain.

Localized Effect of the Intrusions

The intruding molten material produced a certain amount of metamorphism in the surrounding sedimentaries, both as a direct effect of its

[20] Bibliography, No. 21, p. 281.
[21] Bibliography, No. 21, p. 281.
[22] Bibliography, No. 23, p. 658.

high temperature and as a result of the hydrothermal alteration caused by the circulation of heated underground waters. The metamorphic action on the shales, accompanied by an increase in density and diminution of volume, likely gave rise to a considerable amount of brecciation upon the consolidation and contraction of the molten mass, and this resulted in the formation of an aureole of material more porous than the surrounding unaltered beds.

The metamorphic action on the limestones in the immediate vicinity of the intrusions probably caused the formation of cavities as a result of dolomitization, and the igneous rock itself was probably altered by the hot circulating waters, with the probable increase of the void space; a specimen of the basalt which makes up the Furbero sill showing, according to De Golyer and Norman,[23] a porosity of 6 per cent.

The localized effect of the intrusions on the structure of the surrounding sedimentaries presents some uniform features with minor differentia-

FIG. 2.—SECTION ACROSS THE BINN OF BURNTISLAND, FIRTH OF FORTH, SCOTLAND.

tion according to the preëxisting structure, the form of the intrusive mass, and its position above or below ground.

It is reasonable to suppose that the intrusions that do not reach the surface, such as the sill that controls the accumulation of oil in the Furbero field (see Fig. 1), irrespective of their form and preëxisting structure, have produced, as a rule, a certain amount of upward folding of the overlying beds, thus initiating or accentuating a domical structure. It should be noted, however, that minor features of the resulting structure will be controlled to some extent by the attitude of the intruded beds before the intrusion took place.

The structural features developed by basalt dikes, pipes, and necks which reach the surface, were studied by Heaphy[24] as early as 1860, and further corroborated by Geikie, 20 years later,[25] these investigators observing that the beds near basalt intrusions dip rather steeply toward the intrusions (see Fig. 2 and 3), a phenomenon which later has been observed in the Mexican oil region by Jeffreys, De Golyer,[26] Stewart,[27] and others.

[23] Bibliography, No. 20, p. 268.
[24] Bibliography, No. 1, p. 244.
[25] Bibliography, No. 3, pp. 468–470.
[26] Bibliography, No. 23, p. 659.
[27] Bibliography, No. 22, p. 11.

The inclination of the strata toward the intrusions is accounted for by the settling or "drag-back" of the igneous material while cooling, due to its contraction upon solidification; a phenomenon which also explains the extensive amount of brecciation in the indurated shales surrounding the plugs. The amount of shrinkage of molten magmas under these conditions has been estimated to reach as high as 10 or 11 per cent. Barus[28] found that diabase undergoes the following contraction upon cooling from 1,500° C. to 0° C.: from 1,500° C. to 1,093° C., 3 per cent.; upon solidification at 1,093° C., 3.55 to 4 per cent.; from 1,093° to 0° C., 4 per cent. This phenomenon would readily account for the downward drag of the plug with the attendant brecciation and "funnelling" of the surrounding sediments.

In order to allow the necessary space for the invading magma or breccia, and irrespective of the preëxisting structure, a certain amount of upward folding had to take place while the intrusion was in progress,

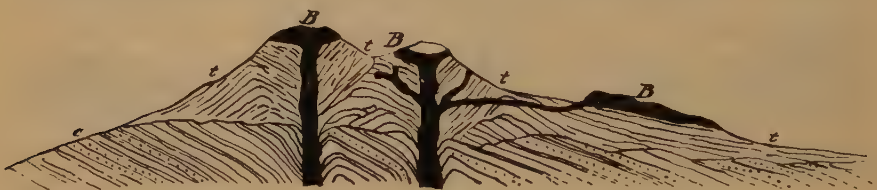

FIG. 3.—SECTION ACROSS LARGO LAW, SCOTLAND. *B*, BASALT; *t*, VOLCANIC TUFF; *c*, CARBONIFEROUS STRATA. (AFTER GEIKIE.)

followed during the cooling and settling stage by the dragging down of the shattered edges of the displaced beds, to an inclined or even vertical position in the immediate vicinity of the intrusion, thus giving rise to an outer annular anticlinal fold and an inner concentric funnel depression immediately surrounding the igneous mass, the anticlinal-ring-and-funnel structure thus developed being made up of a series of minor complex asymmetric folds.

Observations on Similar Phenomena in Other Parts of the World

The phenomena of the inward dip of beds surrounding a volcanic neck is by no means uncommon, the earliest observations, as mentioned before, being recorded by Heaphy[29] in a description of the volcanic country of Auckland, New Zealand. At a later date, Geikie cites[30] the occurrence in Central Scotland, especially in the basin of the Firth of Forth, of volcanic necks, dikes, and sheets which resemble very much those of the Mexican fields in topographic expression, composition, and structural

[28] Bibliography, No. 4, pp. 36–39.
[29] Bibliography, No. 1, p. 244.
[30] Bibliography, No. 3, pp. 459–461.

relations. Here also the material filling up the vents consists of molten rocks which have flowed through the neck and solidified, and breccia or volcanic detritus made up of fragments of the intruded strata, blocks of the intrusive material itself, and tuffaceous material, the whole firmly cemented by either the intrusive lava or secondary alteration products, the igneous material making up the intrusions being usually basalt, dolerite, or porphyrite.

In discussing the structure of the beds adjacent to these intrusive pipes, Geikie writes:[31]

"In the majority of the shore-sections, a remarkable change of dip is observable among the strata round the edge of each vent. No matter what may be the normal dip of the locality, *the beds are bent sharply down toward the wall of the neck*, and frequently placed on end. . . . . This is precisely the reverse of what might have been anticipated, and can hardly be due to the upward volcanic explosions. It is usually associated with considerable metamorphism of the disturbed strata.

"Taking it in connection with the metamorphism, I am inclined to believe that it took place after the long-continued volcanic action which had hardened the rocks round the volcanic pipe, had ceased, and as a result of some kind of subsidence within the vent. So general is this evidence of downward movement in all the volcanic districts where the necks have been adequately exposed, that *this structure may be suspected to be normal to all old volcanic vents*."

Du Toit[32] observed a similar structure in close proximity to the kimberlite dikes and necks in the Kimberley diamond fields of South Africa and analogous occurrences are reported, by Dr. J. C. Branner, from Panama, where numerous basalt necks, some of which send out branches into the surrounding beds, are found cutting the sedimentaries along the Canal. Concerning the shape of the intrusion which forms Gold Hill, General Goethals writes:[33]

"Gold Hill is of basalt, thrown up in a molten state through the sedimentary deposits that already existed, and poured over the deposits on either side of the stem, giving to the vertical section, the general shape of a mushroom."

The Intrusions and the Oil Fields

Relation of the Intrusions to the Origin of Petroleum

The association of the Mexican oil fields with igneous activity has been advanced by Coste as an argument in favor of the inorganic origin of petroleum; thus, in discussing the relation between the occurrence of oil and igneous intrusions, he writes:[34]

". . . it is manifest that these Mexican petroleum deposits are directly connected with vulcanism, *and due* to solfataric volcanic emanations accompanying the upheaval of the basaltic cones."

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[31] Bibliography, No. 3, pp. 469–470.
[32] Bibliography, No. 9, pp. 114–115.
[33] Bibliography, No. 27, p. 30.
[34] Bibliography, No. 15, p. 504.

The geologists who advocate the organic origin of the Mexican petroleum believe that the source of the oil may be ascribed to the original organic matter in what is now the Lower Cretaceous limestone[35] or that entombed in the overlying shales, leaving to the direct action of the heat of the molten intrusions the secondary role of partially distilling the comparatively small amount of organic material which came within the influences of the heated masses.[36]

Effect of the Intrusions on the Migration and Accumulation of Oil

Although the relation between the origin of petroleum and the intrusions is a matter of conjecture, the influence of the intrusions in the migration and accumulation of the oil should not be underestimated. In the first place, the contact zone of metamorphosed and brecciated sedimentaries with their extensive fractures provides a system of channels for the migration of the oil in the Lower Cretaceous limestones to storage reservoirs in higher planes, or to the surface if the channels break the exposed beds. Furthermore, the brecciated zones, made up partly of well-like channels, when effectively capped, would form an excellent container for commercial deposits of petroleum, having a large vertical drainage area of connected passages through which the gas, oil, and underlying water could flow undisturbed by the crumbling and caving attendant upon the flow of these fluids through unconsolidated sediments. The same conditions as regards structure would obtain if the source of the oil were the organic material in the shales above the limestones. In either case, the resulting funnel-and-anticlinal-ring structure would form ideal zones for the accumulation of oil, and even if the amount of hydrocarbons disseminated through the sedimentaries was small, the accumulation near the intrusions would be comparatively large owing to their barrier-like action and to the large horizontal and vertical drainage areas contributing to such concentration zones.

The asymmetric anticlinal-ring structure may be considered especially favorable for the accumulation of oil, while the central funnels, which might become flooded with oil during the early life of the well, would later show the synclinal tendencies for collecting water, as is the case of Gold Hill in the Panama Canal and in several basalt plugs in Mexico.

Dikes would probably give rise to similar structures rather than to plugs or pipes, but of a more elongated character; of added importance for the accumulation of oil being in this case the effect of dikes in blocking the underground flow of oil and, in many cases, of excluding water from an adjoining producing oil reservoir. Huntley, who opposes this view, states that:[37]

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[35] Bibliography, No. 18, pp. 202–206.
[36] Bibliography, No. 25, p. 727.
[37] Bibliography, No. 21, p. 281.

"Basalt dikes cannot be looked upon as barriers to the movement of oil, except in so far as they accompany fractures which tend to trap it."

Jeffreys, on the other hand, lays great stress upon the importance of dikes in governing the migration of oil and water, and finds that these barriers dam the water so effectively that although one well may have "gone to water" its neighbor separated from it by a dike will continue to yield "dry" oil and be unaffected by the adjacent flooded area.

The basalt itself and the metamorphosed zone of shales, as at Furbero, surrounding the vesicular basalt intruded along the bedding planes, because of its altered and brecciated, hence porous, nature, forms likewise reservoirs favorable for the accumulation of commercial quantities of oil. De Golyer and Norman note that:[38]

". . . the intrusion would have been as effective in any form which it might have assumed, provided that it did not outcrop;"

and further, that the dominant factor in the accumulation of oil in this instance is the greatly varying porosity of the rock, rather than the structure.

Summary

As subjects for further study and discussion, the writers present the following tentative conclusions:

1. In a general way, the intrusions which are exposed at the surface occur as mushrooms, pipes or necks, and as dikes, often in a near-vertical position; while those which do not outcrop take more readily the form of sills or minor laccoliths intruded along bedding planes or other planes of weaknesses.

2. The material making up the intrusions belongs to the basalt group of igneous rocks and consists for the most part of basalts, dolerites, and diabase, vesicular in places, and associated with breccia, volcanic ash, and agglomerate.

3. Although the location of some intrusions apparently bears no relation to existing structure, the greater number occur along fractures or other lines of weaknesses in the sedimentaries, probably caused by either the forces that produced the folding or those which accompanied the intrusions of the larger igneous masses.

4. The effects of the intrusions on the adjacent strata may be grouped under three heads: first, a metamorphic change of the sedimentaries which involved a decrease in volume with a corresponding increase in porosity; second, the formation of a more or less vertical system of connected channels around the intrusions for collecting the hydrocarbons disseminated in the intruded sedimentaries; third, the development of a peculiar funnel-and-anticlinal-ring structure in the sedimentaries as a

[38] Bibliography, No. 20, p. 268.

result of the uplifting and subsequent drag back by the solidifying igneous mass.

5. The funnel-and-anticlinal-ring structure is by no means uncommon in connection with volcanic necks and dikes elsewhere. Geikie observes that this structure may be suspected to be normal to all old volcanic vents.

6. The commercial importance of the technical study of the intrusions and related phenomena lies in the correlation of the character and attitude of the actual oil productive zones with the pre-existing structure, and the form and position of the intrusion. As an illustration of the probable effect of the intrusions on the concentration of oil, one may take the case of a plug or dike intruding beds that originally formed an undisturbed monocline (see Fig. 3). It seems reasonable to conclude that the resulting structure would be of the nature of a terrace on the "upper" side of the monocline and a decided asymmetric anticline on the opposite or "lower" side with gradations around the intrusion from one to the other type of fold. Were these facts ascertained, in a particular pool and knowing that the conditions for larger accumulations of oil were more favorable on the "anticlinal" side of the plug, it is obvious that the drilling programs could be more intelligently planned.

7. Although there is no question of the existence of these conditions in the Mexican fields, the writers do not know how numerous such occurrences are, and whether other factors are of greater importance in the accumulation of oil in the region, than the igneous intrusions here discussed.

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Discussion

E. T. Dumble, Houston, Tex.—This paper is a continuation of one that was published by Mr. Garfias, I believe, in the *Journal of Geology*, 1912, in which he gave the results of his investigations in Mexico, showing that the basalt extrusions frequently did not come up as cones but reached the surface as a small neck and then mushroomed out into a sill, penetrating the beds both in limestones and in shales. He has continued

his investigations to show that as these basalt protrusions came up they forced the strata into a funnel shape at the top instead of throwing them out altogether.

While this may be true concerning these particular beds, there are other places at which the basalt does appear clearly as cones and domes, and there the funnel structure does not exist. We know that the sills are there, because we find them in the wells. In one well that I know of we found a 2-ft. sill of basalt at about 1600 ft. and oil below it at about 2100 ft.; others have gone through similar beds. We also know that there are cones, because north of the Ranuco River one well struck basalt at about 800 ft. and continued in it for several hundred feet—did not pass through it at all. This paper of Mr. Garfias will give us considerable help in that oil field.

W. E. Wrather, Wichita Falls, Tex.—I would like to ask for information as to the temperatures of Mexican oils. I have heard a number of conflicting reports about the high temperatures of the water and oil, and the relationship between them.

The Chairman (I. N. Knapp, Ardmore, Pa.)—No one seems to have information as to temperatures in the Mexican fields; but as to the temperature of the oils at Spindletop, it was about 104°; at Anse-le-Butte, 135°. Referring to the hot water along the Comstock lode in Nevada, I remember the water at the Julia mine was 152°. The hottest water was in the old Belcher mine, and was about 170° F.

J. A. Udden, Austin, Tex.—Possibly you will be interested in the occurrence of volcanics in the Thrall oil field. We have records of nearly all the wells in that field.

The Thrall field is about 20 miles east of the so-called Balcones escarpment, which is a break going from northeast to southwest through Texas, on the east side of which the ground has dropped some 1000 ft. (304 m.). East of the fault there is a general monocline extending clear to the Gulf. The first rock that appears on the surface east of the faulted belt is the Austin chalk, which overlies the Eagle Ford shale. The Austin chalk is 300 or 400 ft. (91 or 121 m.) thick. On top of the Austin chalk is 700 or 800 ft. of the Taylor marl, and above that, 400 or 500 ft. of the Navarro marls, with sands. The divisions between those formations are not very well established. On top of that comes the Tertiary. The series dips to the east, with a descent of 60 to 70 ft. to the mile, although in the Thrall field it may be a little steeper.

The Thrall oil field had an accumulation of oil in what I consider to be a volcanic cone which was built on the Taylor marls at the time that they were being laid down. The low cone seems to have started at a time when the Taylor marl was about 200 ft. thick; then came an eruption. Afterward the entire thickness of the Cretaceous was deposited

on top of that. At first we were much in doubt as to whether the volcanics really were contemporaneous or whether the whole igneous mass might not be an intrusion. I had a number of samples of the rock examined by competent petrographers; they did not find so very many samples of anything decidedly characteristic as, for example, tuffs or lavas. It could be conclusively shown that the whole rock has been hydrated, and that has involved a great change, a brecciation, so that what evidence there was seemed a little doubtful. Professor Baker was nearly convinced that the rock was an intrusion in the shale. I did not believe that it was. Afterward Prof. Baker himself found the best evidence that it is not an intrusion but an extrusion in the Cretaceous sea.

I cannot discuss the whole evidence. We have an analogy only 20 miles from Thrall. A non-conformity at the top of a similar deposit is there shown clearly, and there is no doubt whatever that tuffs and other volcanics, which were somewhat tilted, were there eroded to a flat surface and later buried under Austin chalk. This occurred a little earlier in the history of the district, however, and was not contemporaneous with our Thrall deposit. The fact is that tuffs were seen in the Thrall extrusive and we have also found lava, although most of the rock could better be described as porphyritic in structure.

The evidence that I think proves conclusively that the oil-bearing rock in the Thrall field is an extrusion, is that we have recently found a deposit of volcanic dust at about the same horizon as that of this rock in the Taylor marl, 100 miles away. Evidently there was an extrusion in the Cretaceous sea, the volcanic dust having been disseminated by water.

It is interesting to know that there have been other extrusions and intrusions all along that Balcones escarpment, which runs 200 miles north and south, and then bends westward from San Antonio. The question suggests itself: How old is that fault? I have some data on that subject which I have not yet published; I cannot yet interpret them fully.

Near this escarpment, at Georgetown, 30 miles northwest of Thrall, a well was sunk for deep water, not long ago, and at about 1100 ft. a schist-like shale was found, which is cut by innumerable little veins that run perfectly straight. This is different from the Archaean shales that we find in central Texas, and is evidently too old to be Pennsylvanian. The Cretaceous rests almost immediately on that material, although there may be a little of some Paleozoic material on top of it; I cannot say. The first suggestion that occurred to me when I examined those samples was that it was a local phenomenon, being so close to the Balcones fault. As against this idea, in the first place we do not find much evidence of metamorphism along the fault plane in the lower Cretaceous; and in the

second place I have found the same material in two wells, one at Leon Springs, the other east of San Marcos, the latter being about 50 miles from Georgetown well, and the former nearly 150 miles away. In other words, along the Balcones escarpment we seem to have an axis of an ancient series of rocks of which we do not know the age; they are evidently younger than the Archaean in the central mineral region, and I have no doubt whatever that they are much older than the Pennsylvanian.

We know that a considerable thickness of the Pennsylvanian overlies the more ancient Paleozoics further west, where we find the earlier Paleozoic, the Ordovician, and Cambrian. Along the Balcones escarpment, however, both of these are eroded. Where we find the more ancient rocks the Balcones escarpment would appear to be a line of disturbance which certainly must be very old. If it were recent, we ought to experience an earthquake there once in a while, but so far as I have been able to learn, there has never been any indication of recent movement in that way. It is in connection with this old line of disturbance that we find the igneous extrusions and intrusions in the upper Cretaceous, and therefore we shall not be surprised if sometime we find another oil-field like the Thrall, which lasted such a short time.

W. E. Wrather.—Has Dr. Udden noticed any magnetic disturbance in the vicinity of the Thrall extrusion? Is magnetic material present in sufficient quantity to be indicated by the deflection of the needle on the surface?

J. A. Udden.—That is a question on which I would like to ask expert advice. Magnetic surveys were suggested early in our study of the Thrall field, but the field developed so rapidly that we did not have time to carry out any surveys before so much iron piping was distributed throughout the field as to render our observations of doubtful value. I should like to ask whether it would be worth while to attempt to locate a body of igneous rock containing a good many grains of magnetite and lying at a depth of 600 or 700 ft. below surface.

F. B. Plummer, Tulsa, Okla.—I spent two summers on the Wisconsin Magnetic Survey, and have been two years in Texas, and have become familiar with methods of magnetic surveying. When locating magnetic rocks by the dip needle, a great deal of skill is needed for the interpretation of deflections, because different minerals affect the magnetic needle differently. There is a great deal of pyrite in these sedimentaries near Thrall, especially in the Taylor marls, and when this pyrite is oxidized it will deflect the dip needle to a considerable extent. Nevertheless, after some experience with the needle one can distinguish between the deflection caused by oxidized pyrites and that by a magnetic orebody in the basalt. In the basalts that I have examined, I doubt if there is enough magnetite to deflect the magnetic needle perceptibly, at as great

a depth as the Thrall field. One who has had experience with the dip needle should be able to detect the presence of these basalts within 400 ft. of the surface.

Chairman Knapp.—In regard to magnetic surveys, while I was in Louisiana I had the pleasure of several visits from Prof. G. D. Harris at the time he was State Geologist. He told me that around the salt domes in Louisiana he could find absolutely no deflection of the needle. It was, however, his hope that he might be able to demonstrate that buried anticlines could be located by the use of the magnetic needle. On one visit, he spent some time in attempting to test his magnetic theory by actual survey, but the weather proved so stormy during the time at his disposal that he was unable to get sufficient data to be worth publishing. He said, however, that he had determined to his own satisfaction that there were great magnetic disturbances or irregularities in Terrebonne Parish.

W. R. Crane, State College, Pa.—Referring to the subject of magnetic surveys in oil fields, I have had some experience in such work and am satisfied that the results obtained would not be very satisfactory. Magnetic surveys can be made to advantage in locating deposits of magnetite or possibly oxidized pyrite. In a region where there exists considerable metallic iron, such as iron casing, and refuse iron on the surface, it would have a predominating influence and the results obtained would be vitiated. I hardly think that it is necessary, as has been stated, to make daily observations on declination in magnetic surveys, as the change from day to day is extremely slight. The whole subject of magnetic surveying and mapping is fully described in a paper by H. L. Smyth.[1]

L. L. Hutchison, Tulsa, Okla.—I should like to ask whether you know what Prof. Harris' theory was for locating anticlines by magnetic declination; why he expected magnetic disturbances along the anticlines?

Chairman Knapp.—My understanding is that it was not Prof. Harris' idea to locate iron at all but to locate favorable spots to drill for oil. It has been eight or nine years since I went over this matter with him in Louisiana, so what I may say at this time may not be his exact idea. His expectation was, I think, to map the isogonic topography of a certain area and if this showed any marked irregularity in magnetic lines it might lead to some sort of guide as to proper places to drill for oil. I do not remember exactly why he expected that magnetic disturbances would indicate anticlines. I think the argument was that where igneous intrusions occurred a deflection of the magnetic needle would be found, and that if such rocks pushed up under an anticline of

[1] *Trans.* (1896), **26**, 640–709.

stratified rocks it might produce a line of equal deflections as a guide to the axis of anticlines buried under unconsolidated material.

W. van der Gracht, Tulsa, Okla.—I have had considerable experience in tracing underground features and hidden passes of the Tertiary in the Netherlands. The entire northwestern Europe had to be considered and I tried all the time to find some relation between the magnetic disturbances and the hidden underground features under the Tertiary and Quaternary plains. Underground dislocations are extremely intense under the plains. There are faults there of 4000 ft. vertical displacement, and some of them reach 6000 ft. A curious thing is that in all the west of Europe we find that magnetic meridians show in many places very curious anomalies and twists, but wherever we knew the underground, we could scarcely find any relation except in just one instance. There is a Mezosoic uplift in northwestern France which is known as the axis of the Artois. The exposed beds are all Mezosoic, there is no igneous rock known at any depth, as far as I am aware; probably there may be some buried at enormous depths. Nevertheless, this structure influences in a markedly clear way anomalies of the magnetic lines, whereas other strong hidden features in the structure of the ground, which are now well known as faults, and consequent great differences of levels of the old floor under the Netherlands and Belgium and also in France, do not show in the least on the magnetic maps. In other regions there are great disturbances in the magnetic lines, especially in France, and we cannot see what causes them; there is nothing known in the configuration of the underground to explain them. These disturbances seem to be caused by intrusions of highly magnetic igneous rocks at unknown depths, never reached by any exploration. My conclusion was that practically it was impossible to do anything with the magnetic survey in trying to figure out the hidden structural features of the underground, except in exceptional instances. All this, of course, is quite distinct from explorations for magnetic orebodies, which are lying close to the surface.

E. T. Dumble.—In connection with Dr. Udden's remarks about the Thrall fields, it might be interesting to note that the old movement to which he refers was continued in the Tertiary. The eruption of the volcanic ash, which was afterward converted into fuller's earth and is now in use, was repeated during the Jackson period of the Tertiary, and we find these beds of volcanic ash all through the Jackson sedimentaries. Later still we find the continuations of this orogenic action in the uplift of the salt domes, which seem quite clearly to be of Pliocene age, beginning at about the time of the Austin chalk.

Concentration Practice in Southeast Missouri

Closing discussion of the paper of A. P. WATT, continued from page 419.

A. P. WATT (written discussion*).—Mr. Van Winkle considers that my "ideal curve" is based on the work done by Sergic Bagnara. In this, he is in error, as my "ideal curve" has no reference to the "limit curve" of Mr. Bagnara. Had I based my work on that of Mr. Bagnara, I would have given him credit.

Referring to the power consumption per ton of ore crushed, Mr. Van Winkle states, "While this reduction is probably within the economic limit of their practice, the amount of work done is small and as 67 per cent. of the discharge remains at 48 mesh, it is very far from flotation requirements." This statement appears to me to be irrelevant, as no attempt was made to crush to flotation size, but every endeavor was made to produce as little material through 48 mesh as possible. The desire was to limit the tonnage sent to flotation and not to make the greatest possible tonnage.

Mr. Van Winkle also states, "Regarding the statement that if the ball-mill feed were increased to 500 tons a day, the oversize would be greatly increased, I would call his attention to the fact that there are ball-mills, such as the Marcy, that are fitted with 1/8-in. to 3/32-in. grates giving an 8-mesh product; 800 tons would be a very moderate tonnage for one of these mills and the grates eliminate the 'tramp' or oversize particles he speaks of." Apparently Mr. Van Winkle has not read my article carefully for I did not make the statement as he has quoted me. What I did say is, "If a ball-mill were yielding a normal discharge with 310 tons of feed, the oversize would be greatly increased if the tonnage were increased to 500 tons."

It is evident that with 1/8-in. openings in the grates, nothing larger than a 1/8-in. particle can be discharged, but if the grates had 1/4-in. openings and the mill were delivering 310 tons under normal conditions then if the tonnage were increased to 500 tons a day the discharge would contain oversize. Such a condition does not exist with the Marathon.

I cannot agree with Mr. Van Winkle that the difference in end diameters in the rods in the Marathon mill is of no moment. I do not know what he means by "local conditions," but when every rod in an 18,000-lb.

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* Received Feb. 12, 1918.

charge shows a smaller diameter at the discharge end than at the feed end, such results can hardly be due to "local conditions." He quotes Blickensderfer's statement that at the Burro Mountain Copper Co. the taper in the rods was in the direction opposite to that found in the Marathon in southeast Missouri; is it not possible to assume that this difference in diameter of the rods at Burro Mountain was due to "local conditions?"

It would appear that the Marathon rods should wear more at the discharge end than at the feed end, for the following reason: At the feed end the rods are held apart by large particles of ore and the amount of crushing done is probably relatively small. At the discharge end of the mill a very much greater surface of ore is exposed to the rods, and would naturally result in a greater wear of metal at that end.

Lastly, Mr. Van Winkle says, "I am of the opinion that when rods are worn down to 5⁄8 in., or the danger size, there is no great difference in diameter at the two ends.' As we did not have an opportunity in our work to wear the rods down to this diameter I do not actually know whether Mr. Van Winkle's opinion is correct or not, but I fail to see why 1¼-in. rods should have decidedly different diameters at the two ends, while the smaller rods would show no difference. The conditions that would cause a difference in diameter in the larger rods would seem to apply to the smaller ones also.

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